

The
MAKING,
SHAPING *and*
TREATING *of*
STEEL

USS

UNITED STATES STEEL

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Chapter 1

IRON IN ANTIQUITY

Prehistoric and Ancient Use of Iron—The antiquity of man's use of iron is attested by references to that metal both in fragmentary writings and in inscriptions on monuments, palaces and tombs that survived the collapse of such ancient civilizations as those of Assyria, Babylonia, Egypt, China, India, Greece and Rome. In addition to these written records, archeologists have unearthed actual iron tools, weapons and ornaments used by many of these historic ancient peoples, as well as some implements and jewelry of iron in sites in many parts of the world that were occupied by prehistoric peoples who left no written records. The chemical composition and the properties of the metals in these specimens vary widely. Some closely resemble modern wrought iron; some are more nearly like steel as it is known today. For the sake of simplicity, all of the ancient ferrous metals discussed in this chapter will be referred to as "iron." In later chapters the modern meanings of "iron," "steel," "wrought iron," etc., will be clearly defined and the words used in their proper, more exact sense.

Meteoric Iron—Mere mention of the use of iron in the oral traditions or writings of primitive or prehistoric peoples does not necessarily mean that they knew how to produce iron by extracting it from ore. Actually, there is evidence that most of the iron used in earliest times was not man-made, but was obtained from fragments of meteorites. This belief in the origin of the iron used by very ancient peoples is based on three facts. In the first place, practically all of their names for iron, when translated, mean "stone (or hard substance or metal) from heaven," "star metal" or have similar meanings that suggest that the metal they used came from outside the earth. Secondly, chemical analysis of numerous archeological specimens has established that they contain considerable quantities of nickel which likewise is found in similar quantities (usually 7 to 15 per cent, but as high as 30 per cent) in the iron of meteorites. The third instance supporting this belief is that many primitive peoples of relatively recent times used iron from meteorites to make useful implements. In several historical cases, the main masses of huge meteorites from which the natives had laboriously severed bits of the metal were still in existence in the places where they had fallen and still served as sources of supply.

Telluric (Native) Iron—Gold, silver, copper and some other metals known to the ancients often are found on or near the surface of the ground in a fairly pure metallic condition, in the form of nuggets or rough masses. Being bright in appearance, such native metals would be noticed readily and, as they are capable of being shaped by hammering without heating, they would be put to eventual use by primitive peoples. The softness of gold and silver makes them useless for ordinary tools and weapons, and their ultimate chief use would be for vessels and ornament. The metal copper, however, can be hardened appreciably by hammering it

without previous heating, and the very hammering required to shape a tool from native copper might make it sufficiently hard to be useful for many purposes.

Iron, however, is very rarely found in the native state. One of the few known occurrences of native iron is in Northwestern Greenland; the iron occurs as grains or nodules in basalt (an iron-bearing igneous rock) that erupted through beds of coal. Mention might be made of two very rare natural nickel-iron alloys, given the mineralogical names of awaruite (FeNi_2) and josephinite (Fe_2Ni_3), that have been found in the form of granules and small bean-shaped pebbles. It is improbable, therefore, that primitive man could have found any useful quantity of naturally occurring metallic iron, certainly not enough to account for the widespread distribution of iron artifacts that have been discovered by archeologists.

Man-Made Iron—From the foregoing it may be deduced that iron must have been a rare metal for many centuries and that any specimens of it would be highly prized by the possessor. It was not until man learned how to extract iron from its ores at will that it could have become a common metal.

Archeological evidence seems to indicate that a knowledge of how to obtain copper from its ores existed long before iron was intentionally made by man. Mixtures (alloys) of copper and tin that formed bronze, and of copper and zinc that formed brass, provided the ancients with metals that found widespread usage. In a book such as this, devoted to a discussion of iron and steel, space limitations permit only brief mention of these non-ferrous metals. It should be remembered, however, that for many years after man learned how to extract iron from its ores, the product probably was so relatively soft and unpredictable, and undoubtedly hard to work, that bronze in particular continued to be preferred for many tools and weapons. Eventually, iron supplanted the non-ferrous metals for these purposes when men learned how to master the difficult arts of smelting, forging and hardening iron.

The origin of the methods used by early man for extracting iron from its ores is unknown. The rocky, granular or earthy ores gave little promise or outward indication of their valuable content. Some have suggested that men learned the method accidentally when they built fires by chance on crude hearths built of iron-bearing rock, especially if the fire was in a location where a strong, natural draft caused it to burn fiercely. This is a possibility, since what could happen under such circumstances would meet the conditions now known to be required for extracting iron from its ores. These conditions are that the iron-bearing ore should be heated strongly in contact with hot carbon out of contact with air. Small lumps of ore on a hearth, surrounded completely by hot coals, might meet these requirements. The important iron ores consist mostly of combinations of iron with oxygen. The process just

outlined causes the oxygen of the iron oxide to combine with the hot carbon, leaving metallic iron; in modern language, the iron is said to have been reduced from its oxides, and the process whereby the reduction is effected is called **smelting**. Some of the early smelting processes are described in Chapter 11 of this book.

It may be assumed that such chance production of iron occurred often enough in the experience of one individual or tribe to attract attention and eventually to excite a desire to reproduce the process at will. It should be remembered, however, that the high melting point of iron kept it from ever becoming fluid under such conditions. Copper, on the other hand, could be obtained in the fluid state from the smelting operation, and was quite obviously a desirable product that could be shaped either by casting directly into molds or by hammering a solidified lump. Bronze and brass also could be melted in the furnaces available to the ancient metallurgist. The iron reduced in smelting, however, would collect in a loosely coherent mass or "sponge" of metallic granules that would contain much slag in its pores. There would be nothing in its appearance to indicate the potential value of the crude, sponge-like mass. The key step probably occurred when some early man first discovered that if this unattractive mass was hammered while still hot, slag would be forced out and a lump of reasonably sound metal would result. Repeated heating and hammering would eventually be found to further improve the product, since such operations were necessary to the shaping of the lump into a useful implement.

The manufacture of iron on a relatively large scale could be undertaken once the smelting process was established and its limitations, as outlined above, were understood. Some ancient peoples mastered the principles and practices of smelting iron much earlier than others, and many instances in history point to the rapid rise and eventual ascendancy of the iron-producing nations over their backward neighbors. It is not known definitely whether the knowledge of ironmaking spread from a single point of original discovery or whether it was developed independently in several widely separated localities.

Archeological Evidence—Some of the oldest known iron objects found in various localities include one piece found in the Great Pyramid (built about 2900 B.C.) and another in a grave at Abydos (from about 2600 B.C.), both in Egypt. A cube of iron was found in an 1800 B.C. grave at Knossos in Crete. Tombs at Pylos in the Peloponnesian peninsula of Greece contained iron finger rings dating from around 1550 B.C. What was probably an iron dagger was found at the site of Ur of the Chaldees and is believed to date from 3100 B.C. None of the foregoing are believed to have been fashioned from man-made iron. Tools and weapons of man-made iron were discovered at Gerar, near Gaza in Biblical Palestine, and some of the iron knives found there are believed to go back to 1350 B.C.; remains of iron-working furnaces from about 1200 B.C. were also unearthed at this site. The dates given here are only relative, since archeological research is continually turning up new evidence that necessitates frequent revision of chronology.

References made to iron in very ancient writings from China and India suggest that the metal was used in those areas at least as early as 2000 B.C., but there is nothing to indicate that the iron was man-made. Some authorities ascribe the original discovery of practical iron smelting to peoples in India at a very early date.

On the basis of existing evidence, the deliberate smelting of ore to produce iron seems to have begun to be practiced to an ever-increasing extent over a relatively

wide geographic area in the ancient world between 1350 B.C. and 1100 B.C. After the latter date, the art of iron-making seems to have become generally practiced, at least by the more advanced peoples.

It must not be inferred that the peoples of all areas advanced the art of smelting iron to the same degree over the same period of time; isolated peoples in India, Africa, and Malaya, for example, still employ crude iron-smelting processes not unlike those of other more advanced peoples of 3000 years ago. Neither must it be assumed that, as soon as man-made iron became available, iron at once supplanted stone, bronze, brass, and other materials in tools and weapons. In some localities, tools and weapons of both stone and metals were in simultaneous use for some time until the metals completely took the place of stone. Iron gradually and eventually replaced bronze, brass and the other then-known metals wherever the superior properties of iron made it desirable. The properties of bronze and brass, which could readily be melted and cast into shape, depended largely on their chemical composition. The properties of objects made from iron, however, were more dependent upon the care and skill expended in smelting, forging and hardening, as described below.

Products of Ancient Iron Smelting—According to the usual manner in which early smelting processes probably were carried out, the product would be soft, malleable, wrought iron. If the temperature was high enough and the lump of reduced iron, intentionally or unintentionally, was kept long enough in contact with hot charcoal, away from contact with air, the pasty sponge of iron could be made to absorb carbon from the fuel. Absorption of moderate amounts of carbon (up to, say, one per cent) would cause the metal to become capable of being made very hard by cooling rapidly from a high temperature, in a manner comparable to modern steels. "Soft" wrought iron, containing only very small amounts of carbon, cannot be hardened appreciably by such treatment. Carbon might similarly be absorbed during repeated heating of the metal in charcoal forge fires prior to hammering to make a tool or weapon. Carbon could also be added to iron by variations of the crucible and cementation processes discussed in Chapter 13 of this book.

The metal now called "steel," in its simplest form, is essentially an alloy of pure iron with less than 2 per cent carbon. As mentioned above, one of the important properties of iron alloyed with moderate amounts of carbon is its ability to become very hard by rapidly cooling (**quenching**) it from a high temperature, providing that it contains sufficient carbon. This effect evidently was known at an early date. It also was known that the quenched metal, which was very hard and somewhat brittle, could be reheated to a relatively low temperature after quenching to make it less brittle without too drastically lessening the hardness obtained by quenching. This latter process is now known as **tempering**.

Judgment and skill of the operator were the only means of control of any of the early smelting or hardening methods, since nothing was known of the metallurgical principles which governed them. In smelting, the metal was not predictably uniform from one operation to the next, and a large part of the iron in the ore was lost to the slag. This lack of uniformity in the product of the smelting operation made it impossible to employ other than "rule-of-thumb" methods for carrying out the hardening treatment, and failure to obtain the desired results was probably common. In some areas, the iron ores also contained some other metals beneficial to the properties of the iron with which they became al-

loyed during smelting (manganese, for instance), and the metals produced from such ores were so superior as to become justly celebrated, although the reason for the superiority was not then known.

Early Cast Iron—The more carbon an alloy of iron with carbon contains, up to something over 4 per cent carbon, the lower the melting point of the alloy will be. As mentioned earlier, iron reduced from ore in the smelting process could absorb carbon from the fuel in the hearth or furnace if the spongy metal was kept surrounded by the hot charcoal in a location where it was protected from contact with air. It was even possible for at least parts of the "sponge" to absorb enough carbon (say, 2.5 to 3 per cent) so that the melting point of those parts of the metal was lowered to such a degree that molten high-carbon iron was produced. In most cases, the lumps that formed when the liquid metal solidified were thrown away by the early smelters as worthless, since they were not malleable even when heated. Eventually, it was learned how to produce such molten iron deliberately in special types of furnaces, from which it was poured into molds of the desired shapes to make useful articles. Specimens of cast-iron utensils are extant, notably in China, dating back to at least 200 B.C.

As larger and larger smelting furnaces were used, molten high-carbon iron eventually came to be the chief product of smelting. Cast into blocks called **pigs** (whence comes the name **pig iron**) this iron could be remelted as desired for casting into molds, or could be treated by various processes to make wrought iron and steel. Details of wrought-iron manufacture are given in Chapter 11; some early steelmaking processes are described in Chapter 13.

Chapter 2

SOME FUNDAMENTAL PRINCIPLES OF CHEMISTRY AND PHYSICS

PART 1 — THE COMPOSITION, STRUCTURE AND PROPERTIES OF MATTER

SECTION 1

THE RELATION OF METALLURGY TO PHYSICS AND CHEMISTRY

In principle, steelmaking involves only a relatively few, outwardly simple steps (Figure 2—1). The principal iron ores used in the United States consist of combinations of iron with oxygen (iron oxides), mixed with varying percentages of useless earthy materials. In order to remove the oxygen and useless materials, the iron ore along with coke and limestone are charged into the blast furnace. Hot air blown into the blast furnace burns the coke to generate heat. The hot carbon of the coke, and the carbon-bearing gases generated by its burning, combine with the oxygen of the iron oxide, leaving the iron in metallic form. The iron, in the molten state, collects in the hearth at the bottom of the blast furnace. The limestone combines with the useless part of the ore to form molten slag that floats on top of the pool of molten iron. In this process, the iron becomes contaminated with some of the carbon from the coke used as fuel. The product of the blast furnace, pig iron, may contain somewhat over 4 per cent carbon. To make steel from pig iron, some of the carbon must be removed, because most steels contain considerably less than 1 per cent carbon. The unwanted excess of carbon is removed in the steelmaking processes, principally the Bessemer converter and the open-hearth furnace processes, using controlled amounts of oxygen for its removal. The resulting final product, steel, is first poured into molds to form ingots, which then are formed into useful shapes by rolling or other forming processes. Where very complicated shapes that cannot be rolled or forged are required, the molten steel may be poured into molds that give it the shape in which it is to be used without any further forming; such shapes are called steel castings. However formed, the final products, in a great many

cases, are subjected to heat treatment to develop special properties.

The procedures simply outlined above become complex in practice because of a vast array of complicating circumstances. Actually, the problems in each step of manufacture are very complex. Their solution requires considerable technical knowledge and ability. Some previous knowledge of the fundamental sciences of Chemistry and Physics is essential to a study of these problems. The purpose of this chapter is to present, in as concise and simple a manner as possible, those principles of these two sciences that are directly related to the manufacture of iron and steel.

Physics and Chemistry are related sciences, since both are concerned with the properties of material things (collectively called matter, which may be defined as anything that occupies space). Physics is concerned chiefly with the structure and behavior of matter, and with energy in its various forms, whereas Chemistry is devoted to study of the composition of different kinds of matter and the ways in which they interact with each other. The two sciences overlap to such an extent that, in recent years, two new sciences known as Physical Chemistry and Chemical Physics have arisen on the middle ground.

The science of Metallurgy is based on the two fundamental sciences, Physics and Chemistry, and is commonly subdivided into two major fields; namely, Physical Metallurgy which deals with the structure, fabrication and behavior of metals, and Process Metallurgy (also called Chemical Metallurgy) which deals with the smelting of ores and minerals and with the refining and furnace reactions of molten metals.

SECTION 2

COMPOSITION OF MATTER

The Nature of Matter—Matter is the basic working material in the world, and it is associated with all human activities in some way. The human intellect is aware of the existence of material things (things composed of matter) through the senses of sight and touch. Although it is difficult to accurately define matter, cer-

tain facts about it are plainly evident. For example, it occupies space, it is acted upon by the force of gravity, and it possesses certain properties that are dependent upon its composition and structure. Matter which occupies a limited portion of space is termed a body or mass.

STEEL MAKING

FROM RAW MATERIALS TO FINISHED PRODUCTS

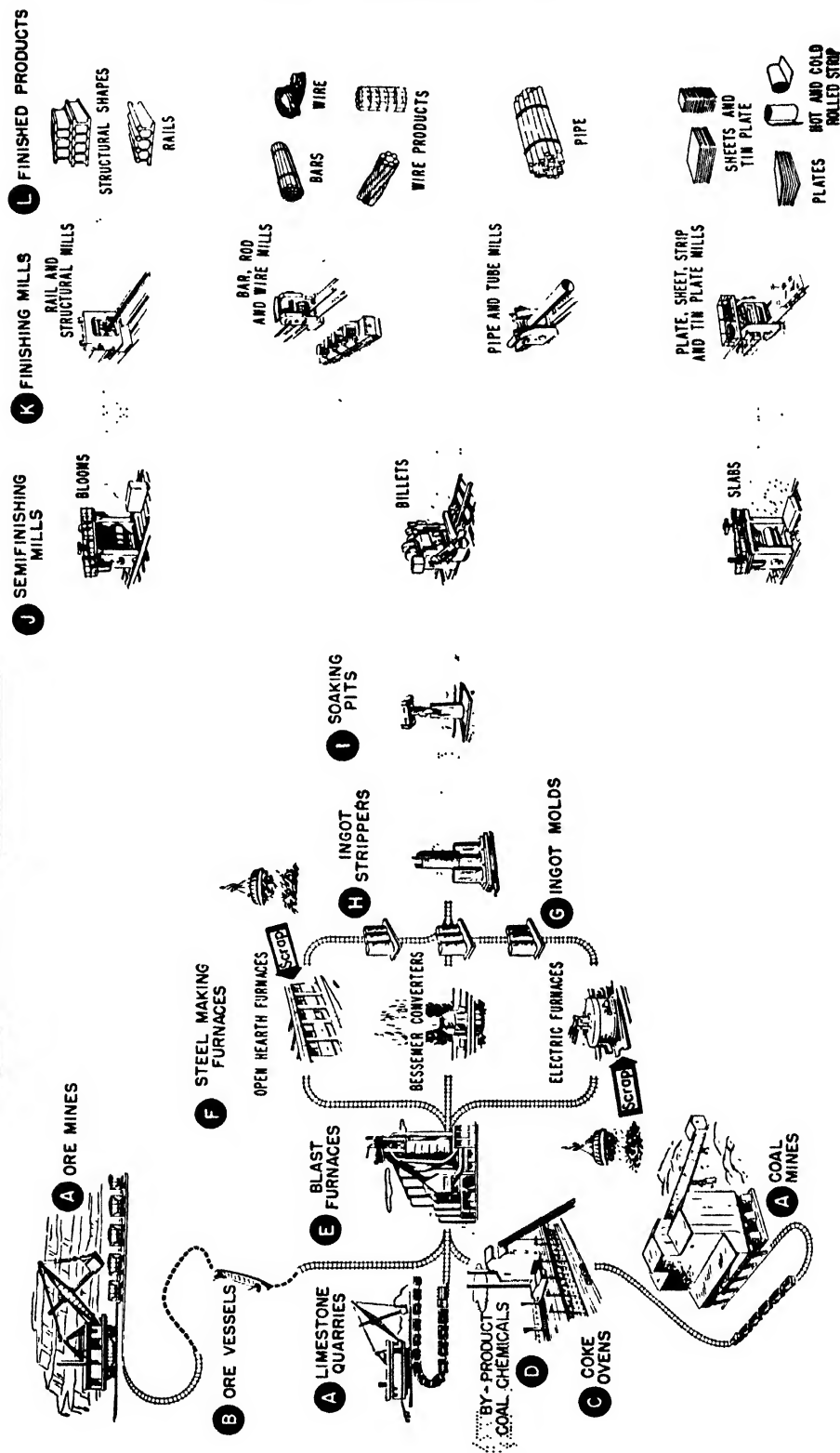


FIG. 2-1. The principal steps in making finished steel products from the primary raw materials (iron ore, coal, limestone).

Mixtures, Compounds and Chemical Elements—Most materials in nature are mixtures; that is, they are composed of a commingling of different kinds of matter. The various kinds of matter in a mixture are called its components or ingredients. For example, iron ore may be a mixture of iron oxides, sand, and clay, and it is possible, by mechanical means, to separate a sample of the ore into these ingredients. It is typical of mixtures that they may contain any proportion of ingredients.

Any material, when sufficiently purified so as to consist of only one kind of matter, is spoken of as a substance. Substances, unlike mixtures, always contain the same ingredient or ingredients, and possess definite and invariable physical and chemical properties.

Most of the many thousands of pure substances known to the chemist can, with greater or less difficulty, be broken down into simpler substances by chemical means; that is, they can be decomposed into simpler substances by:

- (1) The action of heat.
- (2) Electrolysis, or the action of an electric current.
- (3) Chemical reaction with other substances.

However, there is a group of 98 different substances which cannot be decomposed by chemical means. These 98 substances are the chemical elements, usually simply called elements or elementary substances.* All other known substances are made up of various combinations of two or more of these elements; such combinations are called chemical compounds. It has been estimated that about 99 per cent of the solid crust of the earth is made up of a multitude of chemical compounds composed chiefly of only twelve of the chemical elements, approximately as follows:

Element	Per Cent
Oxygen	49.85
Silicon	26.03
Aluminum	7.28
Iron	4.12
Calcium	3.18
Sodium	2.33
Potassium	2.33
Magnesium	2.11
Hydrogen	0.97
Titanium	0.41
Chlorine	0.20
Carbon	0.19
TOTAL	99.00

A chemical compound is a distinct substance formed by a union of two or more elements in definite proportions. Each compound always contains the same elements, united in the same proportions by weight.

Symbols of the Chemical Elements—For convenience and brevity, each element is represented by a symbol. These symbols are composed of the first letter, capitalized, of the English or Latin names of the elements, combined, where necessary as a distinguishing mark, with some succeeding letter. Thus, C = carbon, Ca = calcium, Cd = cadmium, F = fluorine, Fe = ferrum (iron), and so on.

Broad Classification of Chemical Elements—A study of the elements reveals the fact that they consist of two great classes; namely, those that combine with oxygen and hydrogen to form bases, and those that combine with oxygen and hydrogen, or hydrogen alone, to form

acids. The former are sometimes called metals and the latter non-metals, or metalloids. The line of division is not a sharp one. Some elements, called amphoteric elements, form either acids or bases, according to the type of chemical change involved, but the tendency is usually more pronounced in the one direction than in the other. Furthermore, like plants or animals, these two main divisions may be subdivided into families or groups, the members of which possess similar properties. These divisions and groups are discussed in a later section of this chapter.

There are 77 elements classified as metals on the above basis. All of these are solids at ordinary pressures and temperatures, except mercury (Hg), which is liquid. The metals generally conduct heat and electricity well. Many of them are malleable; that is, they can be rolled or hammered into desired shapes without rupturing. The more malleable metals can be rolled or hammered into thin sheets more easily than others. Many metals also are ductile. Ductility permits metals, when pulled, twisted or stretched beyond a certain point, to alter their shape permanently without breaking (unless the stress causing the deformation is too continued and too great, in which case, the metal will deform to the greatest possible extent in an effort to accommodate the load imposed on it, after which it will fracture). The remaining 21 elements classified as non-metals are either gases or brittle solids at room temperature and under atmospheric pressure, with the exception of bromine which is a liquid under these conditions.

Atoms and Molecules—Centuries ago, Greek philosophers evolved the concept that all kinds of material things are made up of tiny particles which they called atoms (Greek: indivisible). They assumed incorrectly that there were as many different kinds of atoms as there were different kinds of natural substances. It was not until the latter half of the Seventeenth Century that the British physicist Boyle defined the term "element" in almost its modern sense and, in conjunction with the work of others, established the concept that all natural substances, however complex, are composed of various combinations of two or more elements, and that there can be only as many kinds of atoms as there are elements. The different atoms can combine to form compound atoms, now generally called molecules (Latin: a small mass). Even atoms of the same element can and do combine to form molecules. Some few elements, such as the inert gases, are called monatomic elements because they are composed of single, uncombined atoms.

The British chemist Dalton, in 1808, propounded a very important hypothesis, now known as the atomic theory, to explain certain phenomena associated with chemical changes. This hypothesis is based upon the following assumptions.

1. The molecules of matter are themselves made up of small particles called atoms.
2. Atoms possess the power of attracting other atoms or otherwise attaching themselves to them.
3. Atoms do not subdivide in ordinary chemical changes.

Based on the foregoing reasoning, an atom may be defined as the smallest part of an element that ever takes part in a chemical reaction. A molecule may be defined as the smallest particle of an element or a compound that has the chemical properties of larger samples of that substance. A molecule may contain one or an unlimited number of atoms of the same or different elements. Two or more simple molecules, all alike, may combine to form what is called an associated molecule; in some substances, this may produce molecules con-

*Since this chapter was written, the discovery of Elements 99, 100 and 101 has been reported.

taining so many simple molecules that the associated molecule becomes a **giant molecule** or, as giant molecules are sometimes called, a **macromolecule**.

Until relatively recently, it was thought that all atoms of the same element had the same mass or weight, the same form, and the same combining power, while atoms of other elements differed in one or more of these respects. As will be learned in the discussion of atomic structure that follows, this supposition is now known not to be precisely true, but it forms the best approach to a practical use of Chemistry.

Atomic and Molecular Weights—As will be learned later in this chapter, Dalton established that, when elements combine with each other, a definite weight of one element **always** combines with a definite weight of the other. For example, 2 pounds of hydrogen will always combine with neither more nor less than 16 pounds of oxygen to form water (H_2O). The definite weights were given the name **combining weights** which were different for each element. The atom is so small that it is useless to hope that its mass or weight will ever be determined by direct individual weighing. However, the weight of one atom of an element must be proportional to the

combining weight of that element. Since the combining mass or weight of hydrogen is the least of all the elements, it is assumed that its atom is the lightest. Therefore, the atomic weight of hydrogen was given a value of "one" by Dalton, and the atomic weights of the other elements became multiples of it. However, in view of the fact that comparatively few compounds are formed by hydrogen with other elements, while oxygen forms the most compounds of any element, it was later decided to make oxygen the standard. Accordingly, the atomic weight of oxygen is made 16, and the atomic weights of other elements are compared with it as a standard, thus making hydrogen 1.008. This system of comparative weights is the basis for the compilation known as the International Table of Atomic Weights. Being relative, it permits calculations in any unit of mass or weight—grams, kilograms, ounces, or pounds. Table 2—I gives the atomic number, atomic weight, melting point and boiling point of the elements.

The **molecular weight** of a compound, for the purposes of this book, may be considered to be the sum of the atomic weights of all of the atoms making up the molecule.

SECTION 3

THE STRUCTURE OF MATTER

Atomic Structure—Each chemical element has definite chemical and physical properties that are different from those of every other element. These properties for any element are determined by the structure peculiar to atoms of that element. Modern physicists have probed deeply into the structure of atoms and, as knowledge increases, it becomes more and more difficult to present a simple description of the probable structure of the atom. Nevertheless, an attempt will be made here to summarize those points that are pertinent to an understanding of how atomic structure affects chemical and physical properties of matter. While there exists a difference of opinion among specialists as to whether the component parts of the atom are discrete particles, electrical charges, or electromagnetic waves, since they exhibit many of the characteristics of each, for the present purposes the components will be considered and referred to as particles for the sake of simplicity of discussion.

Atoms have been stated above to be not subdivided in ordinary chemical and physical changes; in fact, they are nearly indestructible. They are not solid bodies, but possess an open structure that can be penetrated by particles smaller than ordinary atoms if the particles possess sufficient velocity. The structure of the atom may be considered as consisting of: (a) a **nucleus**, surrounded by (b) an **electronic system** at some distance from the nucleus.

The Atomic Nucleus—The nucleus at the center of every atom is dense and nearly impenetrable. As will be shown later, the nucleus is not involved in ordinary chemical or physical changes. While the nucleus of an atom must be very small in size in comparison with the size of the atom as a whole, most of the mass of any atom is concentrated in its nucleus. That portion of the weight of an atom concentrated in its nucleus is called its **nuclear mass**. Two kinds of particles make up the nucleus: **protons** and **neutrons**. Each proton is of unit mass and carries a unit positive electrical charge. Each neutron is also of unit mass, but carries no charge. Unit mass is the mass of a sixteenth part of the most abun-

dant type of oxygen atom (according to physicists) or of an average oxygen atom (according to chemists). The difference is quite small. **Unit positive charge** is equivalent to the positive charge on the nucleus of an atom of hydrogen, equal to 4.802×10^{-10} electrostatic units. Hydrogen has the simplest known atomic structure, having a nucleus consisting of a single proton and an electronic system consisting only of a single electron (a nearly massless particle with unit negative charge, defined more completely below). The charge on the hydrogen nucleus is the smallest that has ever been encountered. The nucleus of every atom carries a positive electrical charge that represents the total of all of the equal unit positive charges on the individual protons that are contained in the nucleus. The total positive charge on the nucleus is independent of the number of neutrons in the nucleus, since the neutrons are not charged. The **atomic number** of an element corresponds to the number of protons in the nucleus and, therefore, to the total number of unit positive charges on the nucleus. The **atomic weight** of an element is determined largely by the weight of the combined number of protons and neutrons in the nucleus (nuclear mass), since each of these types of particles possesses unit mass; the total weight of the electronic system is only a minor part of the weight of an atom.

If a listing of the elements is arranged in the order of increasing atomic numbers, it is observed that the atomic weights do not increase in the same regular way. The atomic numbers and atomic weights of the first 18 elements are given in Table 2—II.

This tabulation clearly illustrates that increases of 1 in atomic number (increases of 1 in the number of protons in the nucleus) do not increase atomic weight by any regular amount.

The Electronic System—Rotating about the nucleus, in orbits at considerable distances from it, are nearly massless particles called **electrons**, each carrying a unit negative charge of electricity. The mass of a proton is somewhat more than 1800 times that of an electron. Besides rotating in orbits around the nucleus, electrons

Table 2—L. The Chemical Elements (See also Table 2—III).

Element	Symbol	Atomic Number	Atomic Weight	Melt. Pt. (°F)	Boil. Pt. (°F)	Element	Symbol	Atomic Number	Atomic Weight	Melt. Pt. (°F)	Boil. Pt. (°F)
Actinium	Ac	89	227	Molybdenum	Mo	42	95.95	4760	8670
Aluminum	Al	13	26.98	1220.4 ± 0.2	3740	Neodymium	Nd	60	144.27	1540 ± 70
Americium	Am	95	(243)**	Neon	Ne	10	20.183	-415.5 ± 0.5	-410.8
Antimony	Sb	51	121.76	1166.9 ± 0.2	2820	Neptunium	Np	93	(237)
Argon	A	18	39.944	-308.9 ± 0.4	-302.4	Nickel	Ni	28	58.69	2651 ± 2	4350
Arsenic	As	33	74.91	1497	1130	Niobium	Nb	41	92.91	4380 ± 30
Astatine	At	85	(211)	Nitrogen	N	7	14.008	-320.4	-320.4
Barium	Ba	56	137.36	1300 ± 35	2980	Osmium	Os	76	190.2	4900 ± 350	9900
Berkelium	Bk	97	(245)	Oxygen	O	8	16.000	-361.8 ± 0.2	-297.4
Beryllium	Be	4	9.013	2340 ± 70	5020	Palladium	Pd	46	106.7	2829 ± 2	7200
Bismuth	Bi	83	209.00	520.3 ± 0.2	2590	Phosphorus (yellow)	P	15	30.975	111.4 ± 0.2	536
Boron	B	5	10.82	4200 ± 550	Platinum	Pt	78	195.23	3224.3 ± 1.8	7970
Bromine	Br	35	79.916	19 ± 0.4	136	Plutonium	Pu	94	(242)
Cadmium	Cd	48	112.41	609.6 ± 0.2	1409	Polonium	Po	84	210.
Calcium	Ca	20	40.08	1560 ± 35	2625	Potassium	K	19	39.100	145 ± 2	1420
Californium	Cf	98	(244)	Praseodymium	Pr	59	140.92	1700 ± 90
Carbon (graphite)	C	6	12.010	6700 ± 180	8730	Promethium	Pm	61	(143)
Cerium	Ce	58	140.13	1100 ± 90	Protactinium	Pa	91	231.
Cesium	Cs	55	132.91	82 ± 4	1275	Radium	Ra	88	226.05	-96	-79.2
Chlorine	Cl	17	35.457	-150 ± 4	-30.5	Radon	Rn	86	222.	5740 ± 110
Chromium	Cr	24	52.01	3430 ± 20	4500	Rhenium	Re	75	186.31	3571 ± 5	8100
Cobalt	Co	27	58.94	2723 ± 2	5250	Rhodium	Rh	45	102.91	102 ± 2	1260
Columbium**						Rubidium	Rb	37	85.48	102 ± 2	1260
Copper	Cu	29	63.54	1981.4 ± 0.2	4700	Ruthenium	Ru	44	101.7	4500 ± 180	8850
Curium	Cm	96	(243)	Samarium	Sa	62	150.43
Dysprosium	Dy	66	162.46	Scandium	Sc	21	44.96	2190	1260
Erbium	Er	68	167.2	Selenium	Se	34	78.96	428 ± 9	4200
Europium	Eu	63	152.0	Silicon	Si	14	28.09	2605 ± 35	4200
Fluorine	F	9	19.00	-370 ± 20	-306.8	Silver	Ag	47	107.880	1760.9	4010
Francium	Fr	87	(223)	Sodium	Na	11	22.997	207.9 ± 0.4	1638
Gadolinium	Gd	64	156.9	85.60 ± 0.04	Strontium	Sr	38	87.63	1420 ± 20	2520
Gallium	Ga	31	69.72	1760 ± 20	3760	Sulphur (yellow)	S	16	32.066	246.2 ± 0.9	832.3
Germanium	Ge	32	72.60	1945.4	5380	Tantalum	Ta	73	180.88	5425 ± 90
Gold	Au	79	197.2	Technetium	Tc	43	(99)
Hafnium	Hf	72	178.6	Tellurium	Te	52	127.61	840 ± 20	2530
Helium	He	2	4.003	-452	Terbium	Tb	65	159.2	621 ± 9
Holmium	Ho	67	164.94	Thallium	Tl	81	204.39	572 ± 5	2660
Hydrogen	H	1	1.0080	-434.6 ± 0.2	-422.9	Thorium	Th	90	232.12	3300 ± 270
Indium	In	49	114.76	313.5 ± 0.2	Thulium	Tm	69	169.4
Iodine	I	53	126.91	237 ± 2	361	Tin	Sn	50	118.70	449.4 ± 0.2	4120
Iridium	Ir	77	183.1	4449 ± 5	9600	Titanium	Ti	22	47.90	3300 ± 180
Iron	Fe	26	55.85	2802 ± 5	4960	Tungsten	W	74	183.92	6170 ± 35	10700
Krypton	Kr	36	83.80	-251 ± 0.9	-242	Uranium	U	92	238.07	2065
Lanthanum	La	57	138.92	1519 ± 9	Vanadium	V	23	50.95	3150 ± 90	6150
Lead	Pb	82	207.21	621.3 ± 0.2	3160	Xenon	Xe	54	131.3	-170	-162.4
Lithium	Li	3	6.940	367 ± 9	2500	Ytterbium	Yb	70	173.04
Lutetium	Lu	71	174.99	Yttrium	Y	39	88.92
Magnesium	Mg	12	24.32	1202 ± 4	2030	Zinc	Zn	30	65.38	737.03	1663
Manganese	Mn	25	54.93	2273 ± 20	3900	Zirconium	Zr	40	91.22	3200 ± 1300
Mercury	Hg	80	200.61	-37.97 ± 0.04	675						

Additional information regarding the elements may be found in Metals Handbook (ASM) 1948 Edition, pages 20 and 21 and in other reference books.

* From 1952 list of the American Chemical Society.

** Figures in () indicate mass of most stable known isotope.

*** See "Niobium" (Nb).

Table 2—II. Comparison of Atomic Numbers and Atomic Weights.

Element	Symbol	Atomic Number	Atomic Weight	Increase in Atomic Weight for Unit Increase in Atomic Number
Hydrogen	H	1	1.008	
Helium	He	2	4.003	2.995
Lithium	Li	3	6.940	2.937
Beryllium	Be	4	9.013	2.073
Boron	B	5	10.82	1.807
Carbon	C	6	12.010	1.190
Nitrogen	N	7	14.008	1.998
Oxygen	O	8	16.000	1.992
Fluorine	F	9	19.000	3.00
Neon	Ne	10	20.183	1.183
Sodium	Na	11	22.997	2.814
Magnesium	Mg	12	24.32	1.323
Aluminum	Al	13	26.97	2.65
Silicon	Si	14	28.06	1.09
Phosphorus	P	15	30.98	2.92
Sulphur	S	16	32.066	1.086
Chlorine	Cl	17	35.457	3.391
Argon	A	18	39.944	4.487

are also considered to spin on their own axes. The unit negative charge on each electron is just sufficient to balance or neutralize a unit positive charge such as that carried by a proton. The number of electrons in any atom of an element, since the atom as a whole normally is electrically neutral, must equal the number of protons in the nucleus, and both are numerically equal to the atomic number of the element. An increase of one in atomic number, then, adds one proton (one unit of positive charge) to the nucleus, and adds one more electron to the electronic system rotating about the nucleus. The electrons do not rotate around the nucleus at random distances from it, but are located in successive groups or shells of electrons (often called energy levels). For convenience, these shells have been designated as the K, L, M, N, etc., groups, shown as follows for the first four groups, with the K group closest to the atomic nucleus:

Group	Maximum Number of Electrons in Group
K	2
L	8
M	18
N	32

It is the electrons in the two outermost electron "shells" of its atoms that largely determine the chemical properties of an element. The innermost electronic "shells" and the nucleus are not affected by ordinary chemical changes. The electrons in the two outermost shells are often referred to as valence electrons.

Periodicity of the Elements—The periodic table shown as Table 2—III* is the modern version of a similar table originated (1869) by the Russian chemist Mendeleev. The elements are arranged in horizontal rows in the order of increasing atomic number, with proper reference to the arrangement of the electronic systems of their atoms.

As the atomic numbers of the elements increase, the atomic weights gradually increase (but not in regular increments, as already described). Another property that is related to the atomic number is the character of

the X-rays emitted by the various elements, the rays becoming more and more penetrating as they are emitted from excited samples of elements of higher and higher atomic number. These two properties that increase as the atomic number increases can be explained on the basis of the gradual increase in the charge and mass of the nucleus.

The periodic table derives its name from the fact that the elements that appear in each vertical column of the table are very similar in chemical properties. In following the horizontal rows, it is found that those properties which depend on the outermost electron groups of the electronic system (chemical properties) are periodically repeated for elements in the same vertical columns, as are some qualitative physical properties. The actinide and lanthanide metals are very similar in their chemical properties because the new electrons added as the atomic number increases are being added in inner electron groups, leaving the two outermost "shells" (that largely determine chemical properties) almost unchanged. It is because the nuclear structure of the atom has negligible effect on chemical properties of an element that the new knowledge of atomic structure has not invalidated the various laws and theories that have long been in use for making practical application of chemistry.

Isotopes—Every element (except hydrogen) is characterized by having the nucleus of the atoms composing it made up a definite number of protons and a number of neutrons that is variable within rather narrow limits. Most atoms of the same element have the same number of neutrons, but some do not and, therefore, they differ somewhat in nuclear mass from other atoms of that element. Atoms of the same element which have different nuclear masses are called isotopes. Since isotopes are the effect of variations in the constitution of the nucleus, due to a larger or smaller number of neutrons being present, the electronic system is not affected to any appreciable degree and it is extremely difficult to chemically separate isotopes of the same element. The separation can be effected, however difficultly, by physical means in some cases; for example, by taking advantage either of the slightly higher boiling point (lower vapor pressure) or slightly lower diffusion rate of the heavier isotope.

Summary on Atomic Structure—In summary, an atom

* Based on a table in "General Chemistry," by H. G. Deming, published by John Wiley & Sons, Inc., New York (Sixth Edition, 1932).

has a nucleus made up of protons and neutrons. The number of protons in the nucleus of an atom of a given element is fixed, and determines the number of electrons in the electronic system surrounding the nucleus. These electrons are present in groups or "shells" at various energy levels. The outer groups of electrons in this system determine the chemical properties of an element. Neutrons in the nucleus contribute to the nuclear mass only, and have no significance in ordinary chemical changes. The number of neutrons is not a fixed quantity for many elements, and there are therefore variations in nuclear mass of these elements that result in isotopes.

The number of neutrons needed to give the required nuclear mass to account for the atomic weight can be calculated. Taking the iron atom as an example: the atomic number of iron is 26, therefore, it has 26 protons of unit mass in its nucleus. Since the atomic weight of iron is 56, 56 minus 26 or 30 neutrons, each of unit mass, would be required in the nucleus to account for the atomic weight of iron. Actually, the atomic weight of iron is somewhat less than 56; 55.86 to be exact. Atomic weights are seldom integral numbers because of what is called the "mass defect," a concept too involved for simple explanation.

The electron "shells" of the iron atom (assuming an exact atomic weight of 56) would contain the following numbers of electrons, since a nuclear positive charge of 26 units would have to be balanced by an equal number of negative charges in the electronic system:

Group	Number of Electrons
K (closest to nucleus)	2
L	8
M	14
N	2
Total	26

By using data in the periodic table, Table 2—III, a similar calculation can be made for the atoms of any element.

Physical, Chemical and Nuclear Changes—Matter is constantly undergoing changes. These changes are of three kinds, namely, one in which the nature and composition of the matter undergoing the change remain the same, called a **physical change**, another in which the nature and composition are affected, called a **chemical change**, and a third in which an element may be converted into one or several other elements, called a **nuclear change**. The freezing of water and the fusion of steel are examples of the first, while the burning of coal is an everyday example of the second; the third type of change is exemplified by the decomposition of unstable elements such as radium and polonium to form new elements, through nuclear changes. Nuclear changes can also be brought about through human intervention. Since 1919, when the first nuclear change was produced artificially, more and more powerful means have been devised for "bombarding" atomic nuclei with swiftly-moving particles to alter the number of electrons, protons and neutrons in atoms of one element to convert it into another element. This is the transmutation process, which was dreamed of by the early alchemists, and which was believed to be impossible by modern scientists until quite recently. As far as the chemist and metallurgist are concerned, however, this great discovery does not detract from the importance and usefulness of the concept of the atom as the fundamental unit of matter, since, in all ordinary chemical and metallurgical reactions and processes, the atom is still the smallest subdivision actively involved. Thus, the chemical laws of combination and reaction between elements are still valid, and the chemical and physical properties of substances are still determined by the kinds, proportions, and arrangements of the atoms which combine to form the larger structural units of matter, such as molecules, fibers, and crystals.

SECTION 4

CRYSTALLOGRAPHY

One of the most important characteristics of metals is that they are crystalline. This means that the atoms of a metal are arranged in a regular, three-dimensional, geometrical pattern. Although the crystallinity of gem stones and some other minerals, such as diamond or quartz, is generally evident from their symmetrical external form and optical properties, it is not obvious that metals are crystalline. There are several reasons for this, namely: (1) Metals usually do not solidify from the molten state as perfect, **idiomorphic**, single crystals with symmetrical external shape; instead, they crystallize as aggregates of small, **allotriomorphic** crystals of irregular shape, tightly bound together at the crystal boundaries. In metallurgy, these small allotriomorphic crystals are termed **grains**; (2) These grains are so small that a microscope with a magnification of several hundred diameters is generally necessary to observe them; (3) Metal grains are opaque, so that reflected-light microscopy (termed **metallography**), on a carefully polished and etched cross-section of an aggregate of metal grains, is required to see the individual grains; and (4) Since the metal grains are opaque, even in thin sections, their optical properties, which are a direct manifestation of crystal structure, cannot be readily measured, as in the case of crystals

of minerals and chemical compounds that can transmit light, each in its own characteristic manner; instead, more penetrating radiation (X-rays or electrons) must be used to measure the crystal characteristics of metals. The study of the characteristics of crystals is known as **crystallography**.

The physical and mechanical properties of metals and other crystalline materials are dependent to a large extent on their particular crystal structure and to imperfections which may exist in that structure. This is the most important reason for studying crystal structure. A second reason for investigating crystal structure both in metallic and non-metallic materials is that no two known chemical substances or elements have exactly the same crystal structure. Even though the geometrical arrangement of atoms may be the same, the distances between atoms will be different. Thus, crystal structure provides a powerful method for distinguishing between various compounds or elements, even in cases where they are chemically identical. For example, gamma iron and alpha iron are both polymorphic forms of the element iron, and therefore have the same chemical constitution; also, calcite and aragonite are polymorphic forms of calcium carbonate (CaCO_3). However, their crystal structures are quite different, so

that they can be identified by this property. In this manner, the determination of crystal structure becomes an important method in qualitative and quantitative analysis.

In order to understand investigations of the crystal structure of metals and other materials, it is necessary to define some of the terms and quantities commonly used in crystallography. The concept encountered most frequently is that of the **unit cell**, which is the basic unit of crystal structure. A unit cell is a small grouping of atoms or molecules, which if repeated indefinitely in three dimensions will result in the development of the crystal. The unit cell, therefore, exhibits all the properties of the large crystal, and it is only necessary to know the structure of the unit cell in order to specify the structure of the whole crystal.

One of the quantities used to specify crystal structure is the **space lattice** or **Bravais lattice**. If imaginary straight lines were drawn between the atoms of a crystal, these lines would form a lattice dividing the space into equal-sized prisms standing side by side with all faces in contact so that no voids would exist. This lattice of lines is called a space lattice and is of fundamental importance in crystallography. The intersections of the lines are called the points of the space lattice. The manner in which the reference lines are drawn is arbitrary, and in many cases it is found convenient to draw the prism so that points are at the centers of the prism faces or in the center of the prism. A space lattice has two important characteristics; the first of these is that, if a straight line is drawn through two adjacent lattice points and continued on at regular, periodic intervals, points identical to the first two will be encountered; the second is that every point in the space lattice has identical surroundings. This second characteristic means that the group of points surrounding a point of the lattice is the same for every lattice point. There are 14 space lattices, shown in Figure 2-2, and it can be proven mathematically that no others can exist, that are not equivalent to all or part of one of these fourteen.

It can be seen that the points of each of the space lattices can best be identified by reference to a set of three coordinate axes. These axes may or may not be of equal length, and may or may not be at equal angles to

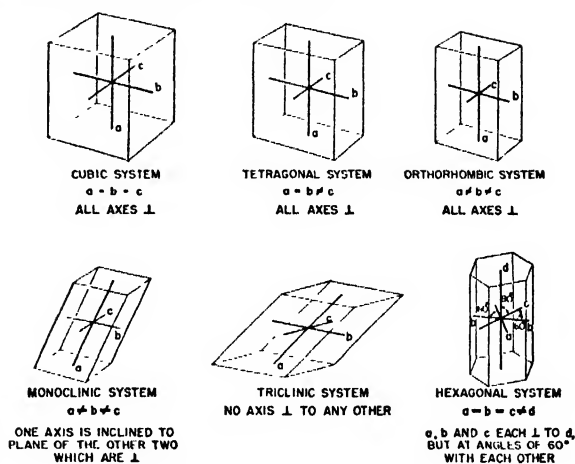


FIG. 2-3. The six symmetry types of crystals.

each other. There are six such sets of axes commonly used. These axes define the six crystal systems shown in Figure 2-3. Frequently, seven crystal systems are defined, by adding what is known as the rhombohedral system, in which the lengths of the axes are all equal and the angles between the axes are equal, but different from 90° . However, the rhombohedral axes can always be converted to hexagonal axes, making their inclusion as a separate crystal system unnecessary. The degree of symmetry of a crystal might be loosely defined as the number of ways in which the unit cell can be rotated, or atoms can be transposed, across some axis or central point, without altering the crystalline arrangement. On this basis, it is interesting to note that the six crystal systems decrease in symmetry in going from cubic to triclinic systems, with the hexagonal system occupying a position between the cubic and tetragonal systems.

Space lattices and crystal systems lead to only a partial description of the crystal structure of a metal or other crystalline solid. If the complete structure is to be specified, it is necessary to consider also the **symmetry classes** and ultimately to determine the proper **space group**. There are 230 possible space groups; however, there are a very large number of crystal structures, since the space group defines only the geometrical arrangement of the atoms and not the distances between atoms. When both the space group and the interatomic distances are known, the crystal structure is completely determined.

Since the atoms in a crystal are arranged in a regular fashion, it is evident that planes can be defined which will contain many atoms. The simplest such planes are those containing two axes of the coordinate system used to define the crystal system. Other planes will, of course, be more complex; their existence, however, should be obvious. These planes are of the utmost importance in discussing properties of a crystal such as its orientation, twinning habit, and slip system. Therefore, some system of nomenclature is necessary to distinguish these planes and to define their orientation, but not their position. The system of nomenclature universally used is that of **Miller indices**. As stated above, it is convenient to define three coordinate axes to describe the crystal system. Beginning at the intersection of these axes and proceeding along any one of them, identical points will be found to occur at periodic intervals, with one of these intervals in each direction defining the unit cell. To

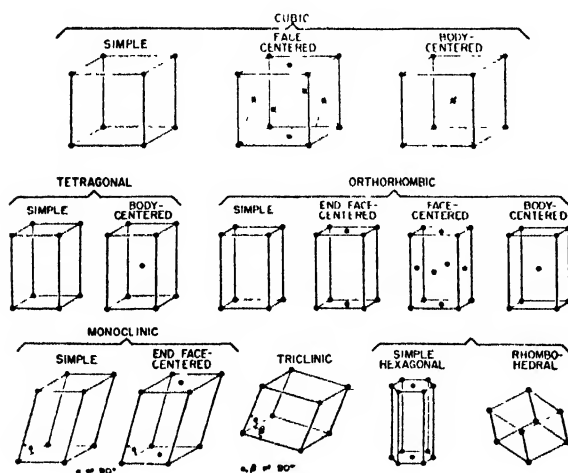


FIG. 2-2. The fourteen general patterns of unit cells. All types of crystal space lattices can be formed from these unit cells.

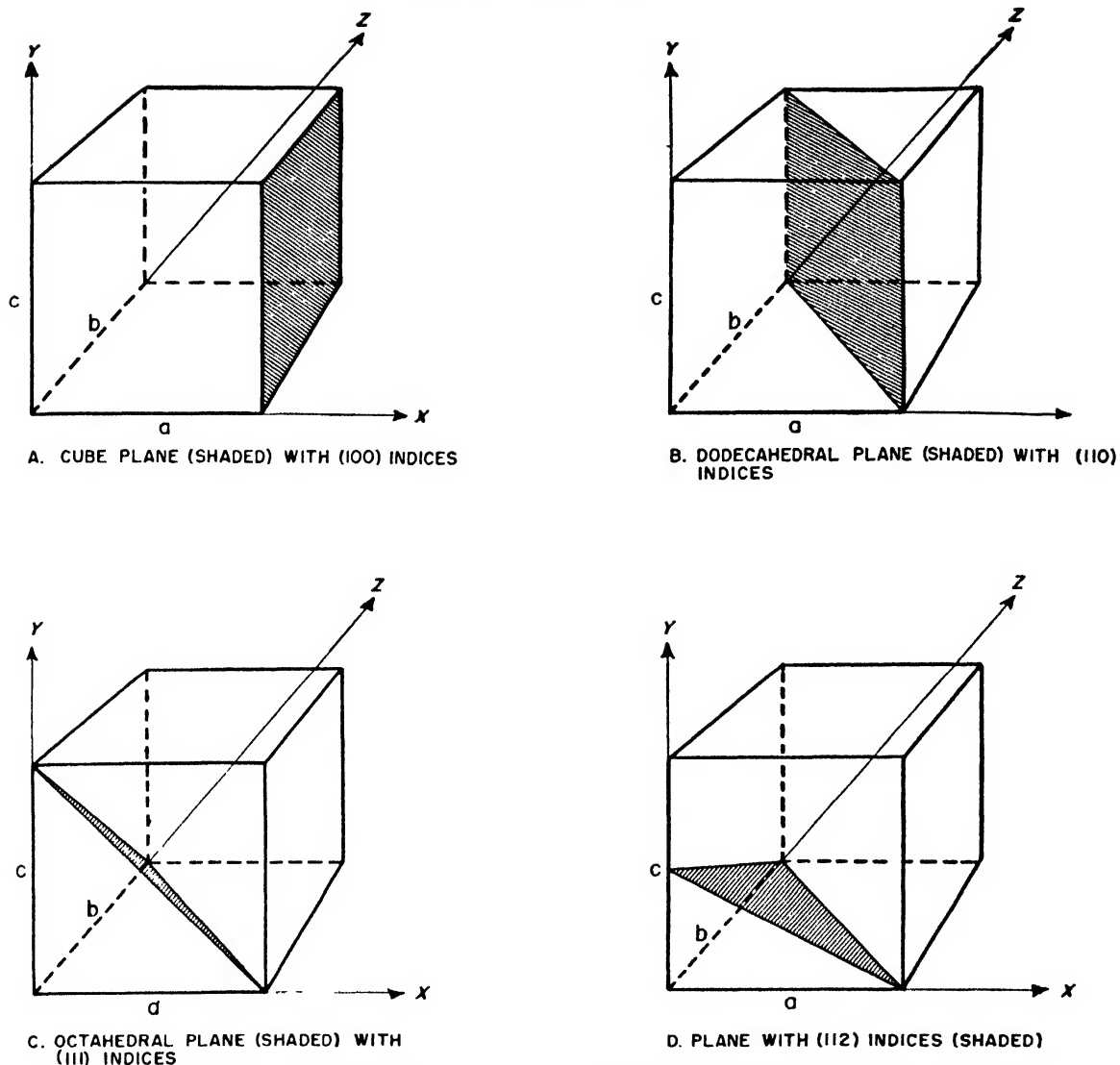


FIG. 2-4. Planes with various indices in cubic crystal.

determine the Miller indices of a plane, it is necessary first to find its intercepts on the three coordinate axes X , Y , and Z in terms of multiples or fractions of the unit cell edge a , b , and c . The values of a , b , and c are known as the **lattice parameters**. If a plane is parallel to one or more axes, the intercept is taken as infinity. The reciprocals of these three numbers are taken, and the reciprocals are reduced to the three smallest integers having the same ratio. These integers, written in parentheses are the Miller indices of the plane. For purposes of general discussion, the letters "h," "k" and "l" are used to represent these integers, thus: (h,k,l) . In an actual example, where a plane intersects the X -axis at three units, the Y -axis at one unit, and the Z -axis at infinity, the intercepts are 3, 1, ∞ , and the reciprocals of the intercepts are $\frac{1}{3}$, 1, 0; when multiplied by 3, these give Miller indices of (1, 3, 0), corresponding to (h,k,l) of the general case. As a result of the reduction to the smallest integers, parallel planes will have the same indices. Figure 2-4 shows some of the important crystal planes with their Miller indices. It should be noted that the coordinate axes extend in both the positive and

negative directions from their intersection, so that negative indices are also possible.

Direction indices are also frequently used in crystallography to specify the direction or orientation of a line in the crystal. If a point is moved from the intersection of the coordinate axes (the **origin**) to some other position in the crystal by means of translations parallel to the coordinate axes, the direction indices of the line joining the origin and the final position will be numbers u , v and w , where u is the number of units of a that the point moved parallel to the X -axis, v is the number of b units that the point moved parallel to the Y -axis, and w is the number of c units that the point moved parallel to the Z -axis. The values of u , v , and w are reduced to the simplest integers, and are written in square brackets $[u, v, w]$ to indicate that they are direction indices. For example, in Figure 2-4, the face diagonal of the base of the parallelepiped will have the direction $[110]$, the X -axis will have the direction $[100]$ and the body diagonal $[111]$.

These definitions of the important terms and concepts will enable us to discuss the crystal structures of the

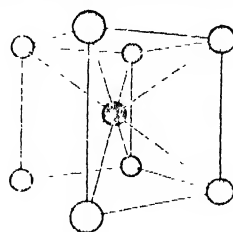
various metallic elements. The term **crystal structure** will be used in the following discussion to cover all of the descriptive information, such as the crystal system, space lattice, symmetry class, space group and distances between atoms (**lattice parameters**), necessary to completely define the crystal structure. Fortunately, metals crystallize in simple structures with a high degree of symmetry; in fact, nearly all metals belong to either the cubic, hexagonal or tetragonal crystal systems.

The simplest crystal structure found in nature is the **body-centered cubic structure**. The atomic arrangement is shown in Figure 2—5a; it is evident that atoms are located at the corners of a cube, and one in the center of the cube. Since the corner atoms are shared by adjacent unit cells in a crystal where this atomic arrangement is repeated indefinitely, only $\frac{1}{8}$ of each of the atoms at the corners is contained in the unit cell and the unit cell is said to contain two atoms. This body-centered cubic arrangement is the structure of metallic iron (α iron) at temperatures below about 1650° F. In iron, the unit cell is 2.861 Angstroms long on each edge; the three lattice parameters are of equal length in the cubic system, that is, $a=b=c$. (One Angstrom unit equals 10^{-8} centimeters.) Other elements having this same crystal structure, with different lattice parameter, include chromium, columbium, potassium, tungsten, and molybdenum.

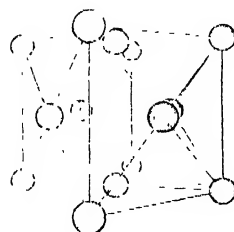
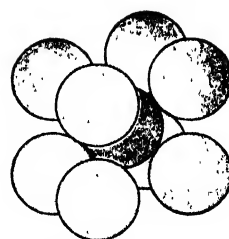
Another simple crystal structure encountered in metals is the **face-centered cubic arrangement** shown in Figure 2—5b. Atoms in this structure are located at the corners of the cubic unit cell and at the centers of the cube faces. Due to the sharing of atoms by adjacent cells, as mentioned above, the unit cell contains four atoms, since $\frac{1}{8}$ of each of the corner atoms and $\frac{1}{2}$ of each of the 6 atoms in the cube faces are in the cell. Iron has this structure (γ iron), with a lattice parameter of 3.63 Angstroms, from a temperature of 1650° F to near the melting point. From about 2600° F to the melting point at 2800° F, iron reverts to the body-centered cubic form (δ iron). Other metals with the face-centered cubic structure include silver, gold, aluminum, copper, nickel, and lead. The face-centered cubic structure is sometimes referred to as the **cubic close-packed structure** because the atoms, if considered as large solid spheres, are arranged so that the total void volume between the spheres is as small as possible. On the (111) planes, the atoms are in intimate contact, and are packed together in the closest possible manner.

A second close-packed structure commonly found in metals is the **hexagonal close-packed arrangement** shown in Figure 2—5c. Crystals having this structure are built up of layers in which atoms are arranged at the six corners and at the center of a regular hexagon. Alternate layers are shifted to permit the atoms of the second layer to fit into the depressions between atoms of the first layer, resulting in the closest possible packing of atoms. The simplest possible unit cell for this structure is a rhombohedron containing two atoms; however, for convenience, the unit cell is usually chosen as the hexagonal prism shown. This unit cell contains three of the rhombohedral cells and is described therefore as being **triply primitive**. Among elements having this structure are zinc, cobalt, beryllium, magnesium, cadmium, titanium and most of the rare earths.

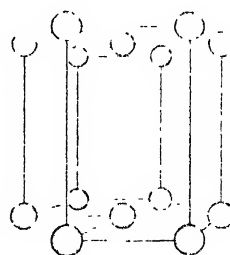
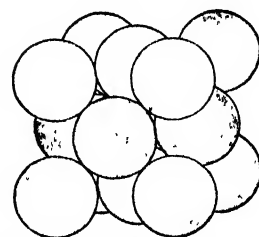
The crystal structure of white tin is not among the three basic types described above; however, because of the importance of tin in the steel industry, its crystal structure will be described. As shown in Figure 2—6 the unit cell is tetragonal with the a and b axes of equal length, and about twice as long as the c axis. In addition to the atoms at the corners, there is one at the center



(a) BODY-CENTERED CUBIC STRUCTURE



(b) FACE-CENTERED CUBIC STRUCTURE



SIMPLE HEXAGONAL STRUCTURE

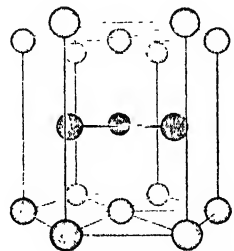
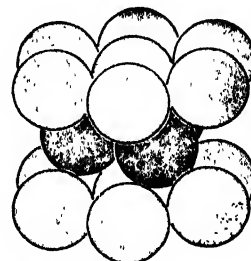
CLOSE-PACKED HEXAGONAL STRUCTURE
(c) TWO FORMS OF HEXAGONAL STRUCTURE

Fig. 2—5. Comparative diagrams showing schematically the arrangement of atoms in the body-centered cubic structure, the face-centered cubic structure, and two forms of the hexagonal structure.

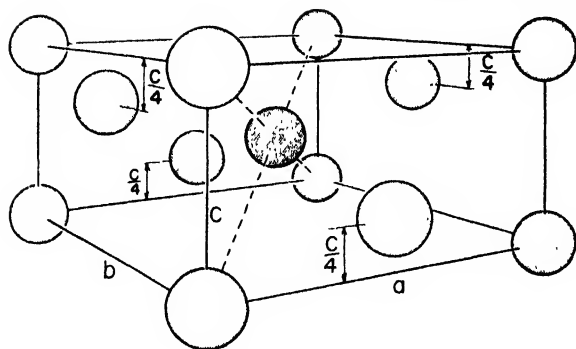


Fig. 2-6. Crystal structure of white tin.

of the cell and one in each of the side faces; however, the atoms in the side faces are not in the center of these faces.

These relatively simple crystal structures are sufficient to describe nearly all of the solid elements and many compounds. However, more complex unit cells are required to describe the structure of other inorganic, metal-organic, and organic compounds. Oxides, sulphates and silicates of metals, for example, are frequently in lower symmetry systems such as the orthorhombic, monoclinic, and triclinic.

Lattice Imperfections—The foregoing discussion has been in terms of ideally perfect crystals where atoms are located only at proper crystal sites and all such sites

are filled with the proper kind of atoms. In real crystals, particularly metals, this is not usually the case; rather, the lattice has numerous flaws or imperfections. The most common imperfections are **interstitial** or **substitutional** atoms of an impurity or alloying element. By interstitial atoms are meant atoms which fit into the holes or interstices between the atoms of the normal structure. Substitutional atoms are those which replace normal atoms in lattice positions. In either case, the impurity or alloying atoms cause a distortion of the lattice, which may markedly affect the physical and mechanical properties; the changes in properties may be either beneficial or deleterious, depending on the property and the application of the material. The second most common imperfections are **dislocations**, which occur when, in two adjacent rows of atoms in the crystal, one row contains one more atom (or one less) than the neighboring row. It is easy to see that this situation will also produce a distortion of the lattice. As a result of thermal fluctuations, dislocations move through the crystal lattice until they are stopped by another dislocation or some other stress barrier. It is these concentrations of dislocations that are responsible for many of the properties of metals such as **strain hardening**, **slip**, and **internal friction**. In fact, the effect of the imperfections in metals is so important that the study of imperfections has developed into a field nearly as large as the study of crystal structure. However, this field is highly theoretical at present, and any detailed discussion of the effect of imperfections is beyond the scope of this section.

SECTION 5

FUNDAMENTAL CHARACTERISTICS OF MATTER

Length, Mass, and Time—In order to describe the fundamental characteristics of matter, and natural phenomena in general, on a reasonably scientific basis, scientists and engineers everywhere must agree on a common terminology in which words and terms are given precise and limited meanings. Furthermore, it is essential to have a system of standards for accurate quantitative measurements, in which the units are defined as exactly and clearly as possible.

Possibly the most fundamental of physical concepts are length, mass, and time, since combinations of these can be used to express all other physical quantities. Scientific workers generally use the **metric system of units** to describe mass and length. The fact that this system is naturally adapted to the expression of measurements in terms of decimals is of great convenience. In this system, the meter (100 centimeters) is the unit of length and the kilogram (1000 grams) is the unit of mass. In nearly all countries, except England and the United States, the metric system is used for everyday as well as scientific purposes. The **English system** is in common use in this country, but legally the inch and pound are now defined in terms of the meter and kilogram.

The unit of length, the **meter**, was originally intended to be one 10-millionth of the distance from the equator to a pole of the earth, although more accurate measurements made since that time show that this is not strictly true. The standard of length now in use is the **International Prototype Meter**, which is the distance, at 0° C (32° F), between two fine parallel lines on a bar carefully prepared and preserved at the International Bureau of Weights and Measures at Sevres, France. An

accurate duplicate standard is maintained at the National Bureau of Standards in Washington, D. C. In the United States, according to law,

1 inch = exactly 2.54 centimeters or 0.0254 meters
1 foot = 0.3048 meters
1 meter = approximately 39.37 inches

By the mass of a body, is meant the quantity of matter it contains, which obviously will be the same anywhere in the universe. The arbitrary unit of mass is the **International Prototype Kilogram**, a block of platinum-iridium alloy carefully preserved at Sevres, France, which was intended to be exactly equal to the mass of 1,000 cubic centimeters of water at its temperature of maximum density (39.2° F or 4° C), but actually differs from this by a very small amount. A duplicate standard is kept at the Bureau of Standards.

It is essential to distinguish between the mass of a body and its weight. By the **weight** of a body is meant the force with which it is attracted toward the earth. The concept of force is intuitive and is frequently associated with the sense of muscular effort in pushing or lifting various objects. From Newton's law of gravitation, this force of attraction of the earth for a body (its weight) is given in the metric system by:

$$W = gM$$

where "M" is the mass in kilograms, "W" is the weight in Newtons, and "g" is the gravitational attraction at the object's position, and its numerical value on the earth's surface is approximately 9.80 meters per second per second. In the English system, the unit of mass has no special name, the weight is expressed in pounds, and "g" is

approximately equal to 32 feet per second per second. A typical equal-arm balance (no springs) compares the mass of an object with a standardized set of masses and will give accurate mass values anywhere. Of course this also measures the relative weights, but in order to obtain the actual or absolute weight of the object, the value of "g" at the place of measurement must also be known, since the value of "g" decreases as the distance of the object from the center of the earth increases. However, since "g" varies only slightly over the earth's surface, the weight of the object can be estimated for most practical purposes by using such a balance, even though "g" is not actually measured. If a simple spring balance could be accurately calibrated at one location, it would give accurate weight values everywhere else, since it measures directly the attractive force of the earth on the object.

All measurements of time are based on the time required for the earth to make one rotation on its axis. Astronomers use *star time* in which one rotation with respect to the fixed stars is 24 hours. For civil and most scientific purposes, *mean solar time* is used, in which 24 hours, or 1,440 minutes, or 86,400 seconds is the average time for one rotation of the earth on its axis with respect to the sun. The unit of time is the second, and this is the same in both metric and English systems.

In dealing with formulas and equations, it is essential not to mix different units of measurement; that is, English units should not be mixed with metric units, and the denomination of the units should be kept consistent (gram-centimeter-second or kilogram-meter-second).

States of Matter—From a casual examination of the surroundings, it is evident that matter may exist in three fundamental states: gaseous, liquid, and solid. A substance may exist in all of the three states; for example, water may exist as a gas (steam), as a liquid, and as a solid (ice). A gas has no definite size or shape, but merely a definite mass; it is composed of molecules or single atoms, which are relatively far apart and free to move because the binding or attractive forces between them are very weak. A liquid has definite mass and volume, but not form; it is composed of molecules or single atoms, which are closely packed together but still are mobile because the attractive forces between them are weak. A solid has definite mass, volume, and shape; it is composed of atoms which are closely situated, tightly bound together, and arranged in a systematic pattern which is known as a crystal structure. Liquids and gases are called fluids on account of their ability to flow. In many cases, they obey the same physical laws. They are distinguished from each other by their relative compressibility: liquids are but slightly compressible, while gases are highly compressible. The volume of a gas can be decreased by the application of pressure; that is, the volume of an ideal gas varies inversely as the applied pressure (Boyle's Law).

Changes of State—It has been stated previously that certain substances can exist in all three physical states; thus, water may exist as steam, liquid, or ice, depending on the pressure and temperature. The state of a substance depends only on the pressure and temperature to which it is subjected. Under ordinary atmospheric temperature and pressure, most substances exist only in a single state; for example, steel is solid at room temperature, and carbon dioxide is ordinarily a gas. However, by applying heat to steel it may be converted to the liquid state (molten steel), and by applying pressure and withdrawing heat, gaseous carbon dioxide may be converted to the solid state (dry ice). The usual sequence of changes of state is: solid \rightleftharpoons liquid \rightleftharpoons gas; how-

ever, some solid substances may be changed directly to the gaseous state, and vice versa, without passing through the liquid state. Changes of state are usually reversible, that is, a solid substance that can be melted by applying heat, can be solidified by withdrawing heat or cooling.

The various changes from one state to another have certain names, thus:

Solid to liquid = melting or fusion

Liquid to solid = freezing or solidification

Liquid to gas = evaporation, vaporization, or boiling

Gas to liquid = condensation

Solid to gas = sublimation

Gas to solid = condensation

Under atmospheric pressure, the temperatures at which these changes occur are characteristic of a particular substance; thus, the melting point of ice (or the freezing point of water) is zero degrees Centigrade, while the boiling point of water (or the condensation point of steam) is one hundred degrees Centigrade. Associated with each change of state there is a transfer of a certain specific amount of energy between each substance and its surroundings. Thus, when a gram of water at 100° C (212° F) evaporates into steam it absorbs 540 calories * of heat energy, and when a gram of steam at 100° C condenses to water, it releases this same amount of heat energy. Also, a gram of water at zero degrees Centigrade gives off 80 calories of heat energy when it freezes into ice, and when a gram of ice melts into water it absorbs the same amount of heat energy. The heat energy involved in the process of melting or freezing is termed the *heat of fusion*; this amount of heat must be added merely to change the material from the solid state to the liquid state, and the temperature of the material undergoing melting remains constant until sufficient heat has been absorbed to effect the change of state. Heat of fusion is usually expressed in "calories per gram," "kilogram calories per kilogram," or "Btu * per pound." The heat absorbed or given off on evaporation or condensation is termed the *heat of vaporization*. Expressed in the same units as those for heat of fusion, heat of vaporization absorbed by a liquid substance represents the quantity of heat required to change the liquid to the vapor state at the boiling point without increasing the temperature of either the liquid or the vapor.

Vapor Pressure—The tendency of a liquid to evaporate is measured in terms of its vapor pressure; that is, when the molecules of the liquid exert great pressure against the surrounding atmosphere, and have a great tendency to jump from the liquid surface into the air (evaporate), the vapor pressure is large and the boiling point is relatively low. The vapor pressure of a liquid, then, is the pressure exerted by the liquid at a given temperature under the conditions that give liquid-vapor equilibrium. For example, if a liquid is boiling in an open container, then the vapor pressure of the liquid at the boiling temperature must be one atmosphere. At a temperature less than the boiling point, the liquid exerts a vapor pressure less than atmospheric pressure. Solids usually exert low vapor pressures. The vapor pressure for any material increases rapidly with the temperature. The relationship between vapor pressure and temperature for most materials can be found in the literature.

Electrical Conductivity—The use of matter to conduct electricity is of almost inestimable value. Metals are

* "Calorie" and "Btu" are defined under "Measurement of Heat" in Section 3 of Part 3 of this chapter.

notably good conductors of electric current, whereas other substances, such as glass and quartz are non-conductors or insulators. The electrical conductivity of a substance is determined by measuring the amount of current that will pass through a given volume of the substance in a certain time when a standard driving force or difference of electrical potential is applied to opposite ends of the piece. The electrical conductivity of the various metals differs considerably; thus, the conductivity of copper is about eight times greater than that of iron.

Density—In handling pieces of different kinds of matter, such as wood and steel, it is at once evident that pieces of about the same size (volume) weigh differently, that is, have different mass. In order to express this difference more definitely, the concept of density is useful. The density of a material is its mass per unit volume; expressed mathematically,

$$D = \frac{M}{V}$$

where "M" is the mass measured in grams, "V" is the volume expressed in cubic centimeters, and "D" is the density expressed in grams per cubic centimeter. In English units "M" may be measured in pounds and "V" in cubic feet; the density then is expressed in pounds per cubic foot. This idea of mass or quantity of matter per unit volume is of course applicable to all three states of matter, solid, liquid, and gaseous. Since the volume of a given mass of material depends on the temperature and pressure acting on it, particularly in the gaseous state, it is necessary to specify these two conditions in giving density values. For gases, a standard set of conditions is used, namely, 0° C (32° F) and the pressure exerted by a column of mercury 76.0 centimeters (29.92 in.) high; these conditions are called normal temperature and pressure (abbreviated NTP).

Specific Gravity—It is oftentimes convenient to express the mass (or weight) of a material in terms of the ratio of its weight to that of some standard substance. This ratio between the weight of a body and the weight of an equal volume of a standard substance is termed specific gravity. For liquids and solids, the standard substance is water; for gases, it is hydrogen or air. Since the volumes of both the body and the standard substance vary with temperature and pressure, it is necessary to specify these conditions for both substances. For solids and liquids, only the temperature is usually specified, since the pressure is assumed to be normal. For gases, the conditions usually are normal temperature and pressure. It is necessary to state which standard is used, since air is much heavier than hydrogen.

Elasticity—When a thin strip of steel is bent moderately and springs back to its initial shape when released, it exhibits the property of solid matter called elasticity. Virtually every solid object is elastic, that is, when bent, compressed, twisted, or otherwise deformed, it develops internal force which tends to restore it to its initial form; the deformation is termed the strain and the force developed is called the stress. Different substances develop different stresses for a given strain, that is, they exhibit different ratios of stress to strain (elastic moduli) which are more or less characteristic of the substance. If the deformation of an object is carried to such an extent that it will not resume its initial form when the applied force is removed, the elastic limit of the material is said to have been exceeded. Within the elastic limit, the stress is proportional to the strain (Hooke's Law). Elasticity is very important in metal-

lurgy and engineering and will be discussed more thoroughly in Chapter 49 of this book, devoted to mechanical testing.

Plasticity—The capacity of a solid body to be strained beyond its elastic limit, and thus to suffer permanent deformation, or change of shape, without fracture, is termed plasticity; it is closely related to other important mechanical properties of solid matter, such as ductility and malleability. Substances vary widely in the amount of permanent deformation that they can withstand before fracture; however, specific measures of this property depend not only on the nature of the substance but also on the temperature, speed of deformation, and the geometry of the stress system.

Tensile Strength—When a substance is stretched progressively, a point is ultimately reached where the stress developed overcomes the cohesive or binding forces tending to hold the material together, and the substance pulls apart; in the case of metals, the relationship between the apparent and true force required to fracture the substance is discussed in Chapter 49, and the exact meaning of tensile strength is defined. Tensile strength is more or less characteristic of a substance, but it depends to an important degree on the condition and plasticity of the substance, the temperature, speed of stretching, and shape of the body.

Hardness—The ability of a substance to resist penetration or, in the case of some substances, to withstand wear or abrasion is termed hardness. This is a rather complex association of two or more simpler properties of matter and is closely related to other mechanical properties, such as tensile strength.

As in the case of elasticity, plasticity, tensile strength and hardness will be discussed more fully in Chapter 49 of this book, dealing with mechanical testing.

Viscosity—The resistance of liquids, semi-solids, and gases to movement or flow is termed viscosity, and it is due to internal friction among the molecules constituting the substance. Viscosity varies widely, depending on the substance; thus, oils and sirups are very viscous, whereas water and alcohol are not so viscous. Gases are only about one-thousandth as viscous as typical liquids, such as water. Viscosity is affected to a marked extent by temperature; in general, the higher the temperature, the lower the viscosity of a fluid. Viscosity is usually measured by the time it takes a given amount of the fluid to pass through a long tube of standard diameter, when there is a standard difference in pressure between the two ends of the tube.

Semi-solids, such as tar, may be thought of as extremely viscous liquids. True solids also exhibit a property very much like viscosity in fluids, thus, the vibrations of a tuning fork die out faster than can be accounted for by the rate at which its energy is given to the surrounding air. In the case of solids, this property is known as internal friction or damping capacity.

Specific Heat and Heat Capacity—The specific heat of a substance is the quantity of heat required to raise the temperature of a unit mass of the substance one degree. The value of specific heat changes with temperature, so that an average value must be used that corresponds to the temperature range involved. The heat capacity of a body of a substance between two temperatures is the quantity of heat required to be added to or extracted from the body, respectively, to raise or lower its temperature the number of degrees between the higher and lower temperatures.

Thermal Expansion—It is a matter of common observation that most substances expand when heated and contract when cooled; a property of matter known as thermal expansion. Over moderate ranges of tempera-

ture, the change in length of a piece of material is proportional to the temperature change. It is also apparent that the change in length is proportional to the initial length; that is, the longer the piece, the greater the change in length. The change in length per unit of length, for a given change in temperature, is named the **thermal coefficient of linear expansion**. This quantity is more or less characteristic of a substance and differs greatly for different substances.

Thermal Conductivity—The ability of matter to conduct or transmit heat is referred to as thermal conduc-

tivity. All solids, liquids, and gases conduct heat, but enormous differences exist in the rates at which heat is transferred through them under similar circumstances. Thermal conductivity is measured by the amount of heat conducted by a unit volume of a substance, in a certain time, for a given driving force or difference in temperature between the opposite surfaces of the piece. In general, metals conduct heat much faster than other substances, such as wood, paper, cloth, or air. The former are called **thermal conductors**, whereas the latter are termed **thermal insulators**.

Chapter 2 (Continued)

SOME FUNDAMENTAL PRINCIPLES OF CHEMISTRY AND PHYSICS

PART 2 — FUNDAMENTAL PRINCIPLES OF CHEMISTRY

SECTION 1

BRANCHES OF CHEMISTRY

Traditionally, the science of Chemistry embraces two great divisions—**Organic Chemistry** and **Inorganic Chemistry**. Organic Chemistry deals with the composition and properties of compounds of carbon with the other chemical elements, usually excepting carbon monoxide, carbon dioxide, cyanides, the carbides and the carbonates. Inorganic Chemistry includes the study of all other substances, including the excepted compounds containing carbon just mentioned. The same general principles and laws are applicable to the study of both Organic and Inorganic Chemistry; the chief distinction between these two divisions of chemical science only involves the classes of compounds with which they are concerned. Since most metallurgical processes involve inorganic compounds, this discussion will deal largely with Inorganic Chemistry.

Physical Chemistry is devoted to the quantitative study of relationships between the chemical and physical properties and the constitution of substances, and is

a powerful metallurgical tool. Two branches of Physical Chemistry pertinent to metallurgical processes are **Thermochemistry** and **Electrochemistry**. **Thermochemistry** deals with the changes of heat content accompanying chemical reactions; since heat is either evolved or absorbed in all chemical reactions, this branch of chemical science is of basic importance in metallurgy, and is closely related to the physical science of **Thermodynamics** which is devoted to the mechanical effects of heat. **Electrochemistry** covers the field of study of the relationships between electricity and chemical changes: chemical changes in cells or batteries produce electricity, passage of direct electric current through molten substances or solutions produce chemical changes. Aluminum made by electrochemical processes, tin electrolytically deposited on steel to make electrolytic tin plate, and copper refined by electrolysis to a high degree of purity, are three examples of the importance of electrochemistry in metallurgy.

SECTION 2

CHEMICAL FORMULAS AND REACTIONS

Chemical Formulas of Compounds—The method of representing the elements by symbols, together with the system of atomic weights, affords a convenient and concise method of representing chemical compounds, or to be more explicit, the molecules of chemical compounds. Thus, by analysis, water is found to be composed of hydrogen and oxygen in the proportion of eight parts of oxygen to one part of hydrogen by weight. These facts are completely expressed by the formula H_2O , which indicates a molecule of a compound composed of two atoms of hydrogen and one atom of oxygen, or, since the atomic weight of hydrogen is 1 and oxygen 16, 2 parts of hydrogen to 16 parts of oxygen (1 to 8). Likewise, the formula Fe_2O_3 represents a compound, the molecule of which is made up of 111.70 parts of iron to 48 parts of oxygen.

Molecules of Elements—In studying chemical changes in which elements are set free, it is found that they are much more active at the instant of their liberation than afterwards. This fact leads to the belief that the instant

an element is set free from its compounds it exists in the atomic condition, but if there is nothing else present with which the atoms can combine, they combine with each other to form **molecules** of the element. This idea cannot be proved in the case of solids, but its correctness is easily shown in the case of gases. From many facts, Avogadro was able to show that **equal volumes of all gases, under standard conditions of temperature and pressure, contain the same number of molecules**. Hence, the molecular weight in grams of all gases occupies a constant volume of 22.4 liters, called the **gram-molecular volume**. Now, the weight of 22.4 liters of oxygen equals 32 grams; of hydrogen, 2 grams; of nitrogen, 28 grams. Dividing these weights by the respective atomic weights of the elements, the quotient is 2 in each case. Hence, the molecules of these elements contain two atoms each, and the correct formulas for these elements are O_2 , H_2 , and N_2 , respectively. Helium and some other elements do not have this tendency to unite into groups; in these elements, atom and molecule are iden-

tical. **Molecules**, therefore, are defined as the smallest particles of a substance that retain the characteristics of that substance.

Chemical Equations—This system of symbols and weights also simplifies the representation of chemical changes. Suppose it is desired to represent the chemical change that takes place when a common substance, like coal, burns. Coal is largely made up of carbon; the element which combines with carbon during the burning (combustion) of coal is oxygen in the air. An invisible gas, carbon dioxide (CO_2), is formed and diffuses into the air. This change, spoken of as a reaction, is represented in the form of an equation. Since matter is conserved, there must be as many atoms on one side of the equation as on the other. This is shown by placing a 2 before O on the left side of the equation, thus, $\text{C} + 2\text{O} = \text{CO}_2$. This process is called balancing. Thus, chemical equations tell not only the formulas of the reacting substances and of the products formed, but also give the proportions involved by weight, and in the case of gases, volume relations as well, if the reaction is balanced in molecules. Thus, $\text{C} + \text{O}_2 = \text{CO}_2$ indicates one volume of oxygen unites with C to form one volume of CO_2 . It will be observed in the latter form of the formula that the two atoms of oxygen involved in this reaction are expressed as O_2 instead of 2O as in the first form. As mentioned above under "Molecules of Elements," oxygen exists naturally as molecules consisting of two atoms. The molecular formula for oxygen (and other substances whose molecular formulas are known) is always used in writing equations to express this fact.

In writing chemical equations, considerable knowledge of a specific character is essential. First, it is necessary to know under what conditions the substances are brought together in order to tell whether or not they will react. If they do react, then the products formed and the formulas of all these substances must be known. As an example, consider the reaction of iron (hot) with water (steam) to form ferrosferric oxide and hydrogen. This knowledge can then be indicated thus: $\text{Fe} + \text{H}_2\text{O} = \text{Fe}_3\text{O}_4 + \text{H}_2$. The equation must be balanced, since the same number of atoms must be on both sides of the equality sign. This can be done by inspection. Finally, the reaction is reversible, for if instead of steam over hot iron, hydrogen be passed over hot iron oxide, iron and water are the products. The reaction is, therefore, correctly written thus: $\text{Fe}_3\text{O}_4 + 4\text{H}_2 \rightleftharpoons 3\text{Fe} + 4\text{H}_2\text{O}$, or $3\text{Fe} + 4\text{H}_2\text{O} \rightleftharpoons \text{Fe}_3\text{O}_4 + 4\text{H}_2$. Many reversible reactions, under conditions which do not permit the products to escape from the field of action, do not proceed to completion, but reach a balanced condition after a time and seem to stop, though as a matter of fact they are progressing in one direction as rapidly as in the other, hence are described as being in dynamic equilibrium.

The rate at which any reaction takes place depends on temperature, pressure or concentration, and the effect of any catalyst present. A catalyst changes the rate of reaction without itself being altered.

Heat of Reaction—The heat of reaction is the quantity of heat absorbed or liberated in a chemical change. Its value depends on the physical state of the reacting materials and products as well as the temperature. It is usual to give the standard heat of reaction which is defined as the change in heat content resulting from the reaction under a pressure of one atmosphere, starting and ending with all materials at a temperature of 18°C or 64.4°F .

Heat of formation represents a special case of the heat of reaction. The reacting materials are the necessary elements and the compound in question is the only product formed. Tables of the heat of formation of com-

pounds will be found in the standard texts on thermochemistry. By combining the heats of formation of various compounds, it is possible to calculate the heat of reaction for any reaction. The heat of formation of elements is zero. The heat of reaction is the difference between the sum of the heats of formation of the products and the sum of the heats of formation of the reactants.

The correction of the standard heat of reaction to the heat of reaction at some other temperature is necessary in actual practice, since few reactions take place at exactly 64.4°F . The calculation is based on a consideration of the initial and final states of any system. The same net amount of heat is required to go from some initial state to some final state regardless of the path or method used in making the change. If this were not true we would go by the path requiring the least heat and come back by the path requiring the most heat. This would give us a net amount of heat, and we would have a source of heat without doing any work or a kind of perpetual motion.

To make the correction, the amount of heat required to raise the reacting materials and products from 64.4°F to the temperature of reaction are calculated from the heat capacities, and any heats of change of state and transformation. Now the standard heat of reaction plus the heat added to the products is equal to the heat added to the reacting materials plus the heat of reaction at the desired temperature.

Kinds of Reactions—As already indicated, all reactions may be placed under one of two heads; namely, those that liberate heat, called exothermic, and those that absorb heat, called endothermic. A more detailed classification, such as the following, is sometimes employed:

1. Direct combination: $2\text{H}_2 + \text{O}_2 = 2\text{H}_2\text{O}$.
2. Direct decomposition: $2\text{HgO} = 2\text{Hg} + \text{O}_2$.
3. Simple replacement or substitution:
 $2\text{H}_2\text{O} + 2\text{Na} = 2\text{NaOH} + \text{H}_2$.
4. Double replacement:
 $\text{BaCl}_2 + \text{H}_2\text{SO}_4 = \text{BaSO}_4 + 2\text{HCl}$.
5. Oxidation— $\begin{cases} 3\text{Fe} + 2\text{O}_2 = \text{Fe}_3\text{O}_4 \\ 2\text{FeCl}_2 + \text{Cl}_2 = 2\text{FeCl}_3 \end{cases}$.
6. Reduction— $\begin{cases} \text{Fe}_3\text{O}_4 + 4\text{H}_2 = 3\text{Fe} + 4\text{H}_2\text{O} \\ 2\text{FeCl}_3 + \text{H}_2 = 2\text{FeCl}_2 + 2\text{HCl} \end{cases}$.

The two processes of oxidation and reduction are of great importance in metallurgy. They have a triple meaning. Primarily, oxidation means the taking on of oxygen by an element or compound, and reduction means the giving up of oxygen. In the case of elements that form more than one compound, if the number of atoms of one that combines with a fixed number of the other be increased, the process is oxidation; if decreased, reduction. In metallurgy an element in the metallic state is said to be reduced. The two processes are inseparable; when one thing is reduced, another is oxidized. In metallurgical operations these two processes are of paramount importance, for all the substances reduced constitute the metallic product and all in oxidized form make up the slag.

SECTION 3

KINDS OF CHEMICAL COMPOUNDS

A close study of a great number of chemical compounds shows that these substances fall into four classes; namely, acids, bases, salts, and nonelectrolytes.

Acids are characterized by the fact that they all have a sour taste when in water solution and change the color of certain chemicals, called indicators. A common indicator is litmus, the color of which is red in the presence of acids.

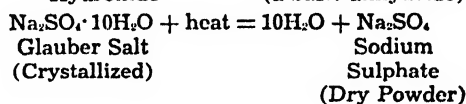
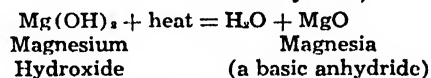
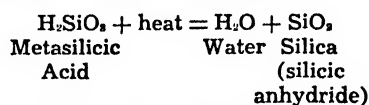
Bases have the power of neutralizing acids, and may be looked upon as their opposites. Examples are quicklime, lye, etc. Bases change the color of red or neutral litmus to blue.

Salts are the products formed when acids are neutralized by bases. Common table salt, made by neutralizing hydrochloric acid with sodium hydroxide, is an example. As a rule acids, bases, and salts are electrolytes, that is, their water solutions will conduct the electric current.

Nonelectrolytes—There are some compounds that do not resemble either acids or bases, nor can they be classed as salts. They are characterized by the fact that their water solutions will not conduct the electric current, so are termed nonelectrolytes. Ordinary sugar is an example.

Anhydrides—Inorganic acids and bases, in general, can be decomposed by heat. Acids and bases are broken up into water and oxides, the oxides being called anhydrides. Acids give acid anhydrides, and bases, basic anhydrides. These anhydrides, at elevated temperatures, have the same power of neutralization that their corresponding compounds possess in water solution. Thus,

acid and basic anhydrides can react to form neutral compounds to which the term **slag** is applied instead of salt. Many salts, in crystallizing from aqueous solutions, unite with or, better, take up a definite amount of water, which does not go to form a new compound, but to form crystals, and is called, therefore, **water of crystallization**. This water is held very loosely by the molecule and is readily given up by it. In some crystals, like those of washing soda, for example, this tendency is so pronounced that they give up their water of crystallization to the air, if its humidity is low. Such substances are said to be **efflorescent**. On the other hand, many dry substances absorb moisture from the air and are, therefore, said to be **hygroscopic**. A few substances will absorb enough water from very moist air to become wet and actually go into solution in the water they absorb. These substances are said to be **deliquescent**. The following reactions will serve to illustrate these facts insofar as they involve chemical changes:



SECTION 4

VALENCE AND VALENCE NUMBERS

Valence—Reference was made in Section 3 of Part 1 of this chapter, under "The Electronic System," to the designation as **valence electrons** of the electrons in the two outermost "shells" or energy levels of the electronic system of the atom. In some chemical changes, an actual transfer of electrons from an atom of one element to an atom of another element takes place. An atom or group of atoms that loses electrons becomes positively charged; any that gain electrons become negatively charged, since the normally electrically neutral state of the atom is unbalanced by the gain or loss of negatively charged electrons. Two atoms that have become oppositely charged may become bound together by these electrical charges of opposite sign that result from an actual transfer of electrons. What are called **ionic substances** are formed in this manner.

Atoms of two (or more) elements involved in a chemical change may, instead of having electrons actually transferred between atoms, share electrons; the shared electrons then link the atoms together to form molecules.

In the case of ionic substances, the charge on an element in a given compound is the **ionic valence** of that element, and is either positive or negative as described for the case of the sodium ions (Na^+) and sulphate radical ions (SO_4^{--}) mentioned later in the discussion of "Ions and Electrolysis."

When molecules of a substance are formed by two

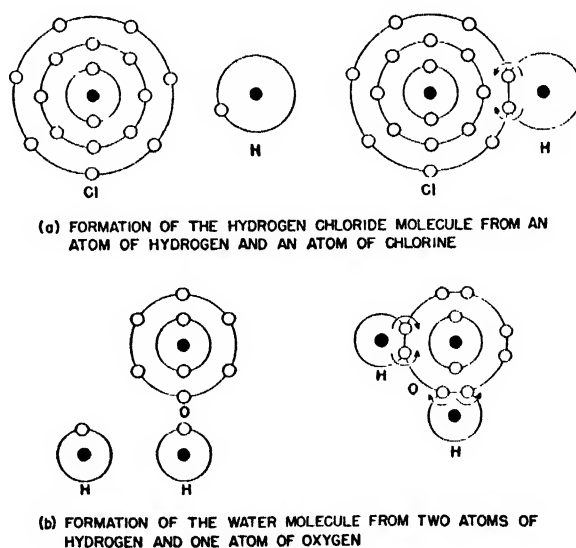


Fig. 2-7. Schematic representation of covalence in which molecules are formed by atoms sharing electrons. These are special cases of non-ionic valence.

atoms sharing electrons, the atoms are said to have **non-ionic valence**. Figures 2—7a and 2—7b show special cases of non-ionic valence, where molecules are formed by atoms sharing pairs of electrons in which each electron of a pair is spinning in an opposite direction; this is termed covalence which, unlike ionic valence, is neither positive nor negative.

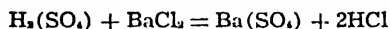
Valence Numbers—In balancing chemical equations, as will be discussed later, the **valence number** representing the **apparent valence** of an element is a useful aid. To determine valence numbers of the atoms in a compound, it is customary to arbitrarily consider each hydrogen atom as having a unit positive charge and each oxygen atom as having a double negative charge. The valence numbers of any other atoms in the compound are then calculated to be either positive or negative and of such magnitude that the algebraic sum of the valence numbers of all the atoms in the compound is zero. For

example, in sulphuric acid (H_2SO_4), the two hydrogen atoms would have a total charge of $2 \times (+1)$ or $+2$, and the four oxygen atoms a total charge of $4 \times (-2)$ or -8 ; the sulphur atom then would have a valence number of $+6$ if the numerical sums of the charges of opposite sign are to be equal. It should be stressed that a valence number of $+6$, as in this case, does not mean that the sulphur atom actually has lost six electrons to acquire a positive charge of $+6$; in other words, valence numbers are for convenience in chemical calculations, and do not necessarily represent **true valence**. Valence numbers indicate the **valence state** of an element in a particular compound. In the case of sulphur in sulphuric acid just mentioned, the valence state of the sulphur would be indicated by S^6 , which is called a **valence state symbol**. Atoms with valences of one, two, three, etc., respectively, are said to be univalent, divalent, trivalent, etc.

SECTION 5

RADICALS

In the molecules of many chemical compounds, certain groups of atoms appear to be more closely bound together than others in the same molecule. In these groups the atoms composing them appear to bear a fixed relation to each other, which remains unchanged during a chemical reaction. Thus, in many wet reactions in which H_2SO_4 is employed as a reagent, the sulphur and oxygen do not separate but remain closely combined, as illustrated in the reaction that takes place between this acid and barium chloride to form barium sulphate and hydrochloric acid:



Such a group of atoms, as typified by (SO_4) above, is called a **radical**. Thus, a radical may be defined as a group of atoms which are combined chemically in such a way that they act as an atom of a single element. Among the more commonly encountered radicals are the sulphate radical just mentioned (SO_4), the nitrate radical (NO_3), the phosphate radical (PO_4), the hydroxyl radical (OH) and the ammonium radical (NH_4).

SECTION 6

IONS AND ELECTROLYSIS

Distilled water, that is, water which contains no dissolved chemicals, has a very high resistance to the flow of electric current. However, when a substance like sodium chloride is dissolved in distilled water, the resulting solution will readily conduct a current. The reason is that the sodium chloride dissociates into electrically charged units, or parts of the molecule, called **ions**. It is these ions which are conducting the current.

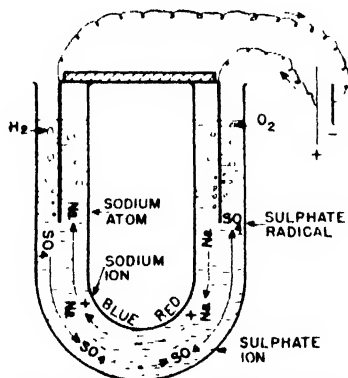
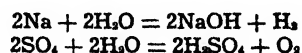


FIG. 2—8. Decomposition of water and formation of sodium hydroxide and sulphuric acid from sodium sulphate by electrolysis.

The following simple experiment may be employed to throw additional light upon this subject. Into the U-tube of Figure 2—8 is placed a solution of sodium sulphate and some neutral litmus, into which are immersed two small platinum rods to act as electrodes for an electric current. Upon closing the circuit, bubbles of hydrogen are given off at the cathode or negative pole and bubbles of oxygen at the anode or positive pole, while the solution about the cathode becomes deep blue in color, showing it is basic, and that about the anode becomes red, showing it to be acid. These facts are explained by assuming that the molecules of dissolved Na_2SO_4 dissociate into parts called ions. This dissociation is indicated thus: $\text{Na}_2\text{SO}_4 = \text{Na}^+ + \text{Na}^+ + \text{SO}_4^{--}$. The sodium ions, Na^+ , carrying a positive charge of electricity, are propelled by the current toward the cathode (the negative pole), while the negatively charged sulphate ions, SO_4^{--} , each carrying a double negative charge, go to the anode or positive pole. Here they give up their charges and become chemically active, decomposing the water thus:



Hydrogen and oxygen are the products. It is the basic sodium hydroxide (NaOH) which causes the solution at the cathode to become blue, and the sulphuric acid (H_2SO_4) which causes the solution at the anode to turn

red. However, if the portions of solution were brought together by stirring, the hydroxide and acid would react to reproduce once more the original sodium sulphate. This experiment is but one example of electrolysis. Any inorganic acid, base, or salt may be substituted for the sodium sulphate; and any conductor of electricity may be used instead of platinum.

It has been found that one gram equivalent weight of matter, that is the atomic weight in grams divided by the valence, is chemically altered at each electrode for each 96,500 coulombs, or one faraday, of electricity passed through the electrolyte. This is known as Faraday's Law. A coulomb is an amount of current equal to the flow of one ampere for one second.

SECTION 7

FUNDAMENTAL LAWS OF CHEMICAL CHANGE

Law of Constancy of Nature—Under the same conditions, reactions involving the same substances will always produce the same results.

Law of Conservation of Matter—Early in the history of science, a most fundamental conclusion was reached; namely, that matter can neither be created nor destroyed; in other words, the amount of matter in the universe cannot be increased or decreased. Thus, in every chemical process, the quantity of matter remains constant and only the form changes; that is, the total mass of substances which react chemically is equal to the mass of the resulting products. In recent years, nuclear physicists working on the splitting of atoms (nuclear fission) have discovered that actually matter can be converted into energy, so that strictly speaking this law is not entirely valid and must be amended. However, this matter need not be considered here, since for ordinary chemical and physical processes the law holds accurately.

The Law of Definite Proportions (Dalton's first law) states that when elements combine to form a substance, they do so (with a few exceptions) in definite proportions by weight. For example, 112 parts by weight of iron will combine with 48 parts by weight of oxygen to form Fe_2O_3 , or fourteen parts by weight of nitrogen will combine with sixteen parts by weight of oxygen to form nitric oxide (NO). The definite weights are called combining weights.

Some pairs of elements form more than one compound, and the combining weights of the elements in these compounds are simple multiples of each other. This fact is stated in the **Law of Multiple Proportions** (Dalton's second law) as follows: when two elements unite to form more than one compound, if a fixed weight of the one is considered, the weights of the other which combine with it are integral multiples of one another. The following compounds are examples:

Compound	Parts by Weight Nitrogen	Parts by Weight Oxygen
Nitrous oxide, N_2O	28	16
Nitric oxide, N_2O_3 or NO	28	32
Nitrogen trioxide, N_2O_5	28	48
Nitrogen peroxide, N_2O_4 or NO_2	28	64
Nitrogen pentoxide, N_2O_5	28	80

Since atoms do not subdivide in chemical changes, it is easy to see that the foregoing two laws must be true, since whole numbers of one type of atom will always combine with whole numbers of other atoms. Since atomic weights are fixed, the combining weights would have to be simple multiples related to the numbers of atoms involved.

Qualitative Considerations—In practical chemical work it is usually desirable to have reactions go to an end. Hence it is desirable to have in mind two laws of equilibrium. According to **Le Chatelier's Law**, the application of a stress, i.e., change of temperature, concentration, or pressure, to a system in equilibrium causes a reaction, displacing the equilibrium in the direction that tends to undo the effect of the stress.

A particular case of this law is van't Hoff's law of mobile equilibrium which states that if the temperature of a system in equilibrium is raised, the equilibrium point is displaced in the direction that absorbs heat. This law applies to both chemical and physical equilibria.

As an aid in writing reactions the following laws may be found of value:

- A. The reaction of two or more substances will go to an end, that is, will be complete, provided:
 - (1) One of the products is volatile at the temperature of the reaction.
 - (2) One of the products is insoluble in the solvent in which the reaction takes place.
 - (3) One of the products is a nonelectrolyte, that is, does not ionize in the solvent.
- B. The speed of a chemical action in a given direction may be increased by effecting a greater concentration of one of the reacting substances. This is a simple, nonmathematical statement of the law of mass action.
- C. Chemical reactions always tend to proceed in the direction that will liberate the most heat, and without the addition of heat from an external source those substances that have the greatest heats of formation will tend to form.

SECTION 8

SOLUTIONS

There are no good rules to predict the extent to which a substance will dissolve in a liquid. In dilute solutions the component present in large excess is called the solvent, and the dissolved substance is called the solute. The concentration of the solute may be expressed in mol fractions. The mol fraction of a solute in a solution is the number of molecular weights of the solute present, divided by the number of molecular weights, or mols,

of all substances present. An example of calculating mol fraction will be found in the section of this chapter on chemical calculations. The sum of the mol fractions of the various components in a solution must be one.

Distribution Coefficient—If two liquids are insoluble in each other and a third component, the solute, is added, the solute will distribute itself between the two immiscible layers in such a way as to satisfy its solu-

bility in the two liquids as well as its ability to react with either or both of the liquids. The ratio of its concentrations in the two layers is called the **distribution coefficient**. The distribution coefficient is a constant for each pair of mutually insoluble liquids with a particular solute. The constant changes with temperature. It is a constant regardless of the amount of solute or relative quantities of the two immiscible solvents.

Vapor Pressure of Solutions—Several rules have been developed for calculating the vapor pressure of solutions. One is **Raoult's Law** for the solvent. This states that the partial vapor pressure of the solvent in equilibrium with a solution is directly proportional to its mol fraction. That is, the partial vapor pressure exerted by the solvent is equal to the vapor pressure of the pure solvent at the temperature in question times the mol fraction of solvent present. If the solute is non-volatile the partial vapor pressure of the solvent is the total vapor pressure. **Henry's Law** states that the partial vapor pressure due to a volatile solute is proportional to its mol fraction. The value of the proportionality constant for Henry's Law must be determined experimentally for each system and at each temperature.

Phase or Constitution Diagrams—Many metals are soluble in each other (at least partially) in the liquid state, and some remain dissolved in each other in the solid state after their liquid solution has cooled sufficiently to change from the liquid to the solid state. As a special part of the study of solutions, a knowledge of the changes in metals and alloys with variation in chemical composition and with mechanical and thermal treatments can be expressed by phase or composition diagrams, which, however, refer only to systems that are in equilibrium. A large number of these diagrams covering most of the binary alloys and many multi-component alloys have been determined experimentally. These have become the basis for deciding compositions and heat treatments to give the desired properties for a particular application. These diagrams, also called **phase, equilibrium composition or constitution diagrams**, are widely used in the refractory field.

The phases of a system are the parts which are separated from one another by definite physical boundaries. For example, a liquid in contact with a solid has a definite physical boundary, thus two phases, a liquid and a solid, are present. Also, since every solid phase is characterized by a distinctive atomic configuration, a discontinuous change in type or spacing of atom packing is necessarily a phase change. Two mutually insoluble

materials, whether as liquids or as solids, show two phases.

Time-Temperature Curves—One method that can be used to determine the phase diagram is to heat a mixture until it melts and then to plot the change in temperature with time as the mixture cools with a constant outside temperature environment. The shape of time-temperature curve obtained will depend on the heat loss of the system and the energy changes occurring within the system. As long as only liquid is present, cooling (temperature-drop versus time) will proceed at a gradually decreasing rate, and the time-temperature relationship will be a continuous curve. However, whenever material is crystallized, or solidified, the heat of fusion is liberated, and this liberation of the heat of fusion within the material furnishes the heat that is being lost from the system, causing the temperature to remain fairly constant. This results in a leveling or "jog" in the time-temperature curve. Whenever all the material has solidified the temperature will again drop, and the time-temperature curve will again become continuous.

If, during the cooling of the solid to room temperature, there is a crystal change or the formation of an additional phase because of mutual solubility decrease, the heat of transformation will be liberated and another jog in the time-temperature curve will be noted. Some of these changes are accompanied by such a small liberation of heat, or at such a slow rate, that the effects on the time-temperature curve are slight and other methods, such as micrographic analysis, X-ray diffraction, dilatometric measurements, etc., must be used to find the temperature where the change to another phase takes place.

After obtaining the time-temperature curves for a number of compositions, phase diagrams can be constructed by plotting the temperature where jogs appear in the time-temperature curves as a function of the composition of the alloy being studied. A study of the freezing of solutions has shown that they fall into two classes: (1) those in which the ingredients in solution in the liquid state remain in solution in the solid state, and (2) those in which the state of solution is not maintained in the solid state, that is, those in which the ingredients separate on freezing. Both classes are encountered in the study of alloys.

An Example of the First Class of Solutions—One of the best examples of the first kind of solution is a mixture of gold and silver. If quantities of these two metals

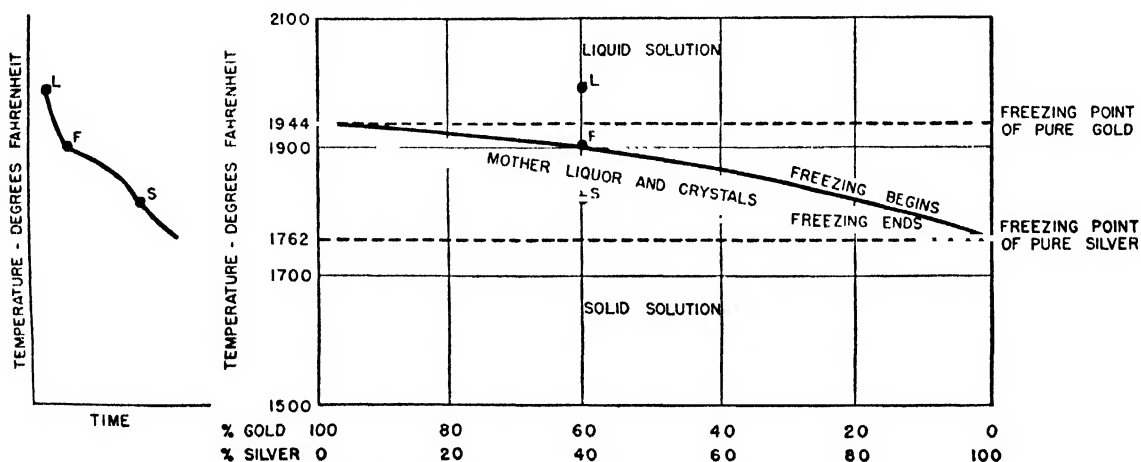


FIG. 2-9. Diagram of the freezing of liquid gold-silver alloys.

be placed in a vessel and heated until they melt, a homogeneous mixture, or a liquid solution, results; and if this mixture be allowed to cool to the solid state, it remains homogeneous, that is, it is a solid solution. A study of many mixtures in which the proportions of gold to silver are varied shows that freezing begins and ends at a different temperature for each mixture. Pure gold freezes at 1444° F (1949° F according to some authorities) and pure silver at 1762° F, and the freezing points of mixtures occur between these two points. Unlike the pure metals, however, these mixtures do not solidify completely at a constant temperature, but their freezing is prolonged through ranges of temperature. These facts, determined definitely by experiment, may be represented by a diagram, or curve, in which the ordinates represent temperatures and the abscissae the percentage of gold or silver or both.

A typical time-temperature curve for this system is shown in Figure 2—9, to the left of the diagram. Point "F" shows the temperature where freezing starts and point "S" is the temperature where freezing ends and below which only solid is present.

To illustrate further, suppose 60 oz. of gold be mixed with 40 oz. of silver, and the whole heated to a temperature of 1994° F. The locus of this point would be at "L" in the region of the liquid state. If now, this molten solution be allowed to cool slowly, crystallization will begin at "F," about 1906° F, and end at "S," about 1814° F. The first crystals formed will be richer in gold than the liquid—the proportion being, in this case, about 90 parts gold to 10 parts silver—while the crystals formed last will be richer in silver than the mother liquid; but the average composition of all will be the same, namely 60 per cent gold and 40 per cent silver. Furthermore, while solidification is going on, a process of diffusion takes place and tends to adjust the composition of each individual crystal, thus bringing it nearer that of the average. Therefore, under normal (non-equilibrium) cooling conditions, each crystal exhibits a continuous variation in composition from the center, which would be the first to freeze, to the outside which would be the last to freeze.

An Example of the Second Class of Solutions—The second class of solutions exhibits complete solubility in the liquid state and only partial solubility in the solid state. An example is the silver-copper series of alloys, the phase diagram of which is shown in Figure 2—10.

The 5 per cent copper—95 per cent silver alloy in this series is typical of alloys containing up to 8.5 per cent

copper, and is indicated as point "N" on the phase diagram. Suppose this alloy be heated to a temperature of 2000° F. As this temperature is above the fusion points of silver and copper, it is sufficiently high to insure that the mixture will be completely liquefied. On cooling this solution, no crystallization takes place until a temperature of about T_1 is reached, at which temperature, crystals of alpha phase (silver in which is dissolved a small amount of copper) begin to separate from the solution, making the remaining solution poorer in silver but richer in copper. The separation of the alpha phase causes a retardation of the rate of cooling, showing that heat is evolved thereby; and the freezing point of the mother liquid is lowered, so that no further separation of alpha phase takes place until more heat is abstracted. This retardation in the rate of cooling is reflected by a distinct change of slope in the time-temperature cooling curve. If the cooling be continued, however, the separation of the alpha phase will also continue. From the time-temperature cooling curves of alloys of different compositions, the points at which freezing starts; i.e., the points at which the slope of the curve changes, are plotted to give the line "MO" shown in Figure 2—10.

When the cooling with the accompanying separation of alpha has reached the temperature T_2 , all of the liquid has frozen to the alpha phase which has an average composition of 95 per cent silver and 5 per cent copper. The temperature at which freezing ends is reflected by another change in the slope of the time-temperature cooling curve and, from cooling curves of alloys of different composition, the temperatures at which this change of slope occurs are plotted against the composition of the alloy (line "MN" on the diagram).

On further cooling, nothing happens until the temperature T_3 is reached. At this temperature, small crystals of the beta phase (copper, in which is dissolved a small amount of silver) begin to separate from the solid alpha phase and this separation continues as the temperature decreases. Therefore, at room temperature, the equilibrium structure of the 5 per cent copper—95 per cent silver alloy consists of primary crystals of the alpha phase in which are dispersed, often in a geometrical pattern, smaller crystals of the beta phase.

The line "NP" on the phase diagram gives, for any particular alloy, the temperature at which the beta phase begins to separate from the alpha phase and thus shows how the solubility of the copper in the alpha phase is affected by temperature. The points for this line are usually determined by X-ray diffraction.

The behavior of alloys having compositions between points "N" and "O" on the phase diagram can be described by considering an alloy containing 20 per cent copper and 80 per cent silver. This alloy begins to freeze at a temperature of T_4 and the freezing proceeds in a manner similar to that described for the 5 per cent copper—95 per cent silver alloy. However, when the temperature of T_5 is reached, part of the alloy is still liquid and the composition of the remaining liquid, designated by the point "O," is called the eutectic composition. At this temperature, called the eutectic temperature, the remaining liquid freezes into minute crystals of alpha and beta phase.

The heat liberated by the simultaneous freezing of the alpha and beta phases causes a distinct jog in the time-temperature cooling curve similar to that which occurs when a pure metal freezes. Thus, when cooling an alloy having a composition corresponding to the eutectic composition, the time-temperature cooling curve is the same as that obtained when a pure metal freezes.

Alloys whose compositions lie to the right of the eutectic composition freeze in a manner similar to that

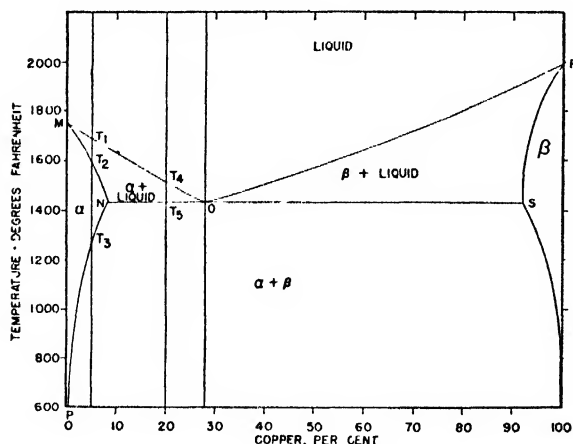


FIG. 2—10. An example of second class of solutions. Freezing of liquid silver-copper.

described above except that for the alloys to the right the beta phase is the first to separate.

Ternary Systems—As the number of components in a system increases, the graphical presentation of equilibrium conditions in that system becomes more difficult. With a three component system, a three-dimensional diagram is required to show all compositions within the system. Each apex of the equilateral triangle forming the base of the three dimensional diagram represents 100 per cent of one of the components, and any point within the triangle must always add up to 100 per cent, simple geometric relationships determining the percentage of each component present. With the diagram thus devoted to composition, there remains the problem of representing the other variables of pressure and temperature. Accordingly, pressure is held constant, usually at one atmosphere, as with binary systems, while tem-

perature is represented by an ordinate projecting upward at right angles from the base composition triangle. Thus, with most ternary diagrams, the reader is really looking at the uneven top surface of a triangular prism whose height at any point on that surface represents the liquidus temperature for that composition. However, temperatures are usually projected onto the base composition triangle where they appear as isotherms, together with the boundary lines of phases in equilibrium at the liquidus temperatures.

While the foregoing is sufficient to provide considerable useful information concerning a ternary system, the ability to follow the devious paths of melting or crystallization and to determine the shifting compositions and percentages of liquids and solids along these paths requires a far more intimate knowledge of phase-rule application than can be provided here.

SECTION 9

CHEMICAL NOMENCLATURE

General Principle—The names of the elements first discovered, and, therefore, unfortunately, the more common ones, are not based on any principle; but of the more recently discovered elements the metals have received names ending in *um* or *ium*, and the metalloids, in *n* or *ne*. In the naming of compounds, however, the old names have been discarded and new ones substituted. The system employed in assigning these new names is this: The name of a compound should show the elements of which it is composed, and as far as possible their relative proportions.

Terminology of Binary Compounds—The simplest compounds are those composed of only two elements. The names of all such compounds are made up of the name of the basic element, if one is present, succeeded by the name of the acid element, which ends in *ide*. Examples: ferrous (iron) sulphide, (FeS); sodium chloride, (NaCl); calcium oxide, (CaO).

In such cases as iron and sulphur, where the same two elements combine to form more than one compound, the compounds, when two in number, are distinguished by changing the ending of the metallic part of the name from *ous* to *ic*; thus, ferrous sulphide, (FeS); ferric sulphide, (Fe₂S₃); stannous chloride, (SnCl₂); stannic chloride, (SnCl₄). In the compounds for which the metallic element is given the *-ic* ending, the apparent valence of the metal is more positive than that of the metal in the *-ous* compounds. For example, the apparent valence of iron atoms in ferrous compounds is +2; in ferric compounds, +3. Often a prefix is added to the name of the acid element to indicate the number of its atoms to the molecule. Carbon dioxide (CO₂) and carbon monoxide (CO) are examples. When radicals are present in a compound, the compound contains the name of the radical unchanged. Zinc nitrate, Zn(NO₃)₂, and ammonium sulphate, (NH₄)₂(SO₄) are examples.

Terminology of Ternary Compounds—The names of compounds that contain three elements, provided they are not derived from acids, may end in *ide*. In this case all three of the elements appear in the name, as sodium aluminum fluoride, (Na₃AlF₆), bismuth oxychloride, (BiOCl). A few ternary compounds derived from *ic* acids have names ending in *ate*, as potassium chlorplatinate, (K₂PtCl₆).

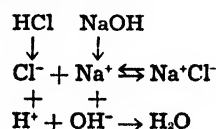
Terminology of Acids—Acids are composed of the acid-forming elements in combination with hydrogen or with hydrogen and oxygen. The name of a given acid is derived from the name of the acid-forming element and may end in *ic*. An example is hydrochloric acid (HCl).

Acids, like sulphuric (H₂SO₄) and orthophosphoric (H₃PO₄), which contain more than one replaceable hydrogen atom are called, as a class, **polybasic acids**; and in individual cases, the different acids are referred to as **di**basic or **tri**basic. For example, potassium and sodium may replace the two hydrogen atoms in H₂SO₄ to form sodium potassium sulphate (NaKSO₄). In all such double salts, both the base-forming elements must appear in the name of the salt.

Terminology of Bases—The base-forming elements form compounds with hydrogen and oxygen in which these two elements appear as a group, OH, called the **hydroxyl radical**. Hence, these compounds are called **hydroxides**. Sodium hydroxide, NaOH, is an example.

Terminology of Salts—Salts take their names from those of the base-forming elements and the acids of which they are composed, changing the endings of the acids. Salts of acids that end in *ic* change this ending to *ide* and *ate*, and those that end in *ous*, to *ite*. Thus, sodium chloride, (NaCl), derived from hydrochloric acid, sodium perchlorate, (NaClO₄), from perchloric acid, and sodium chlorite, (NaClO₂), from chlorous acid, are examples. Other systems of nomenclature are in use, but the ones just noted cover the largest field.

Neutralization—Hydroxyl and Hydrogen Ions—When an acid is mixed with water, dissociation of the molecules takes place and ions are formed which carry positive and negative charges. The positive ions are hydrogen and the negative ions are the acid radicals. When a base dissolves, a similar dissociation of the molecules takes place, the positive ion being the metal and the negative, the group OH, called **hydroxyl ions**. If the solutions of an acid and a base are mixed, a salt is formed from the union of the metal with the acid radical and water is formed from the union of the hydrogen ions with the hydroxyl ions. These changes may be illustrated with hydrochloric acid and sodium hydroxide in solution, and written thus:



From this it will be seen that, chemically defined, an acid is a compound which in solution gives hydrogen ions, while a base gives hydroxyl ions, and the fundamental neutralization reaction is the simple expression, $\text{H}^+ + \text{OH}^- \rightarrow \text{H}_2\text{O}$.

SECTION 10

CHEMICAL CALCULATIONS

Kinds of Problems—Chemical calculations are of great importance because they make it possible for the operator to predict the quantity of material needed for certain chemical and metallurgical reactions. In the case of solids (iron ore, limestone, etc.) the calculations are generally made in terms of weight. In the case of gases (oxygen, carbon dioxide, etc.) the calculations could also properly be made in terms of weight, but it is more convenient to think in terms of volumes, and, since the relation between weight and volume (of gases) is known, many gas calculations are carried out in terms of volumes.

The calculations are possible because chemical elements (such as carbon or oxygen) or compounds (such as carbon monoxide or calcium oxide) always combine with other elements or compounds in definite proportions. For example in the well-known metallurgical reaction in which oxygen (O) combines with carbon (C) to form carbon monoxide (CO), it will always happen that 16 grams of oxygen will combine with just 12 grams of carbon (neither more nor less) to form 28 grams of carbon monoxide. Similarly, when carbon monoxide (CO) gas is burned by reaction with oxygen to form carbon dioxide (CO₂) gas, it will always happen that 16 grams of oxygen will combine with 28 grams of CO to form 44 grams of CO₂. Sixteen grams of oxygen will never require 27 or 29 or 30 grams of CO, but always just 28 grams of CO to make CO₂.

The reason for the foregoing is the fact that one atom of oxygen always combines with just one atom of carbon in the first reaction above, and one atom of oxygen weighs 16 units and one atom of carbon weighs 12 units on the atomic scale.

These circumstances have resulted in a kind of chemical shorthand, in which the first reaction above is written as follows:



To the chemist, this says at once that the elements carbon and oxygen react here in the ratio of one atom of carbon to one atom of oxygen, and since the chemist knows that oxygen has an atomic weight of 16 and carbon has an atomic weight of 12, he knows that 16 units of oxygen will always react with 12 units of carbon, by weight. (The expression " $\frac{1}{2}\text{O}_2$ " is used in the equation to represent one atom of oxygen in recognition of the fact that oxygen normally exists in the molecular state, with two atoms in each molecule.)

A combination of atoms is called a molecule. Thus CO is a molecule. Its weight is called the molecular weight and is equal to the sum of the weights of the atoms of which it is composed. CO is called the formula of carbon monoxide.

The principles of calculation, using round numbers for atomic weights in some instances, are illustrated in the following simple examples.

Four Problems Involving Weight

(1) **Calculation of molecular weight from the chemical formula, the separate atomic weights being known.** This is a simple arithmetical addition. Suppose the formula of copper sulphate is known to be CuSO₄, and suppose the atomic weights are known to be as follows:

Copper (Cu)	atomic weight 63.54
Sulphur (S)	atomic weight 32.06
Oxygen (O)	atomic weight 16.00

Then the molecular weight of copper sulphate CuSO₄ must be:

1 Copper	63.54 × 1 = 63.54
1 Sulphur	32.06 × 1 = 32.06
4 Oxygen	16.00 × 4 = 64.00
Sum	= 159.60

The molecular weight of copper sulphate, CuSO₄, is thus 159.60.

(2) **Calculation of percent by weight of each element in a compound, the atomic weights being known.** Using the data of Problem No. 1 just above, it is recognized that the weight of the copper, Cu, is 63.54 and the weight of the copper sulphate, CuSO₄, is 159.60. Consequently the copper constitutes

$$\frac{63.54}{159.60} \text{ or } 39.81\% \text{ by weight, of the copper sulphate.}$$

In the same way the sulphur constitutes

$$\frac{32.06}{159.60} \text{ or } 20.09\% \text{ by weight, of the copper sulphate.}$$

Likewise, the oxygen constitutes

$$\frac{64.00}{159.60} \text{ or } 40.10\% \text{ by weight, of the copper sulphate.}$$

The total of course adds up to

Copper	· 39.81%
Sulphur	· 20.09
Oxygen	· 40.10
Total	<u>100.00%</u>

(3) **Calculation of the Formula of a Compound, from its Chemical Composition.** A substance, known to be a chemical compound and known to be pure, is of the following composition:

Calcium, Ca	29.44% by weight
Sulphur, S	23.55% by weight
Oxygen, O	47.01% by weight
Total	<u>100.00%</u>

The problem is to find the molecular formula, the atomic weights of the three elements being known to be:

Calcium	40.08
Sulphur	32.066
Oxygen	16.00

It will be seen that this is just the reverse of problem 2, where it was desired to calculate, from the formula, the proportion (by weight) of each element, instead of the formula from the proportions by weight (as in this case).

In this case, a certain number of atoms of calcium, of atomic weight 40.08, make up 29.44% of the total weight. Then 29.44 divided by 40.08 is a number which is proportional to the number of atoms of calcium.

$$\frac{29.44}{40.08} = 0.7344 \text{ for Ca}$$

Similarly, 23.55 divided by 32.066 is a number which is proportional to the number of sulphur atoms.

$$\frac{23.55}{32.066} = 0.7344 \text{ for S}$$

Similarly, 47.01 divided by 16.00 is a number which is proportional to the number of oxygen atoms.

$$\frac{47.01}{16.00} = 2.938$$

The calcium, sulphur and oxygen atoms are thus present in the proportion

$$0.7344 \text{ to } 0.7344 \text{ to } 2.938$$

which is about the same as 1 to 1 to 4.

Consequently, the formula of the above compound would be written CaSO_4 , which means that each molecule of the compound contains one atom of calcium and one atom of sulphur and four atoms of oxygen, and is therefore the compound known as calcium sulphate.

(4) **Calculation of Relative Weights from the Chemical Equation.** Problem: Five per cent of a certain limestone is nonvolatile impurities and 95 per cent is pure calcium carbonate. What will be the weight of lime obtained from calcining 2000 lbs. of this stone?

Weight of impurities = 5% of 2000 = 100 lbs.

Weight of calcium carbonate = 1900 lbs.

Reaction on calcining ... $\text{CaCO}_3 + \text{heat} = \text{CaO} + \text{CO}_2$
Combining or atomic wts. $40 + 12 + (3 \times 16) =$
 $(40 + 16) + [12 + (2 \times 16)]$

Relative or molecular wts. $100 = 56 + 44$

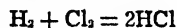
Since 100 lbs. CaCO_3 give 56 lbs. CaO

1900 lbs. CaCO_3 give 1064 lbs. CaO

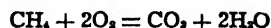
1064 lbs. CaO + 100 lbs. nonvolatile impurities =
1164 lbs. of burned product.

Problems Involving Volume Only

Calculation of Relative Volumes of Gases—Avogadro's hypothesis states that equal volumes of all gases at standard conditions of temperature and pressure, which are 32°F (0°C) and 29.92 in. (760 mm) of mercury barometric pressure, contain the same number of molecules. The weights of these equal volumes are proportional to the molecular weights of the gases. If the weights are expressed in grams, each gram-molecule of the gases involved represents 22.4 liters under standard conditions; if in pounds, each pound-molecule occupies 359 cubic feet. Problems involving volumes of gases only are, therefore, very simple to solve, because the relative volumes are identical with the coefficients of the molecules, as will be evident from an inspection of the following examples.



1 vol. hydrogen + 1 vol. chlorine gives 2 volumes hydrochloric acid gas.

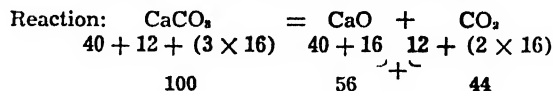


1 vol. methane + 2 vol. oxygen gives 1 vol. carbon dioxide + 2 vol. water vapor.

Problems Involving Both Weight and Volume

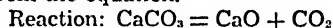
Indirect Method—This method necessitates finding the relative weights of the gases involved from which the volumes may be calculated from the specific gravity, or the weight of a unit volume.

Problem: How many cubic feet of carbon dioxide measured under standard conditions would be given off by 2000 lbs. pure calcium carbonate during the process of calcination? (The weight of one cubic foot of carbon dioxide is 0.1225 pounds at standard conditions.)



Since 100 lbs. CaCO_3 give 44 lbs. CO_2
2000 lbs. CaCO_3 give 880 lbs. CO_2
 $880 \div 0.1225 = 7180 \text{ cu. ft.}$

Direct Method—The fact that molecular weights of gases occupy constant volumes at standard conditions affords a simple direct method for calculating volumes from the equation.



Since the molecular weight of CaCO_3 is 100, then one pound-molecule of CaCO_3 is equal to 100 pounds. From the equation it can be seen that one pound-molecule of CaCO_3 yields one pound-molecule of CO_2 . Since one pound-molecule of any gas is equal to 359 cubic feet at standard conditions:

100 lbs. CaCO_3 gives 359 cu. ft. of CO_2
2000 lbs. CaCO_3 gives 7180 cu. ft. of CO_2

Mol Fraction

Problem: Find the mol fraction of the solute (sulphuric acid) and solvent (water) in a solution containing 10% by weight sulphuric acid in water.

Number Moles = $\frac{\text{Pounds}}{\text{Molecular Weight}}$

Take as the basis 100 pounds of solution.

Chemical Formula	Lbs.	Mol. Wt.	No. Mols	Total No. Mols	Mol Fraction
H_2SO_4	10	98	0.102	5.102	0.02
H_2O	90	18	5.000	5.102	0.98
	100		5.102		1.00

SECTION 11

SOME ELEMENTS COMMON IN STEELMAKING

Oxygen

Atomic Number, 8. Atomic Weight, 16. Molecular Weight, 32. Valence, II. Specific Gravity: Hydrogen Standard, 15.896; Air Standard, 1.1053. Density at 60°F and 30 in Hg.: 1.3553 g. per liter, 0.08461 lbs. per cu. ft.

This element is most widely distributed in nature; 49.85 per cent of the solid crust of the earth, 88.89 per cent of water, and 20.8 per cent of air is oxygen. In air it exists in a free state. In a combined state, it exists in limestone, sand, marble, clay, quartz, iron ore, and many

other substances. It is prepared by merely heating certain of its compounds, some of which are mercuric oxide, potassium chlorate and manganese dioxide; by the decomposition of water by electrolysis; and from the air by purifying processes. Oxygen is a colorless, odorless, tasteless gas, heavier than air and slightly soluble in water. At a low temperature it is converted into a liquid which boils at -297.4°F . The phenomenon of ordinary burning or combustion is due to the chemical combination of oxygen with other substances. It unites with many elements to form a class of compounds, the

oxides. It is necessary to life. Animals die in an atmosphere of less than 16 per cent oxygen. Some important oxides are carbon dioxide (CO_2), carbon monoxide (CO), calcium oxide (CaO), magnesium oxide (MgO), ferric oxide (Fe_2O_3), and ferrosferric oxide (Fe_3O_4). The last two are important as ores of iron.

Hydrogen

Atomic Number, 1. Atomic Weight, 1.008. Molecular Weight, 2.0156. Specific Gravity: Hydrogen Standard, 1; Air Standard, 0.06959. Density at 60°F and 30 in. Hg.: 0.08526 g. per liter, 0.005323 lbs. per cu. ft.

Hydrogen does not occur in nature in a free state, but combined with oxygen it forms water, of which it constitutes 11.11 per cent. In a combined state it occurs also in the bodies of plants and animals, hence, in the volatile matter of coal, in petroleum, and in natural gas, of which it constitutes almost 25 per cent. Water is always one of the products of combustion when a fuel containing hydrogen is burned. It can be prepared by decomposing water with sodium, potassium, hot iron, hot coke, or the electric current; by treating certain metals with certain acids; and by treating aluminum with sodium or potassium hydroxide. Hydrogen is a colorless, tasteless, odorless gas, almost insoluble in water. It can be converted into a liquid that boils at -422.9°F . It is the lightest substance known, being about $\frac{1}{16}$ as heavy as air and $\frac{1}{8}$ as heavy as oxygen. It is combustible and explosive, and combines with oxygen in the weight proportion of 1:8 to form water. Its great tendency to combine with oxygen makes it a powerful reducing agent.

Sulphur

Atomic Number, 16. Atomic Weight, 32.066. Valences: II, IV, VI. Specific Gravity: monoclinic, 1.96; rhombic, 2.06.

This element occurs free in the neighborhood of volcanoes and in underground deposits, from which it may be prepared by purifying processes. In the combined state it is found as FeS_2 , FeCuS_2 , ZnS , and PbS , the last three being valuable ores of copper, zinc, and lead, respectively. It also occurs as the sulphates, CaSO_4 , BaSO_4 , and PbSO_4 , and in animal and vegetable matter. Compounds of sulphur occur in iron ores, in limestone, and in coal and sulphur from these sources combines with the iron in the blast furnace. If this sulphur is present in the iron in too large amounts, it is undesirable because of its possible injurious effects on steel and cast iron.

Sulphur is a brittle, yellow crystalline solid which melts to form a straw-colored liquid. It is allotropic, i.e., can exist in monoclinic, rhombic and amorphous forms. When heated to a sufficiently high temperature, it combines with oxygen to form sulphur dioxide (SO_2), with iron to form ferrous sulphide (FeS), and with most of the metals, forming sulphides. The sulphur in iron or steel is in the forms of FeS and MnS , distributed almost uniformly throughout the metal while in the molten state. Upon solidifying, however, owing to the difference in density and fusion temperature between these compounds and the metal, they may, under normal conditions, segregate to some extent, causing some parts of the solidified mass to show a higher content of this impurity than the average, or of the whole in the molten state. With hydrogen it forms hydrogen sulphide gas (H_2S)—very important in chemistry. Sulphur is used in the manufacture of matches and black gunpowder, also for disinfecting purposes, and for vulcanizing rubber. Its chief use, however, is in the manufacture of sulphuric acid (H_2SO_4), which is a very important material in modern civilization. This acid is obtained by oxidizing sulphur dioxide (SO_2), a gas given off in

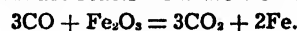
the roasting of FeS_2 , ZnS , CuS , and from the burning of sulphur.

Carbon

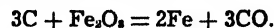
Atomic Number, 6. Atomic Weight, 12.01. Valences: II, IV. Specific Gravity: amorphous, 1.8–2.1; graphite, 2.25; diamond, 3.51.

This remarkable element occurs free in nature in crystalline forms as diamonds and graphite and in the amorphous form as coal. It is the chief constituent of the bodies of plants and animals, of all natural fuels, and of nearly all prepared fuels. It occurs in combined state in limestone, magnesite, marble and other carbonate rocks. Carbon is allotropic; diamond and graphite are familiar. The common amorphous forms are coal, lampblack, charcoal, coke, and bone black. Its density varies with its form. It is infusible, but sublimes at about 3500°C .

Carbon forms many compounds with hydrogen, called hydrocarbons, such as methane (CH_4), ethylene (C_2H_4), benzene (C_6H_6) and acetylene (C_2H_2), each of which is but the first member of a series of related compounds. With oxygen it forms carbon dioxide (CO_2), which is a product of combustion and of respiration. CO_2 is also given off when carbonates, such as limestone, are heated. The reaction is, $\text{CaCO}_3 = \text{CaO} + \text{CO}_2$. Carbon monoxide (CO) is formed in combustion when the supply of oxygen is insufficient for the formation of CO_2 . Thus, in the blast furnace, a fixed amount of air is blown against an excess of hot carbon, which act results in this reaction: $2\text{C} + \text{O}_2 = 2\text{CO}$. Owing to its tendency to combine with oxygen, forming CO_2 , CO is a good reducing agent. So, the CO formed in front of the tuyeres in the blast furnace reacts with the iron oxide thus:



Carbon by itself acts as a reducing agent in the metalurgy of iron.



Iron forms a carbide with carbon, the formula of which is Fe_3C . Carbon is also found uncombined in pig iron in the form of tiny flakes of graphite; hence, the term graphitic carbon. Carbon has a marked effect upon iron. The varied properties of steel and the multiplicity of uses to which it can be applied are due largely to the influence of this element. Carbon in steel up to a certain limit is, therefore, not to be considered as an impurity but as an essential element.

Silicon

Atomic Number, 14. Atomic Weight, 28.09. Valence, IV. Specific Gravity, 2.42.

Next to oxygen, silicon is the most abundant element in nature. It is the most important constituent of the mineral part of the earth. Sea sand, quartz, jasper, opal and infusorial earths are almost pure forms of silica (SiO_2), the principal compound. As silicates, it occurs in clay, mica, talc, hornblende and feldspar. On account of its wide distribution, it forms the chief impurity of iron ore, as well as of nearly all natural mineral deposits. With carbon it forms the carbide, SiC (carborundum). It also forms several acids, chief of which is orthosilicic acid, (H_4SiO_4), which loses water when heated and forms SiO_2 .



Thus, in whatever form silicon may occur in an ore, it is looked upon as SiO_2 . This substance is the great acid of dry chemistry and at high temperatures will neutralize any base with which it comes in contact. In the blast furnace some of the silica (SiO_2) contained in

the charge is reduced to silicon. The amount so reduced varies with the working conditions of the furnace, mainly the temperature. Once reduced, the silicon alloys with the iron and becomes a part of the metallic bath. Most of this silicon is reoxidized and removed in the various processes of making steel. However, silicon is beneficial to steel, so it is sometimes added in small amounts to control the oxygen content and in larger amounts to form important alloy steels.

Nitrogen

Atomic Number, 7. Atomic Weight, 14.008. Molecular Weight, 28.016. Valences: III, V. Specific Gravity: Air Standard, 0.9673.

This element constitutes about 78 per cent of the Earth's atmosphere. It occurs in niter beds as saltpeter (KNO_3), and Chile saltpeter (NaNO_3), also in organic compounds and in coal. It is an odorless, tasteless, colorless, relatively inert gas. With hydrogen it forms ammonia (NH_3); with oxygen, a series of oxides, N_2O , NO , N_2O_3 , N_2O_4 , and N_2O_5 ; and with hydrogen and oxygen, an important acid, nitric acid (HNO_3). Its presence in the air as a diluent in so large amounts makes it an important factor in furnace practices. At lower temperatures it can be made to combine with iron to form a nitride which imparts some hardness even at low concentration.

Phosphorus

Atomic Number, 15.0. Atomic Weight, 30.975. Molecular Weight, 124.096. Valences: III, IV. Specific Gravity: white or yellow, 1.83; red, 2.05 to 2.34; black, 2.69.

Phosphorus, always combined with other elements, occurs widely distributed in limited amounts, particularly in soils. It is found in practically all iron ores. It occurs in deposits as phosphorite and apatite, and it is an important constituent of bone. While phosphorus belongs in the same group of elements as nitrogen, it does not resemble it very much from a physical standpoint. It is allotropic and exists in two forms, as a pale yellow solid that melts readily at the low temperature of 111.4°F , and as a red form quite different in properties. While it is a much more active element, it closely resembles nitrogen chemically. It forms compounds with hydrogen and oxygen, such as PH_3 and P_2O_5 . It generally is found in nature as salts of orthophosphoric (H_3PO_4) and pyrophosphoric ($\text{H}_4\text{P}_2\text{O}_7$) acids. With iron it forms a phosphide, Fe_3P . It is completely reduced in the blast furnace, hence practically all the phosphorus occurring in the raw materials is found in the pig iron. In steel it is an undesirable impurity, except for special applications. It can be partially removed as part of the slag by oxidation followed by neutralization of the oxide with lime.

Calcium and Magnesium

Calcium

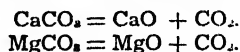
Atomic Number, 20
Atomic Weight, 40.08
Valence, II
Specific Gravity: 1.54

Magnesium

Atomic Number, 12
Atomic Weight, 24.32
Valence, II
Specific Gravity: 1.74

While these two elements belong to different groups, they are very similar so far as the manufacture of iron and steel is concerned. With few exceptions one may be substituted for the other without great inconvenience. Their oxides are the more important bases of dry chemistry. Both elements occur as insoluble carbonates; limestone, marble, chalk, and marl are forms of calcium carbonate (CaCO_3). Magnesite is magnesium carbonate (MgCO_3). When heated, both of these compounds de-

compose into their oxides and carbon dioxide, thus:



CaO represents quicklime, and MgO, magnesia.

These elements also occur together as a double salt of carbonic acid, named calcium magnesium carbonate, $\text{CaMg}(\text{CO}_3)_2$, commonly called dolomite, which gives calcium magnesium oxide, $\text{CaO}\cdot\text{MgO}$, when calcined.



Both calcium and magnesium may be obtained electrolytically from their chloride. Magnesium is also obtained electrolytically from its oxide or by chemical reduction, involving a reducing agent (carbon in one process and ferrosilicon in another), followed by distillation and recovery of the magnesium metal from the distillate.

Lime (CaO), magnesia (MgO), and the double oxide, $\text{CaO}\cdot\text{MgO}$, are all very refractory, but on account of its tendency to slake in air, CaO is not used as such. Practically, MgO is the best basic refractory known, and calcined dolomite is the best available substitute. The oxides are reduced with difficulty, and on account of their cheapness constitute the principal basic fluxes. As MgO is the leading basic refractory, CaO is the leading basic flux. It combines with both silica and phosphoric acid to form readily fusible slags, which have a lower density than iron and consequently float upon the surface of the metallic bath.

Aluminum

Atomic Number, 13. Atomic Weight, 26.98. Valence, III. Specific Gravity: 2.7.

This element in combined form is very widely distributed, occurring as one of the constituents of feldspar, granite, mica, cryolite, and all clays. It is reduced from the oxide, Al_2O_3 , by an electrolytic process, in which a molten bath of cryolite acts as a solvent. It has a strong affinity for oxygen, violently reducing iron oxide, and on this account it is added to steel as a deoxidizing agent. In its compounds aluminum displays decidedly basic properties, forming salts with all the common acids except carbonic acid. In wet reactions, it forms neither a carbonate nor a sulphide, but it readily combines with sulphur at high temperatures. Aluminum hydroxide, $\text{Al}(\text{OH})_3$, however, acts like both an acid and a base. When this compound is heated, it loses water and forms alumina (Al_2O_3), which is found in varying amounts in all the raw materials that enter into the metallurgy of iron. In the blast furnace alumina is never reduced. Its presence, however, has a marked influence on the slag, affecting its fluidity and fusion temperature, important considerations in blast furnace practice. In its purer states alumina is a good refractory, but the cost of production makes its extensive use as such prohibitive. An extended list of other metals are added to the metal aluminum to produce stronger alloys; e.g., Cu, Mg, Si, etc.

Chromium

Atomic Number, 24. Atomic Weight, 52.01. Valences: II, III, VI. Specific Gravity: 6.9.

This element is somewhat rare. In small deposits it is found as chromite, $\text{Cr}_2\text{O}_3\cdot\text{FeO}$, the best neutral refractory known, which, in its purer states, melts at about 3450°F . Chromium is obtained by reduction of its oxides with aluminum or silicon. It is white, crystalline, very hard and does not tarnish. Chromium forms both acid and basic compounds. It is very important in the manufacture of the stainless and other alloy or special steels.

Its chief physical effect is one of facilitating quench hardening, hence it is employed to increase the hardenability of projectiles, armor plate, automotive steels, and tool steels. Chromium is added to alloy steel in the form of ferrochromium, an iron-chromium alloy produced from ores.

Manganese

Atomic Number, 25. Atomic Weight, 54.93. Valences: II, III, IV, VI, VII. Specific Gravity: 7.4.

This element occurs in nature as MnO_2 , its deposits being somewhat limited in the United States. In very small amounts it is widely distributed, and is found in most of the raw materials used in the manufacture of pig iron. About 75 per cent of this manganese is reduced in the blast furnace, and becomes a constituent of all pig iron. It is readily oxidized in the purifying processes and, except for relatively low amounts, the manganese found in steel is added to it in the process of manufacture. Its effect in steel is beneficial because it offsets to some extent the harmful effects of oxygen and sulphur. Higher percentages, 10 per cent to 15 per cent, are employed to produce the special steel known as Hadfield manganese steel.

Iron

Atomic Number, 26. Atomic Weight, 55.85. Valences: II, III. Specific Gravity: 7.864.

This most important metal occurs combined with other elements. It is found in varying amounts in nearly all earthy matter, such as clays, soils, sands, etc. In deposits, it is found as the sulphide (FeS_2), as silicates, as a constituent of chromite, as the carbonate ($FeCO_3$) and as the oxides Fe_2O_3 and Fe_3O_4 . The compound last named is magnetic. Pure iron is grayish-white in color and relatively soft when compared with steel of high carbon content. It is malleable, ductile, and magnetic. The melting point of the pure metal is about $2800^\circ F$, but the usual commercial forms melt at lower temperatures. The presence of certain elements, notably carbon, silicon, phosphorus, or sulphur, in the metal lowers the melting point markedly. Iron forms two series of compounds, the ferrous and the ferric. The more important ferrous compounds are FeO , $Fe(OH)_2$, $FeCl_2$, $FeSO_4 \cdot 7H_2O$; corresponding ferric compounds are Fe_2O_3 , $Fe(OH)_3$, $FeCl_3$, $Fe_2(SO_4)_3$. Most of these compounds are of the highest commercial importance, and many will receive a more thorough treatment later in this book.

Chapter 2 (Continued)

SOME FUNDAMENTAL PRINCIPLES OF CHEMISTRY AND PHYSICS

PART 3 — FUNDAMENTALS OF PHYSICS

SECTION 1

INTRODUCTORY

Physics is a science that deals with matter, energy, and radiation. Most of the discussion on the structure and properties of matter, usually included among the subjects that are related to Physics, already has been presented in Part 1 of this chapter, and need not be repeated here. In the following summary of physical

theories and facts, only those phases of physical knowledge which are essential to an understanding of the manufacture and use of steel are considered; subjects such as mechanics, sound, light and nuclear physics are not discussed since they are not directly involved in an understanding of later chapters of this book.

SECTION 2

ENERGY, FORCE, WORK AND POWER

Nature of Energy—Physics and chemistry are concerned with more than matter. The human senses are aware of a second fundamental factor in nature, called energy, which is not a material thing and is difficult to define satisfactorily. Energy is that which gives a body the ability to move against a resistance, that is, to do work.

Potential vs. Kinetic Energy—In general, there are two classes of energy; namely, kinetic energy and potential energy. Kinetic energy is the energy possessed by a body by virtue of its motion; for example, a moving train, a rotating wheel, and a bullet in flight all have kinetic energy. Potential energy, sometimes called stored energy, is the energy possessed by a body because of its position, or due to its chemical or physical state; for example a weight suspended in the air, an electric storage battery, a charged electric condenser, and a piece of coal all possess potential energy.

It is a matter of common observation that potential energy can be changed into kinetic energy, and vice versa; thus, a pendulum on the down-swing converts potential energy into kinetic energy, which is then reconverted into potential energy on the up-swing. Furthermore, it is true that the amount of potential energy expended is equal to the amount of kinetic energy generated, except for a certain small amount which is lost due to friction in the air and at the supports. It is this loss which prevents the realization of "perpetual motion."

Forms of Potential Energy—There are several forms of potential energy; namely, mechanical, electrical, thermal, and chemical. Furthermore, there are at least two kinds of stored mechanical energy, that is gravitational potential energy, or energy of position, and elastic

potential energy, or energy of distortion; for example, a weight suspended in air possesses gravitational energy because it will do work when it falls, while a wound clock spring possesses elastic energy, since it can do work when it unwinds. Electrical energy may be stored in a condenser, thermal or heat energy in a radiator, and chemical energy is stored in a storage battery or in a piece of coal.

Conversion of Energy—Casual observation reveals that one form of energy can be transformed to another form. An electrical power plant affords an illustration of a complicated transformation of energy. Part of the chemical energy in coal is converted into heat energy which in turn is converted into mechanical energy in steam; the energy of the steam is transferred to the moving machinery of a turbine and thence to a generator which converts this mechanical energy into electrical energy. This electrical energy is transmitted over wires to some distant point where it is reconverted into heat, light, or mechanical energy.

Conservation of Energy—Early experiments on the conversion of mechanical energy into other forms of energy led to the general conclusion that the original amount of energy can always be completely accounted for after such a change. In other words, energy can neither be created nor destroyed, it can only be transformed. This is known as the **Law of Conservation of Energy**. As pointed out previously, recent work in nuclear physics has demonstrated that matter can be converted into energy; that is, in a sense energy can be created; however, in all common chemical and physical phenomena and processes, this exception need not be taken into consideration, since the principle still holds.

Force—It has already been stated, in connection with the force of attraction of gravity exerted by the earth on all objects, that force is perceptible and is associated with muscular effort in lifting or pushing an object. This simple conception is quite satisfactory for engineering purposes with one reservation; namely, that force must be considered a vector quantity; that is, both its magnitude and its direction must be given in order to completely define it. It is clear that when a body is lifted vertically the force of gravity which must be overcome is acting in a direction opposite the motion of the body, so that the applied force required is equal to the weight of the body. However, in pushing the same body along the ground, the force required is not equal to the weight of the body, because the force of gravity acts vertically, whereas the applied force is directed horizontally; actually, only enough force must be exerted in the direction of motion of the body to overcome the friction between the body and the ground.

Work—The concept of work has developed naturally in connection with the lifting of a weight through a definite distance. Thus, if an object of a certain weight is lifted a certain distance, and then is lifted twice as high, it is intuitively felt that twice as much work has been done in the second case. Similarly, if the weight of the object is doubled, the work expended in lifting the object a certain distance is also doubled. In other words, the work done in moving a body is equal to the product of the applied force and the distance moved in the direction of the force. Expressed mathematically:

$$W = wd$$

where "w" is the weight of the body expressed in pounds or newtons, "d" is the distance the body is moved in the direction of the force expressed in feet or meters, and "W" is the work which is measured in foot-pounds or newton-meters. One newton-meter is also called a joule. Sometimes another metric unit of force, the dyne (equal to 1/100,000 of a newton), is used; in this case, the unit of work is the dyne-centimeter, which is more commonly called the erg. One joule is equal to ten million ergs. It is apparent that energy and work are meas-

ured in the same units, since expended energy is called work, or energy may be thought of as virtual work.

Power—The time required to do a given amount of work is usually an important factor. For example, a person may not be able to carry a heavy object to the top of a steep hill in a few minutes, but this may be possible within a few hours; the difference is merely a matter of time. The rate of doing work, or of expending energy, is called power. In mathematical terms:

$$P = \frac{E}{t}$$

where "E" is energy or work expressed in foot-pounds or joules, "t" is the time in seconds, and "P" is the power measured in foot-pounds per second, or joules per second. One joule per second is also called a watt. Before the machine age, the basis for measurement of power was the rate at which an average horse could do work; this standard rate was termed a horsepower. Obviously, such a standard could not be made precise and was not practical to maintain, so eventually it was agreed that

1 horsepower = 550 foot-pounds per second
or 1 horsepower = 746 watts (or joules per second)

Transmission of Energy—Another very important and useful characteristic of energy, aside from its capacity for conversion from one form to another, is that it can be transmitted from place to place over rather long distances. Thus, electrical energy can be generated at a central station located near a coal supply or waterfall, and this energy can be conveniently transmitted to distant towns and farms, or steel mills or other industrial plants where it can be converted into useful mechanical work. Other forms of energy also can be transmitted, although usually over shorter distances; for example, heat from a furnace can be transferred to radiators in the different rooms of a building through the medium of water, steam, or air, and mechanical energy can be transmitted by means of wheels, axles, levers, and rods.

SECTION 3

HEAT

Nature of Heat—It was originally believed that combustion was concerned with a mysterious stuff called phlogiston which was somehow released by all substances when they burned, or that water over a fire became hot because a subtle weightless fluid called caloric was added to it. Thus, the early tendency was to consider heat as an intangible kind of matter, rather than a form of energy. Later experiments on the temperatures developed from the friction of moving mechanical parts showed that the temperature rise was proportional to the amount of mechanical work done; this led to the conclusion that mechanical energy could be converted into heat and that heat must be a form of energy. Further work demonstrated that heat itself is some sort of internal motion and that it is in fact the motion of the atoms and molecules which constitute matter.

Temperature—The human body is aware of temperature through the sensations of hot and cold; that is, temperature refers to hotness, and the term usually implies the quantitative measurement of hotness by some more accurate means than the human senses. The temperature of a region of matter is a quantity that depends on the average kinetic energy of the atoms or molecules

in that region. The atoms or molecules of any substance are in a rapid and continual motion, and all types of atoms and molecules at the same temperature have the same kinetic energy. When heat energy is added to a body in a fixed state, the average speed of its molecules is increased, and its temperature rises proportionally to the increase in average molecular kinetic energy. The temperature is truly zero only when all molecular motion ceases, and this point is called absolute zero.

The simplest temperature scale, and the most convenient for scientific purposes, is one which has its zero-point based at absolute zero. Such a scale exists, and is known as the absolute or Kelvin scale; however, the temperature scales in everyday use are based arbitrarily on the freezing and boiling points of water; the temperature interval between these two fixed points being subdivided into small increments called degrees. On the Centigrade scale, used in ordinary engineering and scientific work, the zero point is based at the freezing point of water and the boiling point of water is assigned the temperature of 100 degrees, so that there are one hundred equal subdivisions or degrees between these two points. The absolute or Kelvin scale uses the same

size divisions or degrees as the Centigrade scale, although its zero point is based at absolute zero, rather than at the freezing point of water. In the Fahrenheit scale, which is in common use in all English-speaking countries, 32° F is based at the freezing point of water and 212° F is based at the boiling point of water; thus, there are 180 equal subdivisions or degrees between these two fixed points. The relationship between these three temperature scales is shown below:

Fixed Point	Absolute	Centigrade	Fahrenheit
Absolute zero	0°	-273°	-459°
Freezing point of water.	273°	0°	32°
Boiling point of water..	373°	100°	212°

The conversion of temperatures from the Centigrade to Fahrenheit scales, or vice versa, can be accomplished by substitution in the following expressions:

$$F = (9/5 C) + 32$$

$$C = 5/9 (F - 32)$$

where "F" is the temperature in degrees on the Fahrenheit scale and "C" is the temperature in degrees on the Centigrade scale.

Thermometry and Pyrometry—In principle, any physical phenomenon that varies systematically with temperature can be used to measure temperature. Most solids, liquids, and gases expand with increasing temperature, and the amount of expansion is commonly used as an indicator of temperature. Mercury and alcohol expand considerably with moderate increase in temperature, and hence are used in glass-enclosed thermometers for measuring ordinary temperatures. Mercury solidifies at -38.2° F (-39° C) and boils at 675° F (357° C), whereas alcohol freezes at -178.6° F (-117° C) and boils at 173.3° F (78.5° C), so that thermometers employing these substances can only be used within these ranges. Other properties of matter are used to measure temperature. Of course, all of these methods require calibration against fixed reference points in the temperature scale, such as the freezing and boiling points of water, or the melting points of pure metals.

Two solids, such as copper and iron, having quite different thermal expansion coefficients can serve as a thermometer if joined together in the form of a "bimetallic strip." On heating, the copper expands more than the iron, and the strip bends one way; on cooling, the copper contracts more than the iron, and the strip bends the other way. The position of the end of the strip can be used to indicate the temperature.

When two wires of dissimilar metals, such as copper and iron, are joined at one end, an electromotive force (e.m.f.) is generated, which results in a difference in electrical potential (voltage) between the two free ends of the wires, if there is a difference in temperature between the junction and the free ends, and this difference in voltage is proportional to the difference in temperature; thus by using an instrument for measuring the voltage (voltmeter or galvanometer), the temperature can be determined. This device is called a thermocouple.

The resistance of an electrical conductor, such as a metal wire, varies enough with temperature for the measurement of resistance to provide an excellent precision method of determining temperature over a wide range. This principle is utilized in resistance thermometers.

Still another type of thermometer employs optical principles for temperature measurement. The color of a hot, glowing body varies with the temperature. Comparison of the color of this object with the color of a

controlled hot filament of an electric light permits very high temperatures to be measured. Such a device is called an optical pyrometer.

The intensity of the light given off by a hot, glowing body also changes with the temperature, and thus by measuring the light intensity, or comparing it to certain standard intensities, the temperature can be determined. This principle is utilized in radiation pyrometers.

Measurement of Heat—In the conversion of mechanical energy to heat energy through friction, the temperature rise in the heated object is nearly proportional to the amount of mechanical energy transformed; furthermore, it is found that to raise the temperature of 10 kilograms of water 1° C requires ten times as much heat energy as is needed to raise the temperature of 1 kilogram of water 1° C. Therefore, the change in heat energy of a body, that does not change state in this temperature range, is proportional both to the temperature change and to the mass of the body. From this, it is clear that temperature and amount of heat energy are two entirely separate and distinct quantities, although they are related.

Heat energy could be measured in units of foot-pounds or joules, but because the methods of heat measurement are not ordinarily related to mechanical effects, a special and more convenient unit of heat energy is used. In the metric system, changes in heat energy are expressed in kilocalories. A kilocalorie (1000 calories) is the amount of heat energy that must be added to one kilogram of water to raise its temperature 1° C. In the English system, the unit of thermal energy is the British Thermal Unit (abbreviated Btu), which is the amount of heat required to increase the temperature of one pound of water 1° F. One Btu is equivalent to about 0.252 kilocalories.

The heat required by various substances to raise their temperatures the same amount differs considerably. In order to express differences of this sort quantitatively, the concept of specific heat is used. In the metric system, specific heat is defined as the number of kilocalories per kilogram of a certain substance that is required to change the temperature of the substance by 1° C. This provides a simple way to express the change in heat energy of any substance when its temperature is changed:

$$\text{Change in heat energy} = MS(T_2 - T_1)$$

where "M" is the mass in kilograms, "S" is the specific heat of the substance in kilocalories per kilogram per degree Centigrade, and "T₂" and "T₁" are the final and initial temperatures, respectively, in degrees Centigrade.

Heat vs. Changes in State of Matter—As previously mentioned, when a substance undergoes a change of state it also suffers a change in heat energy. Thus, when a solid substance melts, it absorbs a certain definite amount of heat energy (per unit mass of material) from its surroundings, and when it resolidifies, it releases this same quantity of heat; this is known as the heat of fusion of the substance. Similarly, when a liquid substance boils or condenses, it absorbs or evolves a certain definite quantity of heat, called the heat of vaporization of the substance.

Another kind of change in state of matter, that involves a change in heat energy, is a change in the crystal structure of a solid substance. For example, solid iron undergoes a rearrangement of its atoms at about 1650° F (900° C), whereby its crystal structure changes from a body-centered (alpha) to a face-centered (gamma) cubic lattice of atoms; this change is accompanied by a change in heat energy, amounting to about 4 calories per gram, which is called the heat of transformation.

Such changes in crystal structure are called **polymorphic** or **allotropic changes**. As in the cases of melting and boiling, the temperatures at which these changes occur are affected by pressure, but at ordinary atmospheric pressure they are constant and characteristic of the substance and the particular transformation.

Conversion of Heat into Mechanical and Electrical Energy—The conversion of mechanical energy into heat energy has already been discussed and it has been pointed out that the ratio of the amount of heat generated to the mechanical energy expended is a constant called the **mechanical equivalent of heat**. In the metric system, 1 calorie = 4.186 joules, and in the English system 1 Btu = 778 foot-pounds. Industrially, the change of mechanical energy to heat energy is not nearly as important as the reverse change of heat energy to mechanical energy, for the burning of coal or oil to produce heat and, in turn, mechanical energy in steam turbines and internal combustion engines, has been the world's greatest source of useful work. Heat can also be converted directly into electrical energy, as in thermocouples, or electrical energy can be converted directly into heat energy, as for example, in electric heaters and resistance furnaces, wherein a heavy electric current is passed through a poor electrical conductor. The electrical equivalent of heat is given by the expression:

$$\text{Change in heat energy} = 0.24 IRt$$

where "I" is the electrical current measured in amperes, "R" is the resistance measured in ohms, and "t" is the time in seconds. The ampere and ohm are defined in Section 7 of this part of Chapter 2.

Effects of Temperature on Physical Properties—Heat is capable of producing marked changes in the physical properties of matter. Adding heat to a gas or liquid increases its molecular energy, and thus increases its pressure or volume. Solids expand when heated and the amount of expansion (thermal coefficient of expansion) differs for every substance and, frequently, it is different for the same substance at different temperatures. Changes in heat energy, and thus in the temperature or state of a substance, are usually accompanied by changes in its mechanical properties (modulus of elasticity, ductility, hardness, and tensile strength), in its electrical properties (conductivity), and in its magnetic properties (permeability).

Heat Flow—Heat energy may be transferred from place to place by three different mechanisms—conduction, convection, and radiation.

a. Conduction—When heat flows through a body by the transference of the kinetic energy of individual atoms or molecules without mixing, it is said to flow by **conduction**. When one end of a bar is heated, the atoms or molecules are given greater vibratory motion, that is, greater kinetic energy; in some way, probably by collisions, these atoms or molecules share this increased energy with their neighbors, which in turn pass this energy along to those beyond, and so on. Thus, the energy of agitation (heat) is propagated along the bar.

The measurement of heat flow by conduction is most easily understood in connection with solids, since in this case, convection is not present, as it is in liquids and gases, to complicate the situation. The basic law of heat transfer can be written in the form of a rate equation:

$$\text{Rate} = \frac{\text{Driving Force}}{\text{Resistance}}$$

in which the driving force is the temperature difference across a solid body, since it is apparent that heat can flow only when there is an inequality of temperature. This law, known as **Fourier's Law**, states that the rate of

heat flow through a body is proportional to the temperature drop, to the area, and inversely proportional to the thickness of the body. The mathematical expression of Fourier's Law is:

$$\frac{Q}{t} = \frac{kA(T_2 - T_1)}{L}$$

where "Q" is the amount of heat energy transferred in time "t," "A" is the area of the body perpendicular to the direction of heat flow, and "(T₂ - T₁)" is the temperature difference between opposite sides or ends of the body, "L" is the thickness of the body in the direction of heat flow, and "k" is a constant which is defined by this equation and is called the **thermal conductivity** of the particular substance constituting the body. If "Q" is measured in Btu, "t" in hours, "A" in square feet, "T₂" and "T₁" in degrees Fahrenheit, and "L" in feet, then k is expressed as Btu per hour per square foot per degree Fahrenheit per foot.

When each of the terms in the equation defining "k" is equal to one, "k" is termed the **coefficient of thermal conductivity**. The numerical value of the coefficient of thermal conductivity depends upon the substance of which the body is made and upon its average temperature. The thermal conductivities of liquids and gases are very small in comparison with those of most solids. For example, at 212° F the thermal conductivity of silver is 240 Btu per hour per square foot per degree Fahrenheit per foot, that of building brick is about 0.8, that of water about 0.35, and of air 0.017. In general, the variation of conductivity with temperature is linear, that is,

$$K = a + bT$$

where "a" and "b" are constants and "T" is the temperature.

It is of interest to note that thermal conductivity and electrical conductivity are closely related. In general, the best electric conductors have the highest thermal conductivity. The relationship between thermal and electrical conductivities is given by the **Wiedemann-Franz Law**:

$$\frac{K}{\sigma} = aT$$

where "K" is the thermal conductivity, "σ" is the electrical conductivity, "T" is the absolute temperature, and "a" is a proportionality constant. This law holds fairly well for most metals within restricted temperature ranges.

b. Convection—When heat flows by actual mixing or physical turbulence, the mechanism is known as **convection**. Convection is restricted to heat flow in liquids and gases. It is practically impossible for heat to flow through fluids by pure conduction without some convection, because of the eddies set up by the changes of density in the fluid with temperature. For this reason, the terms conduction and convection are frequently used together, although in many instances, the predominant mechanism is convection; for example, the heating of a room by a steam radiator is largely a matter of convection. The heating of a room by a hot-air furnace is an example of almost pure convection.

In the transfer of heat from a solid body to a liquid, a thin, comparatively stagnant film of liquid exists at the surface of the solid, which is of great importance in determining the rate of heat transfer. Heat is transferred through this thin, stationary, fluid film more by conduction than by convection, and since the thermal conductivities of fluids are very small, the rate of heat transfer is very slow. In many processes, such as the

quenching of steel, the heat transfer through the surface film of liquid on the steel is the limiting factor which controls the rate of cooling of the steel. In general, the greater the turbulence in the liquid and the lower its viscosity, the thinner the stagnant film and the higher the rate of heat transfer.

c. Radiation—The transfer of energy through space by means of electromagnetic waves is termed radiation. If radiation travels through empty space, it is not transformed to heat or any other form of energy, nor is it diverted from its path. If, however, matter appears in its path, the radiation will be absorbed or reflected. It is only the absorbed energy that appears as heat, and this transformation is quantitative. Thus, a polished opaque surface or mirror will reflect most of the radiation striking it, whereas a black or matte surface will absorb most of the radiation received by it and will transform such absorbed energy quantitatively into heat.

In practice, radiation is an important method of transferring heat, which is especially true in the making and treating of steel where furnace temperatures are relatively high. The fundamental law covering heat transfer by radiation, known as Stefan's Law, states that the total energy which is radiated by a body per second is proportional to its surface area and to the fourth power of its absolute temperature. In mathematical terms,

$$E = cAT^4$$

where "A" is the area of the radiating surface in square feet, "T" is the temperature of the body in degrees absolute, "c" is a constant equal to 1.72×10^{-8} , and "E" is the total energy given off per second in Btu.

Actually all bodies at temperatures above absolute zero radiate energy according to Stefan's Law. Thus, in the case of a small body of temperature " T_1 " and area "A" completely surrounded by a hotter body of temperature " T_2 ," the net amount of heat transferred per second from the hotter body to the colder body is the algebraic sum of the radiation from the two bodies; thus, Stefan's Law may be written:

$$e = cA (T_1^4 - T_2^4)$$

where "e" is the net amount of heat transferred from the hotter body to the colder per second, " T_1 " is the absolute temperature of the hotter body, and " T_2 " is the absolute temperature of the colder body. This assumes, however, that all of the heat radiated by either body falls on the other body and is absorbed by it. In practice, most objects do not fulfill this last assumption. Instead of absorbing all of the radiant energy that falls on them, they absorb only a part of it and reflect the rest. Only an ideal black body absorbs all, and reflects none of the radiant energy falling on it. Most objects with a matte black surface approach the theoretical black body closely enough for practical purposes; polished metal bodies and white bodies, on the other hand, deviate widely from the ideal black body. It is true, however, that inside a closed space, where the temperature is uniform throughout, all objects, no matter what their substance, color, or surface condition, behave as black bodies. This greatly simplifies radiation calculations and measurements in many important cases; thus the interior of a furnace may be considered to be uniform to the extent that all objects in it may be assumed to be black.

SECTION 4

MAGNETISM AND MAGNETIC FIELDS

Natural Magnets—Certain rocks have long been known to possess the ability to attract iron and are called natural magnets. These rocks are now recognized as magnetite (Fe_3O_4), an iron ore. The term magnetism refers to this ability to exert forces on a piece of iron from a distance. When elongated pieces of such rocks are freely suspended from threads, or floated by some suitable means on water, they orient themselves with their longest axis in a north-south direction; in this form, they represent the earliest crude form of compass, which led to their being called lodestones or "leading stones." The north-seeking end or pole of the piece is called the north pole of the magnet and the south-seeking end is called the south pole. The attractive force of any magnet is greatest at its ends, or poles.

Artificial Magnets—Lodestones today are merely scientific curiosities, since it is possible to make artificial magnets that are far stronger than natural ones, and in shapes most convenient for their intended use. Familiar forms are the bar magnet and the horseshoe magnet. One method for making an artificial magnet is to stroke, with another magnet, the material to be magnetized. In another method, the material to be magnetized is placed inside a coil of wire carrying a current of electricity. The most modern method consists of suspending the piece to be magnetized on a conductor carrying a heavy electric current. The last two methods are possible because, as will be learned later, there are magnetic effects caused by electric currents.

If the material being magnetized is relatively soft (pure) iron, only a light electric current is necessary to magnetize the piece, but the result is only temporary. If

hardened steel is used for the magnet, it requires a heavier electric current and a longer time to produce a magnet, but once magnetized the steel will retain its magnetic properties for a long time. Magnets made from materials which lose their magnetism readily are called temporary magnets while magnets made from steel and other materials that retain their magnetism are called permanent magnets. Permanent magnets are commonly made from steel. Tungsten steel and alloys containing cobalt are used extensively. An alloy of aluminum, nickel and cobalt also is used where very strong magnets are to be made.

Magnetic Fields—Two magnet poles in the same vicinity exert forces on each other. They repel each other if they are like poles (both north poles or both south poles), attract if they are unlike. In the region about each of the poles a magnetic field of force exists; that is, any magnetic pole there would experience a force proportional to the field strength. The variation in the direction and strength of the magnetic field about a bar magnet can be demonstrated by placing a sheet of paper over a magnet and sprinkling iron filings over it. The iron filings will assume a pattern similar to that shown in Figure 2-11, the number and direction of the lines depending on the strength of the magnet. The chains of filings show the direction of the lines of force in the magnetic field. What apparently is empty space, definitely contains energy, for the particles of iron making up the filings experience forces which move them into the indicated pattern. It should be remembered that this same field completely surrounds the magnet, and that the pattern shown by the filings in this experiment

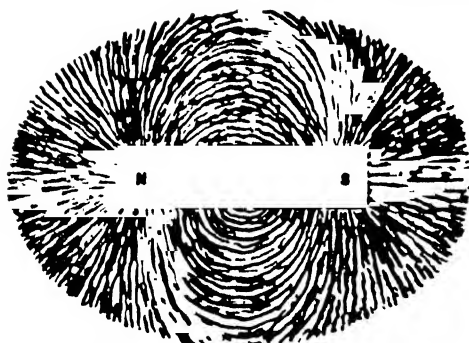


FIG. 2-11. Pattern assumed by iron filings scattered on a sheet of paper or glass placed over a bar magnet.

represents only a cross section of the magnetic field. The lines of force are assumed to emanate from the north pole of the magnet and to reenter it at the south pole.

Earth's Magnetic Field—The earth itself is a huge natural magnet, having magnetic poles like any other magnet, and possessing a magnetic field. A lodestone, or a magnetized needle (compass) always aligns itself in a north-south direction because it is acted upon by the earth's magnetic field of force. The lodestone or needle aligns itself with the lines of force of that field. It is interesting to note that the magnetic pole of the earth which attracts the north (or north-seeking) pole of a magnet is actually a magnetic south pole, and vice versa. It also may be noted that the magnetic poles of the earth do not coincide with the geographic poles, and that the earth's magnetic field is neither uniform nor constant.

Magnetic Field Strength—Magnetic field strength or magnetizing force is measured in terms of force acting on a unit magnetic pole. A unit pole is defined in such a way that two unit poles placed one centimeter apart in air repel each other with a force of one dyne. A dyne is defined as that force which will give a mass of one gram an acceleration of one centimeter per second per second. The unit of field strength, the gauss, is a force of one dyne exerted by the field on a unit magnetic pole. The field strength is also commonly expressed in terms of the number of lines of force intersecting a unit area perpendicular to the direction of the field; the lines of force are numerically equal to the field strength in gauss. Magnetic flux refers to the total number of lines of force passing through a section of a magnetic field, and it is equal to the product of the area and the field strength. The unit of magnetic flux, the maxwell, is the flux through an area of one square centimeter perpendicular to a field having a strength of one gauss.

Magnetic Induction—Materials capable of being attracted by a magnet or of acquiring magnetism from a magnet are called magnetic materials. Magnetism may be imparted to magnetic materials by bringing them into direct contact with a magnet or by bringing them into a magnetic field. The degree of magnetism induced will depend upon the strength of the magnetic field. For example, a small piece of soft iron in contact with one of the poles of a bar magnet, so long as it is in contact, will itself act as a magnet and attract other pieces of iron. Similarly, a piece of soft iron bar adjacent to, but not in contact with, one of the poles of a bar magnet will be magnetized. In this latter case, if one end of the soft iron bar is adjacent to the south pole of the bar magnet, that end of the soft iron piece will become a

north pole and its opposite end will become a south pole. This can be demonstrated by the action of the end of the soft iron piece farthest from the bar magnet upon a magnetic needle or compass (Figure 2-12). In neither

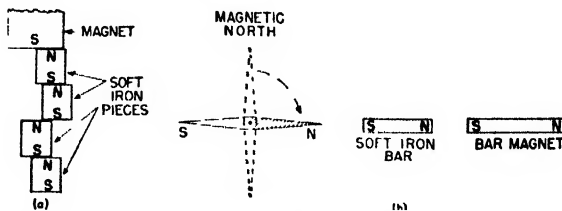


FIG. 2-12. Magnetism imparted to pieces of soft iron (a) by contact with magnets and (b) by bringing a piece of soft iron into a magnetic field.

of the above cases is the soft iron itself a magnet except when in contact with, or influenced by, the magnetic field of the bar magnet. The kind of magnetism exhibited by the soft iron is said to be **induced** and the action is called **magnetic induction**. The magnetism induced in soft iron is temporary, but permanent magnetism may be induced in hardened steel if the bar magnet used is strong enough, depending upon the degree of hardness of the steel.

Among the commoner magnetic materials are soft (pure) iron as referred to above, nickel, cobalt, manganese and chromium. Pure iron, however, possesses the best magnetic properties. Magnetic materials offer very little resistance to the passage of lines of force. A magnetic field will pass through a nearby piece of iron rather than take a shorter route through the air because of the lower magnetic resistance of iron as compared to air (Figure 2-13). Non-magnetic materials, e.g., cop-

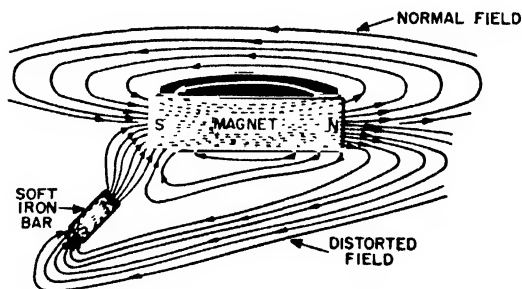


FIG. 2-13. Field distortion caused by soft iron bar.

per and aluminum are not attracted to magnets nor will they attract a magnet. Some alloy steels are non-magnetic, and some elements lose their magnetic properties when heated.

Permeability—Any substance capable of acquiring magnetic properties when under the influence of a magnetic field is said to be **permeable**. Permeability is measured by the number of lines of force which can be concentrated within a unit area of the material. All materials have a **saturation point** beyond which it is impossible to concentrate any more lines of force within a unit area. Soft iron is an example of a highly permeable material.

Retentivity—The degree to which the material resists becoming magnetized or losing its magnetism is the **retentivity** of the material. Soft iron is highly permeable,

but it loses its magnetism almost immediately after being removed from the magnetic field, and thus has a low degree of retentivity. Hard steel, however, can be made into a permanent magnet by being acted upon by a magnetic field and thus has a high degree of retentivity.

Reluctance—The opposition to passage of magnetic lines of force characteristic of a given material is termed its reluctance. The reluctance of a magnetic circuit is directly proportional to the length of the circuit, and inversely proportional to its section. Ordinarily, magnetic materials exhibit less reluctance than non-magnetic materials. Also, non-magnetic materials have practically the same reluctance at all flux densities, while the reluctance increases with the flux density (number of lines of magnetic force) for magnetic materials.

Theory of Magnetism—Actually, all materials are magnetic, although most of them are only about one-millionth as magnetic as iron. The ultimate source of magnetism is not well established, but evidence points to the molecules and atoms constituting matter. According to an older theory, the atoms are considered to be tiny magnets themselves, some much stronger than others. In ordinary materials, the atomic magnets are

oriented at random, and their effects normally cancel each other. In the case of iron, each atomic magnet, when placed in a magnetic field, tries to line up with the field; if more of these atomic magnets line up in one direction than in any other, their effects are additive, so that the whole piece becomes a powerful magnet. However, if this iron magnet is heated to a dull red color, the alignment of the atomic magnets is so disturbed by thermal vibration that the piece of iron loses its magnetism. At present, this older belief has been modified to regard discrete portions of a body undergoing magnetization as becoming successively magnetized. These portions involve groups of atoms instead of single atoms, and are referred to as **domains**. The effect of temperature in destroying natural magnetism in a body is conceived to be the result of the sudden breaking-up of the arrangement of atoms forming domains when some definite temperature (the Curie point) is reached.

According to either theory, every magnetic substance has a saturation point beyond which the magnetic flux density cannot be increased because, once all the elementary atomic magnets in a substance are lined up, or all possible domains are established, not much further increase in magnetism can be obtained.

SECTION 5

ELECTROSTATICS

Nature of Electrostatic Charges—According to the concept of the structure of matter outlined in Part 1 of this chapter, the nucleus of any atom is composed of protons and neutrons and carries a positive charge that is exactly balanced by the sum of the unit negative charges of the electrons in the electronic system surrounding the nucleus. All matter, therefore, normally is uncharged or neutral. However, if certain substances are rubbed together, some of the electrons can be "scraped off" the surface of the one and added to the surface of the other. Thus, if a glass rod is rubbed with silk, electrons are transferred from the surface of the glass to that of the silk. The silk, with an excess of electrons, then carries a negative charge, and the glass which is left with less than its normal quota of electrons acquires a positive charge. In a similar manner, if a hard-rubber rod is rubbed with fur, the rod acquires a negative charge while the fur becomes positively charged. These charges, designated as **electrostatic charges**, constitute what is known as **static electricity**, so called because the charges are at rest. Charged bodies are said to be **electrified**. Bodies having a positive charge, due to a deficiency of electrons, are said to be at a higher potential than those carrying a negative charge resulting from an excess of electrons.

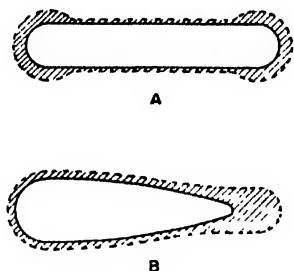


FIG. 2-14. Distribution of charge of conductors.

An electrostatic charge is distributed over the surface of an electrified body. The density of the charge at any part of the surface depends upon the shape of the body. For example, a charge on an electrified sphere is distributed uniformly over the surface of the sphere. In the case of an egg-shaped body, the greater part of the charge is concentrated at the sharper end of the egg; on a uniform rod, there is a concentration of charge at each end. (Figure 2-14).

Electrostatic charges can be transferred from one body to another by contact. A pith ball, if touched by a positively charged glass rod, will have some of the positive charge imparted to it. If a second pith ball, positively charged in the same manner, is brought near the first positively charged ball, the two will mutually repel each other. If both balls are charged negatively by contact with a negatively-charged rubber rod, they again will repel each other. However, if one ball is given a positive charge and the other is given a negative charge, the two balls will mutually attract each other. Thus, like electrostatic charges repel and unlike charges attract each other. The electrical force of attraction or repulsion acting between two charged bodies is directly proportional to the product of the charges on the two bodies and inversely proportional to the square of the distance between them.

Induced Electrostatic Charges—It is possible for an electrically charged body to induce another electric charge in nearby bodies; that is, an apparently neutral body can be charged by bringing a charged body near to it, but not in contact with it. The induced charge is explained as the effect of attraction at a distance between, say, a positive charge, and the electrons in the initially uncharged body. The electrons are displaced slightly toward the inducing positive charge, making a negative charge at that end of the originally neutral body and leaving an equal positive charge at the opposite end; when the body inducing the charge is removed, the second body returns to its formerly neutral condition. This type of induced charge is called a **tempo-**

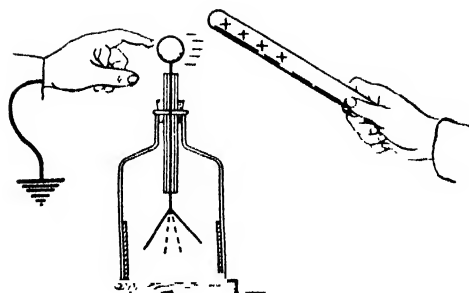


FIG. 2-15. Induced electrostatic charge.

rary induced electrostatic charge. As shown in Figure 2-15, if a positively charged rod is brought near an insulated metal sphere, a negative charge will be induced on the side of the sphere adjacent to the rod, and a positive charge will be induced on the opposite side. If the sphere is "grounded" by touching it on the positively charged side, electrons will flow from the positively charged rod holds the negative charge (electrons) bound on the opposite side. If first the finger, and then the rod, is removed, the sphere will be left with an excess of electrons, or a negative charge. This constitutes a **permanent induced electrostatic charge**.

Electric Discharge—If two bodies are in proximity to each other and one is electrified with a positive charge so as to be at a considerably higher potential than the other, it will, as described above, induce an opposite charge on the other body. It is possible to continue to increase the potential to a point where electrons will rush from the negatively-charged body across the intervening distance to the positively-charged body to balance the positive charge, thus causing a spark. The nature of the substance separating the two bodies determines how high a potential difference is necessary before transference of electrons can occur. Lightning results from a sudden discharge of a heavy charge of static electricity when, for example, a negatively charged cloud passes over the earth, inducing an op-

posite charge on objects below, such as trees and buildings. When these charges build up to a potential large enough to overcome the insulating resistance of the air, the air becomes a conductor and the excess electrons on the cloud rush to neutralize the positive charge below, the discharge spark being the familiar lightning flash. If the cloud is positively charged the flash will be upward, or if the induced charge is on another cloud the flash will occur between the clouds.

Static Electricity and Electricity in Motion—The electric current consists of electric charges in motion, as opposed to static electricity which consists of electric charges at rest. Some materials readily permit passage of the electric current and are known as **conductors**. Most metals are good conductors of electricity because of the "unbound" or "free" electrons characteristic of the outer shells of electrons of metallic atoms. Any electric potential will easily set in motion these electrons and a current will flow.

In other materials, the outer shells of electrons are bound more closely to the atom, and only very high potentials can cause transference of electrons from atom to atom to cause a flow of current. These latter materials are called **insulators**. Actually there is no sharp distinction between insulators and conductors, as materials merely differ enormously in conductivity.

When electric charges are added to or taken from insulators, or when electric charges move through conductors, experiment shows that it is generally the negative electrons that actually move. As soon as any movement of charges occurs, static electricity is immediately transformed into an **electric current** (electricity in motion).

It has been conventionally assumed that electric current flows from points of higher potential (+) to ones of lower potential (-). From the foregoing discussion, it is seen that electrical manifestations are due actually to flow of electrons from one point to another. The direction of flow of electrons, it is obvious, is opposite to the direction in which current has always been assumed to flow. It is necessary in discussing the flow of electric current to realize that although the current is assumed to flow from positive (+) to negative (-) it is the flow or drift of electrons from minus (-) to plus (+) that permits electrical forces to do useful work.

SECTION 6

CHEMICAL GENERATION OF ELECTRIC CURRENT

Since, in order to produce an electric current or flow of electricity, it is necessary to make electric charges move, a force called an **electromotive force** must be exerted on them. In electricity, this force is supplied by a **difference in potential, or voltage**; thus, there is a difference of potential between the two ends of a wire when there are forces present to make the charges in a wire flow as an electric current.

The simplest method of producing a difference in potential, and thus generating an electric current, is to use an **electric cell or storage battery** which converts chemical energy into electrical energy. Such a cell may consist of two dissimilar metals, such as zinc and copper, in the form of rods called **electrodes**, immersed in a bath of dilute sulphuric acid, which is called the **electrolyte**. The action of such a cell is as follows: zinc tends to go into solution as positively charged zinc atoms, or ions. Each positive zinc ion that goes into solution leaves a

corresponding excess negative charge on the zinc electrode. At the copper electrode, positive hydrogen ions from the dilute sulphuric acid (H_2SO_4) take electrons from the copper to form neutral hydrogen atoms. This leaves the copper electrode with a deficiency of electrons, that is, an excess positive charge. In this way, a potential difference is established between the two electrodes. However, there will be no flow of electric current until an external conducting wire is connected between the two electrodes; then, negative electrons flow through this conductor to the positive electrode or point of higher potential. Since the current generated by a battery always flows in only one direction (conventionally considered to be from the positive to the negative electrode), it is called **direct current**. Direct current can also be produced mechanically, as will be described later. As mentioned earlier, flow of electric current is conceived to take place from a point of higher

potential (+) to a point of lower potential (—). The conventional direction of current flow, therefore, is opposite to the direction of actual flow of electrons.

Only relatively small amounts of electricity can be

generated chemically, so devices called **generators** for producing electrical energy from mechanical energy have been developed; these are discussed in Section 12 under "Mechanical Generation of Electric Current."

SECTION 7

POTENTIAL, CURRENT, RESISTANCE

Potential—In order to move a charged particle (e.g., an electron) against an electrical force, work must be done on it. Consequently, potential energy must be added to it. If the particle is made to move from one point to another, the amount of potential energy (number of joules) added to the particle, per coulomb of charge on it, is called the potential difference between the two points. The common unit of potential difference is the volt. The volt is practically defined as that potential difference which will cause a current of one ampere to flow through a resistance of one ohm. The ampere and ohm are defined below. When an electrical force exists which is capable of moving a free positive charge from one point to another, the first point is said to be at a higher potential than the other, or it is said to be positive with respect to the other that is referred to as negative.

Current—When two ends of a wire conductor are connected to two points at different potentials, such as the terminals of a battery, an electric current flows in the wire. The mechanism of this current flow is roughly as follows. Initially, the conductor has equal numbers of positive and negative charges in its atoms. The atoms are packed together so closely that their electronic systems overlap to some extent, and it is comparatively easy for the outer electrons (referred to as "free" or "unbound" electrons) to pass from one atom to another when a small force is applied to them. The battery sets up a difference of potential between the ends of the wire and thus provides a force that causes the negative electrons in the wire to migrate from atom to atom toward the point of higher potential. This electron flow toward the positive terminal provides the actual means whereby electrical forces can be made to do useful work. It has been mentioned earlier that current flow has always been assumed as being from the point of higher potential (+) to the point of lower potential (—) or opposite to the electron flow. Materials differ considerably in the ease with which electrons can be made to migrate from atom to atom, or in other words, they differ considerably in electrical conductivity.

The magnitude of the current depends simply on the rate of flow of electrons through the conductor; that is, if each electron carries an electric charge, "e," the electric current, or rate of flow of electric charge, is determined by the net number of electrons per second carried through a section of the wire. If "n" electrons move through the wire each second, the charge moved per second will be "ne." The unit of current, the ampere, is legally defined as that unvarying current which when passed through a solution of silver nitrate (AgNO_3) in water in accordance with standard specifications will deposit silver at the rate of 0.001118 grams per second. It follows that, if the current is steady, the total charge passing through the section in a given time is merely the product of the current and the time. A coulomb is the quantity of electricity transferred when a current of one ampere flows for one second.

It is important to realize that steady electric currents ordinarily exist only in complete circuits, for such cur-

rents are simply the continuous circulation of electric charges; that is, the source of voltage circulates electric charges around through conductors like a pump circulates water. Furthermore, a steady electric current, like flow of water in a pipe, has the same value at all locations in a simple, unbranched circuit.

Resistance—The electric current in a conductor would be limited only by the number of available electrons, which is practically infinite, if it were not for the effect of electric resistance. This effect comes into play because the moving electrons do not pass freely and unhindered through the conductor, but collide with the atoms of the conductor and bounce off or trade places with electrons in the atoms. The resulting increase in the agitation of the atoms means an increase in heat energy. The greater the electric current, the greater the number of collisions, and the more electric energy is changed to heat energy. The electric current cannot exceed the value at which all electric energy becomes heat energy.

If the resistance of a conductor is increased, say by changing the material, the potential difference between the ends of the conductor needed to produce a given current will increase. The relationship between potential difference, current, and resistance is given by Ohm's Law, described in Section 8 immediately following. The unit of resistance is the ohm, which is the resistance of a column of mercury 106.3 centimeters in length, of uniform cross section and weighing 14.452 grams at 0° C.

The resistance of a wire is inversely proportional to the electrical conductivity of its material. As previously pointed out, materials differ considerably in conductivity. The resistance of a conductor depends on its size and shape as well as the nature of its material; thus, the larger the cross-sectional area of the conductor, the smaller its resistance, and the longer the conductor, the greater its resistance. In fact, the resistance of a conductor of uniform size and shape is directly proportional to the length and inversely proportional to the area; expressed mathematically,

$$R = R_s \frac{L}{A}$$

where "R" is the resistance of the conductor expressed in ohms, "L" is the length in meters or feet, "A" is the area in square millimeters or circular mils (area of a circle 1/1000 inch in diameter), and "R_s" is a proportionality factor or constant, called the **specific resistance** or **resistivity** of the material expressed in ohms per meter per square millimeter, or in ohms per circular mil per foot. Specific resistance or resistivity is merely the inverse of conductivity.

The resistance of metallic conductors usually increases as the temperature rises; for example, the resistance of pure metals increases by about 4 per cent for an increase of 18° F (10° C), whereas alloys usually have a smaller rate of change. In some alloys, the resistance remains practically constant over a wide range of temperature. The resistance of liquids, such as those

used in storage batteries, decreases as the temperature rises. If the temperature coefficient of resistance (α) of a material, and the resistance (R_0) of the material at

0°C , are known, the resistance at any other temperature (t) can be calculated from the equation, $R_t = R_0(1 + \alpha t)$.

SECTION 8

FLOW OF DIRECT CURRENT IN CONDUCTORS

Ohm's Law—The relationship between the difference of potential between two ends of a conductor and the current flowing in the conductor is given by Ohm's Law, which states that the current is directly proportional to the difference of potential or voltage; expressed mathematically,

$$V = RI$$

where "V" is the difference of potential in volts, "I" is the current in amperes, and "R" is the proportionality constant, the resistance, expressed in ohms.

Ohm's Law applies only to a uniform or steady current, that is, to a current of constant magnitude; it does not always hold for alternating current, such as is commonly used in lighting and power circuits (see section 13 of this part of Chapter 2: "Characteristics of Alternating Current"). Furthermore, the relationship is strictly true for metallic conductors only when there is no change in the temperature.

Resistance in Series—The preceding discussion of resistance has been limited to the flow of current through a very simple circuit, that is, through a conductor of uniform resistance connected to a storage battery. In practical circuits, it is customary to connect several different resistances together in the same complex circuits; in this case, the resistances may be coupled end to end in series, as shown in Figure 2-16, or in parallel, as

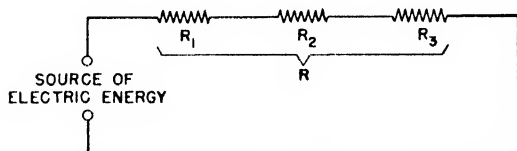


FIG. 2-16. Resistances in series. The combined resistance (R) is equal to the sum of individual resistances ($R_1 + R_2 + R_3$).

shown in Figure 2-17. When the resistances are connected in series, the total or combined resistance is equal to the sum of the individual resistances. Thus in Figure 2-16,

$$R = R_1 + R_2 + R_3$$

where "R" is the total resistance, and " R_1 ," " R_2 ," and " R_3 " are the individual resistances.

Resistances in Parallel—When the resistances are connected in parallel, as in Figure 2-17, the combined

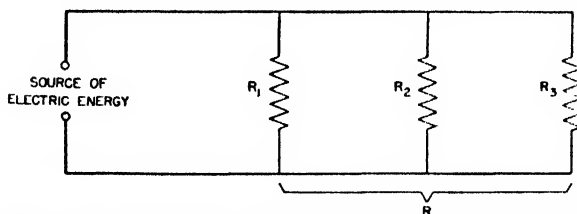


FIG. 2-17. Resistances in parallel. The combined resistance (R) is less than any one of the individual resistances (R_1 , R_2 , or R_3).

resistance is less than the resistance of any one of them. This is apparent from the hydraulic analogue of this electric circuit, shown in Figure 2-18, consisting of

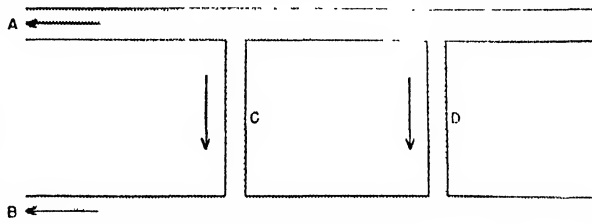


FIG. 2-18. Hydraulic analogue of electric resistance in parallel. Water mains (A and B) connected by small pipes (C and D) in parallel.

two large water mains connected by a single smaller pipe C. When the pipe D is added, another path is provided for the flow of water, hence more water will flow through the circuit and the total resistance to flow is decreased. The expression for computing the total resistance (R) of three resistances (R_1 , R_2 , R_3) in parallel is

$$\frac{1}{R} = \frac{1}{R_1} + \frac{1}{R_2} + \frac{1}{R_3}$$

Law of Divided Circuits—When a current in a complex circuit divides, part flowing in one branch and part in another, the branch possessing the smaller resistance will conduct the larger part of the current. In fact, the ratio of currents in the two branches is inversely proportional to the ratio of the resistances of the branches. Thus in Figure 2-19,

$$\frac{I_1}{I_2} = \frac{R_2}{R_1}$$

where " I_1 " and " R_1 " represent the current and the resistance of one branch, and " I_2 " and " R_2 " the current and the resistance of the other branch.

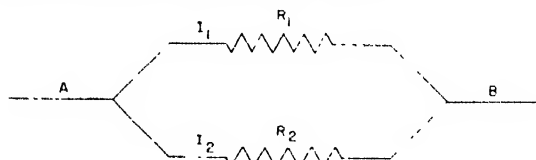


FIG. 2-19. A divided circuit. The ratio of the currents (I_1 and I_2) in the two branches is inversely proportional to the ratio of the resistances (R_1 and R_2) of the branches.

Frequently it is desirable to have only part of a current flow through an instrument. For example, in order to measure a large current with a galvanometer or ammeter, which is not capable of carrying the whole current, a conductor may be connected across the terminals of the instrument so that the current will divide, part going through the instrument and part through the conductor of known, calibrated resistance. A conductor used in this way is called a **shunt**.

SECTION 9

ELECTROMAGNETISM

Magnetic Fields and Electric Currents—There is an important connection between magnetism and electricity which is that a magnetic field is produced around a wire carrying an electric current. This can be demonstrated by the deflection of a compass needle (magnet) when it is brought near a vertical wire carrying a current, or by the positions assumed by iron filings in concentric circles around the wire on a plane (sheet of paper) perpendicular to the wire, as indicated in Figure 2--20. The lines of force are circular and lie in planes

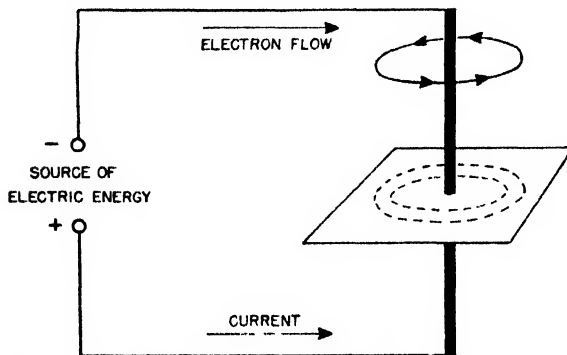


FIG. 2--20. Magnetic field about a wire carrying a current, shown by iron filings on a sheet of paper.

perpendicular to the current. The direction of the magnetic field with respect to the direction of the current is given by the **right-hand rule**; that is, if the wire is grasped in the right hand, so that the thumb points in the direction of the current, the fingers will point around the wire in the direction of the magnetic field. The strength of the field is directly proportional to the current in the wire.

This same rule applies even though the wire may be bent into a loop (Figure 2--21). When several loops of

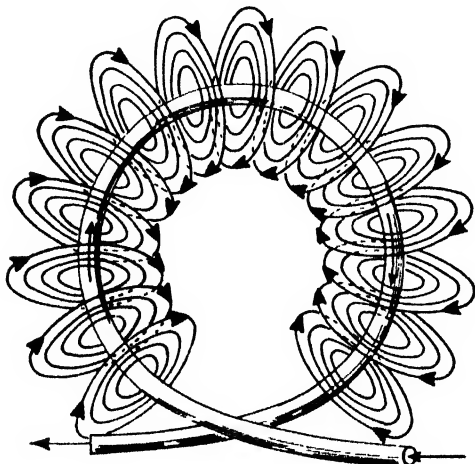


FIG. 2--21. Magnetic field about a single loop of wire carrying a current.

wire are wound to form a coil, most of the magnetic lines will pass through the entire coil (Figure 2--22). Such a loose coil of wire is known as an **air-core solenoid**. If now, an iron core is placed within the coil to

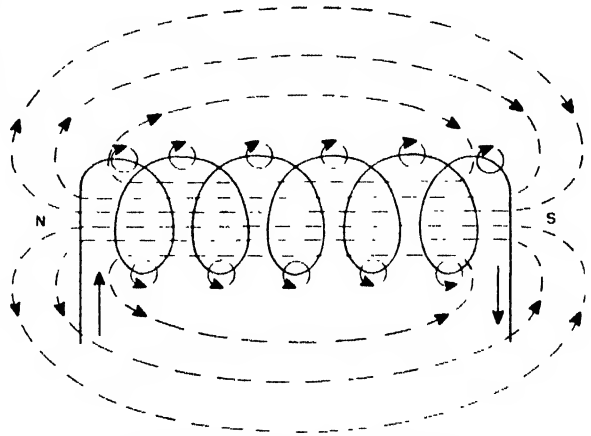


FIG. 2--22. Magnetic field about a solenoid carrying a current

form an **iron-core solenoid**, the strength of the magnetic field about the coil is increased tremendously. The reason for this is that when the iron core becomes magnetized, the field of force of the magnet lines up with the field of the coil to produce an intense magnetic field. This arrangement consisting of an iron core in a helical wire coil (Figure 2--23) is commonly called an **electromagnet**.

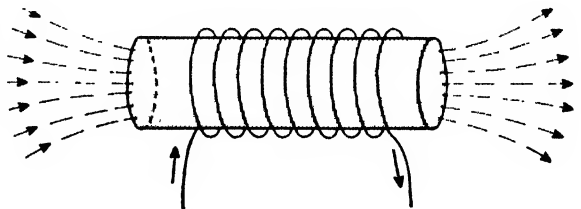


FIG. 2--23. Solenoid with iron core, showing concentration of magnetic flux.

The strength of the magnetic field within a large solenoid or coil with an air core is given by:

$$H = \frac{4\pi NI}{10}$$

where "H" is the field strength in gauss, "N" is the number of turns of wire in the coil per centimeter of coil length, and "I" is the current in the wire in amperes. The field strength of an electromagnet can be calculated from this expression: in practice, the strength of electromagnets is rated in terms of **ampere-turns** (product of the current in amperes times the number of turns of wire in the coil— IN). When the coil contains an iron

core, which is more permeable than air, the same number of ampere turns in the coil will yield a much greater field strength as mentioned above. The exact amount of the increase in field strength will depend on the permeability of the iron core—the higher the permeability of the iron core the greater the increase in field strength over that of the coil alone.

The permeability of iron may vary considerably, depending upon its purity, microstructure, and state of internal stress. In fact, the relationship between the magnetizing force of the wire coil alone, and the strength of the induced magnetic field when the iron core is inserted is given by:

$$B = \mu H$$

where "H" is the magnetizing force of the coil alone in gauss, "B" is the strength of the induced magnetic field when the core is inserted, and " μ " is the permeability of the iron. If the value for the field strength "H" given above is inserted in this expression:

$$B = \frac{4 \pi \mu NI}{10}$$

The value of " μ " is not constant but varies with the magnetizing force "H," as shown by the typical curve of Figure 2-24. The field strength of the electromagnet

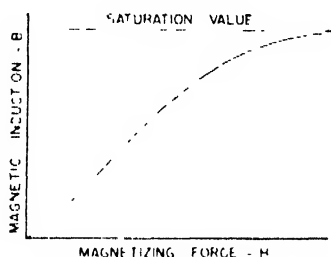


FIG. 2-24. A typical magnetization curve for iron.

can be increased still further by providing a more complete iron circuit so that the magnetic lines of force travel largely through the iron, that is, by extending the iron core outside of the coil until its two ends nearly meet in the shape of a C.

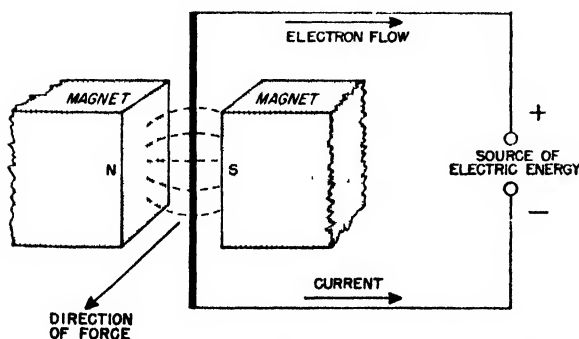


FIG. 2-25. Force on a wire suspended in a strong magnetic field.

Another equally important aspect of the relationship between electric currents and magnetic fields is that a wire carrying a current experiences a mechanical force when placed in a magnetic field. If a wire carrying a current is loosely suspended in a strong magnetic field, the wire will tend to move in the direction indicated in Figure 2-25; if the direction of the current in the wire is reversed by changing the battery connections, the wire will move in the opposite direction. It will be noted that, in either case, the force on the wire is perpendicular to both the magnetic field and to the current. This mutual perpendicularity between the three directions of force, field, and current is characteristic of relationships between mechanical, magnetic, and electrical phenomena. The magnitude of the force depends on the current, the strength of the magnetic field, and on the length of the wire in the field; that is, the force is proportional to the current times the magnetic field strength times the length of the wire. Since the force, current, and length can be readily measured, it is convenient to define the unit of magnetic field strength in these terms.

If a single loop of wire is suspended between the poles of a magnet, and a current is produced in the wire loop, as in Figure 2-26, the direction of the current in the

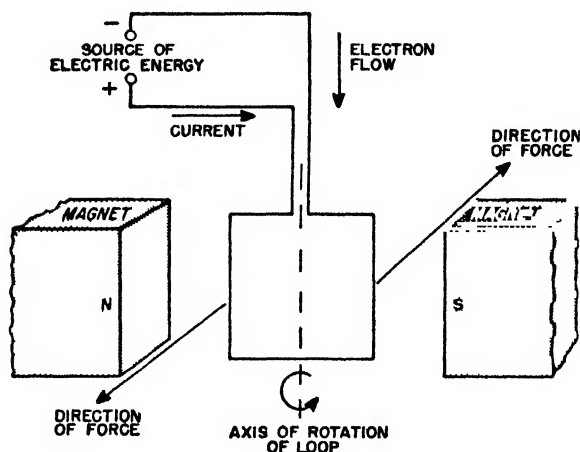


FIG. 2-26. Loop of wire carrying a current and suspended in a strong magnetic field. Forces tend to cause rotation.

left-hand part of the loop is such that the force is directed toward the reader, whereas in the right-hand part, the current is opposite, so the force is directed away from the reader. These equal and opposite forces tend to make the wire loop rotate. The effectiveness of the forces tending to produce this twist is called the torque, and in this case it is the product of one force and the distance between the two forces. The tendency of a loop of wire carrying a current to rotate in a magnetic field is the basis for electric motors which transform electric energy into mechanical energy (see Section 14). Conversely, if a wire carrying no current originally is moved through a magnetic field so as to cut the lines of force of the field, an electric current will flow through the wire; this principle is employed in generators converting mechanical energy into electrical energy, as discussed in Section 12.

SECTION 10

ELECTRIC ENERGY AND POWER

Energy—Electric energy is determined in much the same way as is mechanical potential energy. The potential energy that is changed to other forms of energy when a body of weight "W" and mass "m" falls through a distance "h" is given by the expression:

Potential Energy = $Wh = mgh$ (foot-pounds or joules)

Similarly, the electric energy that is changed to other forms when an electric charge "Q" falls through a difference of potential "V" is given by:

Electric Energy = QV (joules)

In practice, the charge "Q" is not measured directly, instead the current "I," which is the rate of flow of electric charge, is measured, and the total charge passing through a section of the circuit in time "t" is obtained from the relation

$$Q = It$$

which is true if "I" does not change. If "V" is also constant, the quantity "It" can be substituted for "Q" in the expression for electric energy:

Electric Energy = VIt (joules)

Inasmuch as electric energy is readily transformed into heat energy in resistances, it is often desirable to know the amount of heat energy generated when a steady current "I" flows in a resistance "R" for time "t." The electric energy transformed into heat energy is:

$$VIt = I^2Rt \text{ (joules)}$$

since the potential difference "V" between the two ends of a resistance is equal to "IR" volts.

It has been pointed out earlier that the mechanical equivalent of heat is given by

$$\begin{aligned} 4,186 \text{ joules} &= 1 \text{ calorie} \\ \text{or } 4,186 \text{ joules} &= 1 \text{ kilocalorie} \end{aligned}$$

Therefore, the above equation for electric energy transformed into heat energy may also be written:

$$\begin{aligned} VIt &= \frac{RI^2t}{4,186} \text{ (kilocalories)} \\ &= 0.00024 RI^2t \text{ (kilocalories)} \end{aligned}$$

Power—Frequently it is desirable to know how much energy is being generated or consumed per unit time rather than the total energy, in other words, to know the power of a generator or motor.

From the above discussion on electric energy, it is apparent that, if "V" and "I" are constant,

$$\text{Power} = \frac{\text{energy}}{\text{time}} = VI \text{ (joules per second or watts)}$$

The unit of electric power is the watt; since this unit is very small compared to the power generated and used today, a larger or multiple of this unit, called the kilowatt, is generally used (1 kilowatt = 1000 watts).

There are several ways of expressing electric power in terms of resistance, current, and potential difference. Since $V = IR$, then

$$\text{Power} = VI = I^2R = \frac{V^2}{R} \text{ (watts)}$$

In a simple circuit, the power dissipated in a resistance can be determined by measuring the two quantities in any one of these expressions.

SECTION 11

ELECTROMAGNETIC INDUCTION

If a coil of wire is connected to a galvanometer and a magnet is pushed into the coil, the galvanometer pointer momentarily deflects, indicating a current, but it returns quickly to zero when the magnet is held stationary within the coil. When the magnet is pulled out of the coil, the pointer deflects in the opposite direction, but again becomes zero almost immediately. It may be concluded then that only moving or changing magnetic fields induce electric currents, as no current is generated in the coil when the magnet is at rest. Evidently, either the magnet or the coil can be moved in order to produce a current. Thus mechanical energy can be converted into electric energy by moving a coil and a magnet with respect to each other.

A current can also be produced by means of two coils, instead of a magnet and a coil (Figure 2-27). When a current is produced in the first coil, a magnetic field quickly radiates from this coil and its lines of force spread out and cross the wires in the second coil nearby. This electromagnetic disturbance induces a momentary current in the second coil. The pointer of a galvanometer connected to the second coil deflects momentarily and then returns quickly to zero. However, no current flows in the second coil as long as the current in the first coil is steady. When the current in the first coil is stopped, by opening a switch, again there is a momentary current

in the second coil, but in the opposite direction. Evidently, current is induced only when the current in the first coil is changing; that is, when the magnetic field produced by it is changing. By reversing the direction of the applied current, or by turning one of the coils end for end, the direction of the induced current indicated by the galvanometer is reversed. Thus, the second coil must be subjected to a changing magnetic field if cur-

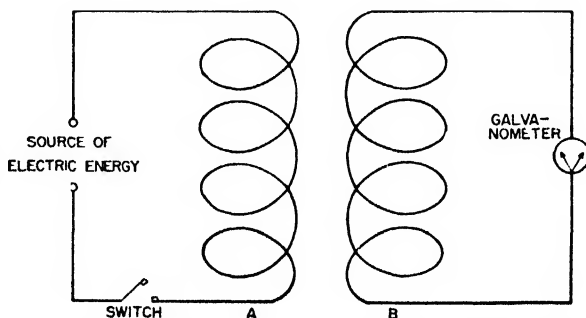


FIG. 2-27. Changing current in coil A induces a current in coil B.

rents are to be induced in it. The changing field, therefore, can be produced either by varying the current in a nearby stationary coil, or by actually moving the first and second coils with respect to each other. If this experiment is repeated with the two coils placed on a single iron core, the induced current in the second coil is greatly increased, because iron greatly increases the magnetic field produced by the current in the first coil and tends to lead the lines of force from the first coil through the second.

Self-Induction—Whenever the current in a solenoid changes, the lines of flux cut every turn within the solenoid itself and induce within the coil a voltage opposing or counter to the change. This phenomenon is known as **self-induction**. As the current decreases, there is a forward voltage or electromotive force (e.m.f.); as it increases there is a back electromotive force. Thus, the induced electromotive force tends to keep the current

unchanged. This is analogous to the action of mass in mechanics and may be thought of as electrical inertia. The measure of self-induction is the **coefficient of self-induction** or **self-inductance** (L).

Laws of Electromagnetic Induction—From experiments such as those described above, the following laws pertaining to electromagnetic induction can be deduced:

1. Any change in the number of lines of force passing through a closed conducting circuit induces a current in that circuit.
2. The direction of the induced current is always such that its magnetic field opposes the motion which produces it (**Lenz's Law**).
3. The electromotive force of the induced current is directly proportional to the rate at which the number of lines of force are increased or decreased, or to the rate at which the lines of force are cut.

SECTION 12

MECHANICAL GENERATION OF ELECTRIC CURRENT

Alternating-Current Generator—If a single loop of wire is placed in a strong magnetic field and is rotated by means of a torque applied to it with a crank or pulley (Figure 2—28), the wire on the left moves in one direction through the magnetic field, and the wire on the right moves in the opposite direction. Therefore, the voltages induced in both wires set up a current in the same direction in the loop. However, as the loop is rotated a half revolution further, the two wires ex-

change directions through the magnetic field, so that the induced voltages are opposite to the previous ones, and the current flows in the opposite direction through the loop.

The change in total voltage during one complete rotation of the loop is represented graphically in Figure 2—28. The voltage increases from zero at position "P" to a maximum in one direction, called positive, as the wires pass one pair of poles and then decreases to zero a half revolution from position "P." As the loop is rotated further and the wires pass the opposite pair of poles, the voltage reverses and becomes a maximum in the opposite direction, called negative; it then decreases to zero again as the loop returns to position "P." This type of device produces an alternating positive and negative voltage, resulting in a current first in one direction and then in the opposite direction, and therefore is called an **alternating-current (AC) generator**. The number of complete reversals or cycles of voltage per second is termed the **frequency** of oscillation of the voltage. This type of generator, in a more complex form, is in many ways more useful than one which produces a voltage, and therefore a current, always in the same direction.

Direct-Current Generator—In order to obtain voltage and current from a generator in one direction only, a commutator-brush switching arrangement must be introduced. Such an arrangement illustrated in Figure 2—29 reverses the connections to the armature at the proper times, so that the current will always be in the same direction. By providing several armature coils of many turns of wire on an iron core and connecting them to commutator segments, and by using electromagnets to supply the fixed magnetic field, a relatively steady **direct current** can be produced when the armature is rotated.

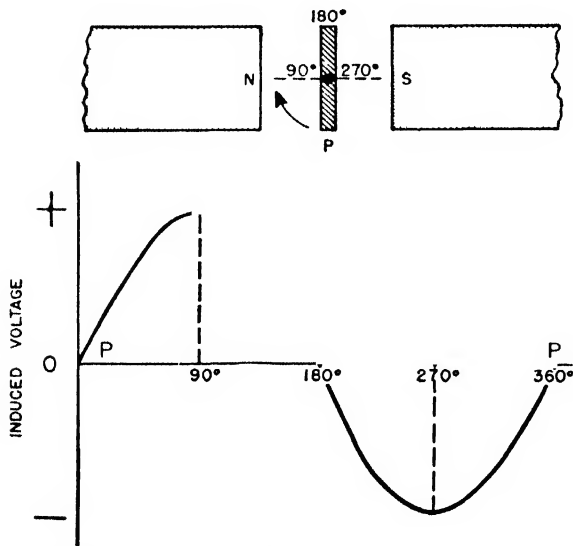


FIG. 2—28. Change in voltage induced in a wire loop during one complete rotation in a magnetic field.

SECTION 13

CHARACTERISTICS OF ALTERNATING CURRENT

The flow of alternating current is different in many respects from that of direct current—many new phenomena appear. For example, when two alternating cur-

rents are fed into a conductor, the resulting current may be anything from zero to the sum of the two, depending on their relative phases. Furthermore, the voltage may

be out of phase with the current, so that although there may be both current and voltage, there may be no power. Also, alternating current can flow where direct current cannot and vice versa.

Frequency—In the discussion of the simple two-pole alternating generator, it was pointed out that the generated voltage fluctuates from zero to a maximum in the positive direction, back to zero, then to a maximum in the negative direction, and finally back to zero, during one revolution of the armature. This complete reversal of voltage, consisting of two alternations, is called a cycle. If the armature rotates 60 revolutions every second, the frequency of the resulting voltage is 60 cycles per second, the most commonly used frequency. In practice, to avoid the necessity of running a generator at such a high speed, several pairs of poles are arranged in a circle around the armature, so that north and south poles alternate in position. If there are 10 poles (5 pairs), 5 cycles will be generated per revolution, and the generator can be run at 12 revolutions per second, or 720 revolutions per minute, in order to produce 60-cycle current.

It should be pointed out that the alternating current fluctuates in a systematic way (sinusoidal wave form) just as does the voltage. When alternating current flows through a simple resistance, the voltage and current waves are in step (phase) and are represented together as a single wave; that is, the voltage is greatest at the same time that the current is greatest. Under certain conditions, however, the voltage may not be in phase with the current, and this situation is represented by two waves, one displaced along the time axis with respect to the other.

Polyphase Current—In the simple alternating-current generator possessing a loop or two coils in the armature, described previously, much of the space in the magnetic field is wasted. This space can be utilized by placing additional pairs of coils in the armature; thus, if a second pair of coils is added, a second independent current supply is provided. The voltage generated in these coils is just 90 degrees out of phase with that of the first pair of coils, because the magnetic flux through this second pair of coils is a maximum when that through the other pair is zero; this is a **two-phase generator**. Frequently three such sets of independent circuits are employed in the armature to produce **three-phase current**.

Effective Values of Alternating Current and Voltage—It is not possible to use the average value of the alternating current sine wave as a measure of current, be-

cause this value is zero. However, the energy delivered, or heat developed, is equal to I^2R , the value of I^2 , though varying, is always positive. Therefore, the square root of the average value of I^2 may be taken as the effective value (I_e) of alternating current (root mean square current). The effective value of AC voltage (V_e) has the same meaning. The root mean square values are approximately seven-tenths of the maximum values for the currents and voltages. It is these effective values which are read directly on AC ammeters and voltmeters.

Impedance—When alternating current flows through a simple resistance, the relation between current and voltage is given by Ohm's Law,

$$I_e = \frac{V_e}{R}$$

but, in general, Ohm's Law cannot be used for alternating current. Alternating current differs from direct current in at least two important respects; that is, alternating current is impeded by the self-induction of a coil, whereas direct current flows unhindered, and alternating current effectively passes through a capacitor according to an appropriate impedance, whereas direct current cannot flow at all. For alternating current, it is necessary, therefore, to introduce a new property of a circuit, called impedance (Z), to replace the term resistance in Ohm's Law:

$$I_e = \frac{V_e}{Z}$$

The impedance (Z) includes a resistance term and a reactance term. This latter term is frequency dependent and is actually composed of two terms, the inductive reactance and the capacitive reactance. The mathematical expression for the impedance Z is:

$$Z = \sqrt{R^2 + \left(2\pi fL - \frac{1}{2\pi fC}\right)^2}$$

Power Factor—Any self-induction in a circuit causes a difference in phase between the voltage and the current, so that the current either forges ahead or lags behind the voltage, and the two do not reach their maxima and minima together. Consequently, at any instant the product of the voltage and current, which represents the energy available for use, is less than the total energy supplied to the circuit. The ratio, expressed in per cent, between the useful voltage that is available for overcoming resistance of conductors, to produce heat, or to run a motor, and the actual or measured voltage, is called the **power factor**.

SECTION 14

PRINCIPLES OF MOTORS

In rotating the armature of a generator, mechanical energy is converted into electrical energy, whereas in operating a motor, electric energy is converted into mechanical energy, so that the motor and the generator perform operations which are the converse of one another. The amount of electric energy generated is never quite equal to the mechanical energy input and vice versa, because a certain small amount of mechanical energy is dissipated as mechanical friction and heat in the generator or motor. In many respects, a generator is like a water pump in that it forces electrons to circulate through wires to electrical machines or appliances, where they do useful work and then return to the generator.

Direct-Current Motor—In order to make a loop or coil of wire, suspended between the poles of a magnet, rotate continuously when a direct current is passed through the wire, a switching arrangement, called a **commutator**, must be provided, so that the current through the coil is reversed in direction just as the coil aligns itself with the fixed field magnetic poles. The coil is then magnetized in such a way that it must rotate still further to line up in the opposite position. This process is repeated indefinitely, producing a continuous rotation of the armature coil. The coil is always reversed at just the proper time to carry on the rotation of the coil. The commutator consists of two copper segments insulated from each other and connected to the ends of the coil.

The current is carried to the commutator segments by carbon contacts called "brushes" which are forced by springs to rub against successive commutator segments. This simple motor arrangement is shown diagrammatically in Figure 2-29.

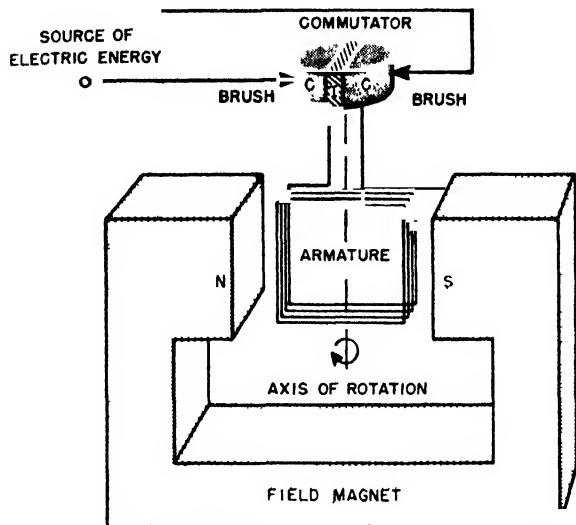


FIG. 2-29. Principle of direct-current motor. The commutator consists of two metal segments (C) separated by an insulator (I).

In practice, the armature consists of several coils of many turns of wire wound on an iron core to provide large magnetic forces. Also, the fixed magnetic field is usually produced by sending some of the current from the source of electric energy into field coils wound on iron cores, instead of by means of permanent magnets. When the field coils are placed in parallel with the armature coils, as in Figure 2-30, the motor is called a **shunt motor**. This is the usual connection for large direct-current motors. When the field coils are connected in series with the armature coils, the motor is called a **series motor** (Figure 2-31). Most small motors are of the latter type and can be used on either direct current or alternating current. Another type of direct-current motor combines some of the advantages of the series and shunt windings; this is the **compound-wound motor** (Figure 2-32).

Alternating-Current Motor—Inasmuch as alternating current is more conveniently transmitted than direct current, motors have been developed especially for operation by power from this source. Alternating-current motors are simpler in construction than direct-current motors. Thus, in Figure 2-33 a closed wire loop is shown suspended between the poles of an electromagnet which has alternating current applied to it. The current in the loop is induced by an increase in the magnetic field, caused by an increase in voltage and current in the electromagnet windings. If the magnetic field then starts to decrease, as the coil rotates 90 degrees past the first position, the direction of the induced current in the loop will reverse, and the mechanical forces will be such that the rotation continues indefinitely in the same direction. The number of revolutions per second will be equal to the alternating-current frequency in cycles per second. It is evident, therefore, that the main differences between an alternating-current motor and a direct-current motor are: the

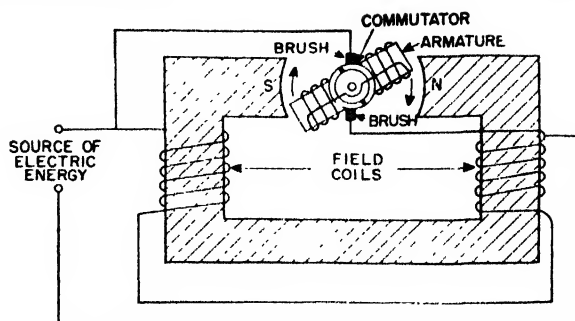


FIG. 2-30. Simple shunt-wound direct-current motor.

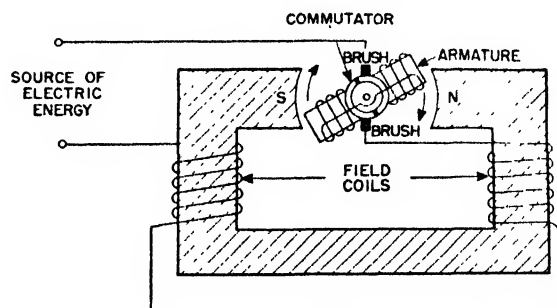


FIG. 2-31. Simple series-wound direct-current motor.

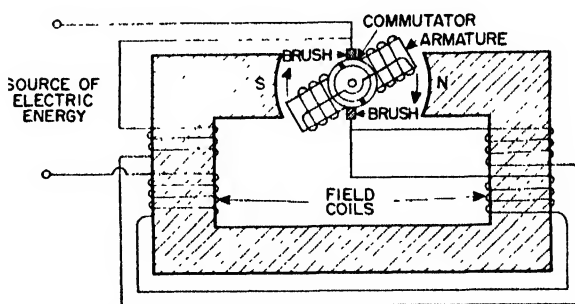


FIG. 2-32. Simple compound-wound direct-current motor

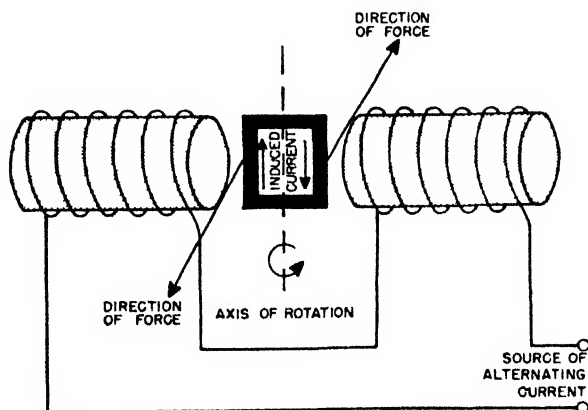


FIG. 2-33. Principle of alternating-current induction motor.

armature current is induced instead of being supplied to the armature through brushes, the direction of the induced current is reversed automatically twice during each alternating-current cycle in the electromagnetic

windings, so that the need for a commutator is eliminated, and the speed of rotation is controlled by the alternating-current frequency. Further discussion of alternating-current motors will be found in Chapter 24.

SECTION 15

PRINCIPLES OF TRANSFORMERS

If two coils of wire are wound on an iron core, and one coil, called the **primary**, is connected to a source of 60-cycle alternating current, such as an AC generator, the voltage across the primary coil will increase from zero to a maximum value in the positive direction in $\frac{1}{240}$ second, return to zero in the next $\frac{1}{240}$ second, increase again to a maximum but in the negative direction in the third $\frac{1}{240}$ second, and will finally return to zero in the fourth $\frac{1}{240}$ second. The current fluctuates in a similar manner following the voltage, continuously repeating the cycle every $\frac{1}{60}$ second. As the current in the primary coil reverses direction twice each cycle or $\frac{1}{60}$ second, the magnetic field accompanying this changing current sweeps back and forth across the other coil, called the **secondary** coil, at the same rate. This fluctuating magnetic field induces an alternating voltage in the secondary coil, even though there is no direct connection between the two coils. A device of this kind is known as a **transformer** (Figure 2-34). Transformers

in the secondary coil as in the primary. In general, the ratio of the voltage across the primary (V_1) to that across the secondary (V_2) is the same as the ratio of the number of turns in the primary (N_1) to that in the secondary (N_2), or expressed mathematically:

$$\frac{V_2}{V_1} = \frac{N_2}{N_1}$$

Accompanying this change in voltage there is a change in the current, but in the inverse ratio; that is, if the voltage is stepped up 100 times in the secondary, the current will be reduced about 100 times. In general, for a transformer, the product of the voltage (V_1) and current (I_1) in the primary is equal to the product of the voltage (V_2) and current (I_2) in the secondary; expressed mathematically:

$$\frac{V_1}{V_2} = \frac{I_2}{I_1}$$

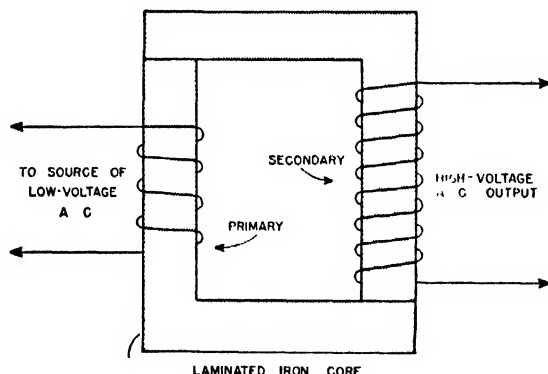


FIG. 2-34. Simple step-up transformer.

will not operate on continuous direct current, because no induced current can be produced by a steady direct current. For this reason they are nearly always operated on alternating current because it is simpler to use AC than to provide an interrupted direct current. The iron core serves, as in an electromagnet or armature, to guide the magnetic field from one coil through the other. In practice this iron core is laminated; that is, it is made up of several thin iron strips, which arrangement increases the resistance of the core to the flow of parasitic or eddy currents that tend to cause heating of the core and reduce the efficiency of the transformer.

The transformer is important because of its ability to "step up" or "step down" the voltage from a given source. If a voltage 100 times greater than that available is required, it is merely necessary to provide 100 times as many turns in the secondary as in the primary; on the other hand, if a voltage of $\frac{1}{100}$ of that available is needed, it may be obtained by using $\frac{1}{100}$ as many turns

Even though a voltage is induced in the secondary, there is no current whatever in the secondary until it is connected to some device that completes the circuit. Furthermore, the current in the primary is nearly zero if there is no current in the secondary. When current flows in the secondary, the primary current increases correspondingly, so that the electric power output and input are always in balance. Thus, the transformer is a kind of self-adjusting device.

In transmitting electric energy through wires over long distances, it is customary to employ high voltages and low currents, rather than low voltages and high currents; the reason for this is that in low-voltage, high-current transmission, over a long length of wire, the voltage drop (IR) is large, and even worse, the power loss (I^2R) is very great. Thus, step-up transformers are employed to raise the voltage from alternating-current generators, which is usually 120 to 3,000 volts, up to voltages of 12,000 to 120,000 for transmission. At the place where the electric energy is to be used, the voltage must then be reduced again, by means of a step-down transformer, to voltages of 110 or 240, in order to operate ordinary motors, lights, and appliances. A modern high-voltage, low-current transmission line is shown schematically in Figure 2-35.

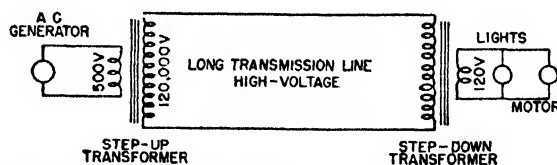


FIG. 2-35. High-voltage, low-current transmission system.

SECTION 16

CURRENT RECTIFICATION

Since, for economic reasons, alternating current is almost universally used for transmission and distribution, some method must be available to change it into direct current when the latter is more desirable for industrial use. The two principal types of equipment used for changing large amounts of alternating-current power to direct-current power in the steel industry are motor-generator sets and rectifiers.

Motor-Generator Sets—The simplest form of motor-generator set consists of an alternating-current motor and a direct-current generator. In practice, several direct-current generators may be operated by one alternating-current motor. Each of the generators can be designed so that the voltage at which the direct current is delivered is different for each generator.

Frequency Changers—Motor-generator sets are not always built for the purpose of changing alternating current into direct current. Sixty cycles per minute is the most commonly used frequency in this country. However, in some sections of the country, industry requires alternating current at the frequency of 25 cycles for use in motor drives. A special type of motor-generator set, called a **frequency changer** is employed for this purpose. The alternating-current motor for such a set is designed for the frequency at which the power is delivered and the generator is designed to deliver alternating current of the desired frequency.

Mercury-Arc Rectifiers—For changing large amounts of alternating-current power to direct-current power, an electronic type of rectifier more commonly known as the mercury-arc rectifier is employed in steel-plant service. The principle of operation of a mercury-arc rectifier is illustrated schematically in Figure 2-36. A rectifier may be described as a device which will permit current to pass through it in only one direction. Thus, when alternating current is applied, current passes through the rectifier during the positive half cycles, but not during the negative half cycles. If single-

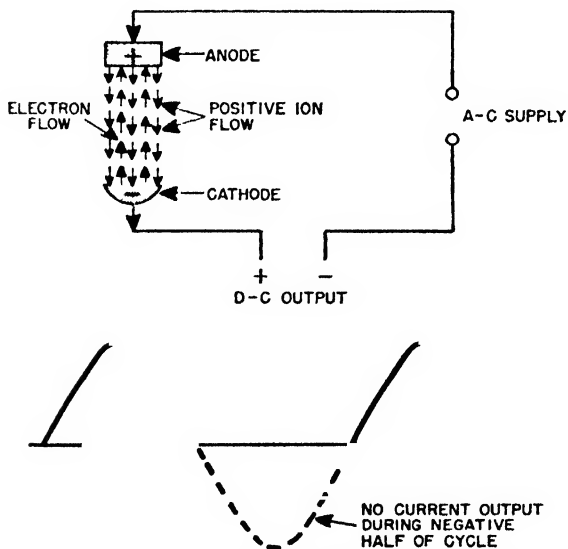


FIG. 2-36. Principle of operation of all electronic rectifiers. With this simple arrangement, half-wave rectification occurs. (Courtesy, "Iron and Steel Engineer.")

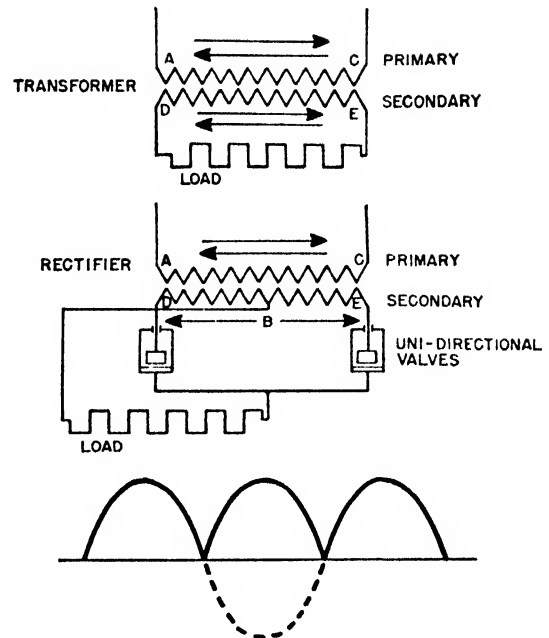


FIG. 2-37. Diagram of single phase mercury arc rectifier arrangement to provide full-wave rectification. (Courtesy, "Iron and Steel Engineer.")

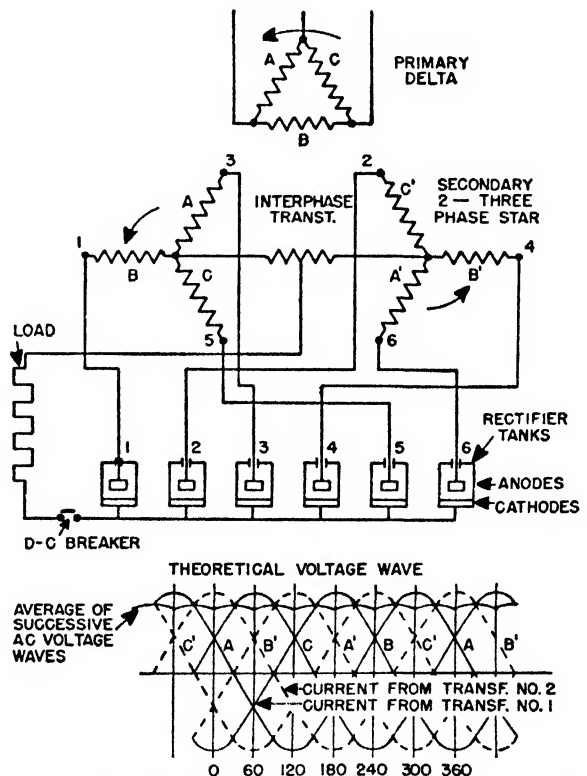


FIG. 2-38. Diagram of six phase ignitron or excitron type mercury arc rectifier. (Courtesy, "Iron and Steel Engineer.")

phase alternating current is rectified in this manner, half-wave rectification results, as indicated at the bottom of the illustration.

All electronic rectifiers depend upon a flow of electrons from cathode to anode. This flow will not start until the anode is positively charged with respect to the cathode. Once started, the flow of electrons continues until the anode becomes negative with respect to the cathode. This happens at the end of each positive half cycle. Mercury vapor from the pool that serves as a cathode provides the conducting medium for the arc through which the current flows.

The mercury-arc rectifier depends for the start and continuation of electron flow upon the ignition and maintenance of a hot spot upon the cathode each positive half cycle. This hot spot provides the necessary vaporization of mercury.

The simplest practical form of full-wave rectification consists of a single-phase transformer with a rectifying element in each leg of the secondary circuit. The return circuit is connected to the mid-point of the transformer winding (Figure 2—37). As the current in the primary of the transformer flows from left to right, a voltage is

set up in the secondary from right to left. The mercury-arc rectifier in the left-hand leg of the secondary permits flow of current in one direction only, so that the current flows through that rectifier, through the load circuit, and back to the neutral connection in the secondary. When the direction of current is reversed in the primary of the transformer, the flow of current in the transformer secondary is through the right-hand half of the secondary winding, through the rectifier at the right, through the load circuit, and back to the neutral connection of the secondary. The rectified voltage is always positive, varying from zero to a maximum positive value for each half cycle. The dotted line indicates how the voltage would have changed from positive to negative if rectification were not employed.

In order to obtain large power capacities for practical use, it is necessary to obtain direct-current voltages of a more uniform value than can be obtained by single-phase rectification. To obtain this result, a six or more phase rectifier is normally used (Figure 2—38). By this means, the voltage never falls outside of set limits and never becomes zero, so that a satisfactorily steady direct current is delivered to the load circuit.

Chapter 3

FUELS AND COMBUSTION

SECTION 1

INTRODUCTORY

Any substance capable of producing heat by combustion may be termed a **fuel**. However, it is customary to rank as fuels only those which include carbon and hydrogen and their compounds. Wood was the earliest fuel used by man. Coal was known to exist in the fourth century B.C., and petroleum was used by the Persians in the days of Alexander. Prehistoric records of China and Japan are said to contain references to the use of natural gas for lighting and heating.

Heat generated by the combustion of fuel is utilized in industry directly as heat or is converted into mechanical or electrical energy. Fuel has become the major source of energy for manufacturing enterprises. In America, fuel has been produced and exploited to a greater extent than in any other country. The United States became the leading coal-producing country of the world in 1889, and in 1902 took the lead in crude-petroleum production from Russia. The United States is also the leading producer of natural gas.

Fuel enters significantly into manufacturing costs, as it generally represents one of the four largest items of expense. In some industries it is the largest. The steel industry is one of the three major consumers of coal but, since coal is used so universally, consumes only about 15 per cent of the total produced. In periods of high pro-

duction, the steel industry expends annually over a quarter of a billion dollars for coal. A modern fully-integrated steel plant consumes approximately a ton of coal for each ton of steel ingots produced. The steel industry also consumes large quantities of natural gas and petroleum. A comparison of petroleum consumption in the steel industry to total production is not so significant as with coal. However, some steel plants rely on oil as their major fuel.

The enormous annual consumption of coal, petroleum, and natural gas in the United States has provoked interest in the natural resources of these materials. Our known resources are still the greatest in the world, but the best and most readily accessible petroleum and natural gas are said, by some authorities, to approach depletion, at the present high rates of consumption, before 2000 A.D. However, when the less desirable coal beds are included, the known coal reserves have been estimated to be sufficient to last anywhere from three hundred to two thousand years. Some industries are developing synthetic fuels, particularly liquid and gaseous fuels from lower-grade coal, in anticipation of the time when their use will be justified economically. Efficient fuel utilization has been intensified not only by the prospect of depletion, but also by rising costs of fuel production.

SECTION 2

CLASSIFICATION OF FUELS

The Classification of Fuels—Fuels are classified usually into three general divisions; viz., **solid**, **liquid**, and **gaseous**. Fuels in each general division can be classified further as **natural**, **manufactured**, or **by-product**. Fuels found in nature sometimes are called **primary fuels**; those manufactured for a specific purpose or market, together with those which are the unavoidable by-product of some regular manufacturing process, are called **secondary fuels**. The primary fuels serve as the principal raw materials for the secondary fuels. Table 3—I gives a classified list of the important fuels. It also lists some interesting by-product fuels, many of which have been utilized by industry to conserve primary fuel.

Importance of Each Class—Although there has been a decline in the use of coal, and a proportional increase in petroleum and natural gas consumption, it is improbable that either of the latter two fuels, because of the great difference in reserves, will ever surpass coal as an industrial fuel, unless they are made synthetically from coal. Coal is the major fuel of public utilities for the generation of power and is essential to the steel industry for

the manufacture of coke. It has been supplanted to a large extent by liquid fuels for the generation of motive power by railroads. It is a major raw material in many chemical plants as a source of carbon, hydrogen, and their compounds. Industrial coal consumption is about five times that of household.

The growth of petroleum consumption has been phenomenal in the past forty years due to the increasing demand for its distillation products. Gasoline, the most important product, is used as a motor fuel. Light oil is used for Diesel engines. Distillate and residual fuel oils, and some crude petroleum of too low commercial value for distillation, are used for industrial and domestic heating. Crude and refined petroleum of various grades is used for lubrication of all types of machinery and prime movers. Petroleum is the base for many synthetic products and is competitive with coal chemicals in a number of important applications.

The marketed production of natural gas increased fourfold in the twenty years prior to 1945, and the quantity used in that year was approximately 165,000,000 tons

Table 3—I. Classification of Fuels

General Division	Primary Fuels	Secondary Fuels	
	Natural	Manufactured	By-Product
Solid	Anthracite coal Bituminous coal Lignite Peat Wood	Semi-coke (low temperature coal distillate)	Charcoal—low temperature distillation of wood
		Coke Charcoal Briquettes { Coal slack and culm Lignite Peat Sawdust Petroleum residue Pulverized coal	Wood refuse—shavings, trimmings, tan bark, sawdust, etc. Bagasse—refuse sugar cane Anthracite culm—silt refuse of anthracite screening Coke Breeze { By-product coke—screenings Petroleum coke—petroleum residue Waste Materials { Corn from Grain { Barley Wheat Buckwheat Sorghum
Liquid	Petroleum	Gasoline Kerosene Alcohol Colloidal fuels Fuel oil { Residual oils Distillate oils Crude petroleum Naphtha Vegetable oils { Palm Cottonseed	Coal distillates { Tar Naphthalene } coke manu- Pitch facture Benzol Acid sludge—petroleum refining residue Pulp mill waste
Gaseous	Natural Gas	Producer gas Water gas Carburetted water gas Coal gas Oil gas Reformed natural gas Butane ^(a) Propane ^(a) Acetylene Hydrogen	Blast furnace gas—pig iron manufacture Coke oven gas ^(b) —coke manufacture Oil refinery gas Sewage gas—sewage sludge

^(a)Liquefiable heavier constituents of natural gas.

^(b)Considered by-product of coke manufacture in steel industry but a manufactured fuel in the gas industry. Gas industry produced 45 billion cubic feet of coke-oven gas and purchased 93 billion cubic feet (a total of 138 billion cubic feet) for sale in 1952, of a total national production of 923 billion cubic feet. (Based on figures from "Gas Facts," American Gas Association, 1952 Data; and "Mineral Industry Surveys," U. S. Dept. of the Interior, Bureau of Mines, July 1953.)

of coal equivalent, equal to about one-third the annual production of coal. During 1950, marketed production of natural gas was estimated to have increased 16 per cent to 6,281 billion cubic feet (263,800,000 tons of coal equivalent), the largest annual increase in quantity and percentage of the previous decade. Natural gas has replaced coal to a considerable extent for domestic and industrial

heating due to the installation of very large pipe lines from producing to consuming centers, the rise in the price of solid fuel, the relative level in the price of natural gas over the intervening time, and its convenience, cleanliness, controllability and versatility as a fuel. The by-product gaseous fuels—coke-oven gas and blast-furnace gas—are major iron and steel industry fuels.

SECTION 3

PRINCIPLES OF COMBUSTION

Fuels consist essentially of one, or a mixture of two or more, of four combustible constituents: (1) solid carbon, (2) hydrocarbons, (3) carbon monoxide, and (4) hydrogen. In addition to these combustible constituents, nearly all commercial fuels contain inert material, such as ash, nitrogen, carbon dioxide, and water. Bituminous coal is an example of a fuel which contains all four of the combustible constituents named above, and coke is an example of a fuel containing only one (solid carbon). The constituents which make up liquid

fuels and many coals are quite complex, but since these complex constituents decompose or volatilize into the four simpler constituents named above before actual combustion takes place, a knowledge of the combustion characteristics of these constituents is sufficient for nearly all practical applications. All of these four constituents of fuels except carbon are gases at the temperatures where combustion occurs. Combustion takes place by combining oxygen, a gas present in air, with the combustible constituents of a fuel. The complete combustion

of all fuels generates gases. It is apparent, therefore, that a review of the properties, thermal values and chemical reactions of gases is necessary for an understanding of any class of fuel.

Since fuels are used to develop heat, a knowledge of heat terms and the principles of heat flow are essential for the efficient utilization of this heat. The combustion of fuels involves, besides combustion reactions, the factors and principles which influence speed of combustion, ignition temperature, flame luminosity, flame development, flame temperature and limits of flammability. The ensuing divisions of this section deal generally with these subjects. Sections 5, 6 and 7, respectively, deal specifically with the combustion of solid, liquid and gaseous fuels.

Calorific Value of Fuels—Heat is measured under the English system in terms of **British thermal units (Btu)**; under the metric system the corresponding unit is the **caloric**. A **Btu** is defined as 1/180 of the amount of heat required to raise the temperature of a pound of water from 32° F to 212° F, or the average amount of heat per 1° F in this range. A **large caloric** is 1/100 of the heat required to raise the temperature of a kilogram of water from 0° C to 100° C or the average amount of heat per 1° C. The large caloric is equal to 1000 small calories and to 3.9683 Btu. The heat given up or absorbed by a body between two temperatures, providing no change of state or of allotropic form is involved, is known as **sensible heat**. The heat given up or absorbed by a body when a change of state takes place and no change of temperature is involved is known as **latent heat**. For example, a pound of water absorbs 180 Btu of sensible heat when being heated from 32° to 212° F, and absorbs 970.4 Btu of latent heat when one pound of water at 212° F is changed to steam at 212° F. Sensible heat and latent heat are used frequently in combustion calculations, particularly in problems dealing with flue-gas losses. Their significance is indicated in describing gross and net heating value of fuels.

The **gross heating value** of a fuel is the total heat developed by the combustion of a fuel at constant pressure after the products of combustion are cooled back to the

starting point, assuming that all of the water vapor produced is condensed; that is, the gross heating value includes both sensible and latent heat. The **net heating value** of a fuel is defined as the heat developed by the combustion of a fuel at constant pressure after the products of combustion are cooled back to the starting point, assuming that all of the water vapor remains uncondensed. Accordingly, the net heating value includes only sensible heat. The starting point usually is taken at either 32° F or 60° F. A starting point of 60° F has been used in all the tables and figures in this chapter, as it is generally the base for combustion calculations in the steel industry.

When a fuel contains neither hydrogen nor hydrocarbons, no water vapor is produced by combustion and the gross and net heating value will be the same, as in the case of burning carbon or carbon monoxide. The heating value or calorific value of a fuel may be determined on a dry or wet basis. The determination may be made by laboratory tests employing **calorimeters**, or by calculation. The process of determining the calorific value of solid and liquid fuels by a calorimeter consists in completely oxidizing the fuel in a space enclosed by a metal jacket (called the **bomb**) so immersed that the heat evolved is absorbed by a weighed portion of water contained in an insulated vessel. From the rise in temperature of the water, the heat liberated by one gram of the fuel is calculated. The best types of calorimeters for solid and liquid fuels are those called **oxygen-bomb calorimeters** in which the fuel is burned in the presence of compressed oxygen. Gas calorimeters are of different construction to permit volumetric measurement of the gas and its complete combustion under non-explosive conditions, as well as absorption of the heat produced in a water jacket. The **Junkers-type continuous-flow calorimeter** is a common type. The usual basis for reporting the calorific value of a gas is **gross Btu per cubic foot of saturated gas measured at 60° F and 30 in. Hg**. A **saturated gas** is one which contains the maximum amount of water vapor it can hold without any condensation of water taking place.

The heating value of a given fuel can be obtained by multiplying the calorific value of each gas by its per-

Table 3—II. Essential Gas Combustion Constants^(a)
(60° F and 30 In. Hg, Dry Gases) *

Gas	Formula	Molecular Weight	Specific Gravity Air = 1.0	Heat of Combustion				Cu. Ft. per Cu. Ft. of Combustible					
				Btu per Cu. Ft.		Btu per Lb.		Required for Combustion			Flue Products		
				Gross	Net	Gross	Net	O ₂	N ₂	Air	CO ₂	H ₂ O	N ₂
Carbon.....	C	12.01	14,093	14,093
Hydrogen.....	H ₂	2.016	0.06959	325.0	275.0	61,100	51,623	0.5	1.882	2.382	1.0	1.882
Oxygen.....	O ₂	32.000	1.1053
Nitrogen.....	N ₂	28.01	0.9718
Carbon Monoxide.	CO	28.000	0.9672	321.8	321.8	4,317	4,347	0.5	1.882	2.382	1.0	1.882
Carbon Dioxide...	CO ₂	44.01	1.5282
Methane.....	CH ₄	16.041	0.5543	1013.2	913.1	23,879	21,520	2.0	7.528	9.528	1.0	2.0	7.528
Ethane.....	C ₂ H ₆	30.067	1.04882	1792	1641	22,320	20,432	3.5	13.175	16.675	2.0	3.0	13.175
Ethylene.....	C ₂ H ₄	28.051	0.9740	1613.8	1513.2	21,644	20,295	3.0	11.293	14.293	2.0	2.0	11.293
Propylene.....	C ₃ H ₆	42.077	1.4504	2336	2186	21,041	19,691	4.5	16.939	21.439	3.0	3.0	16.939
Acetylene.....	C ₂ H ₂	26.036	0.9107	1499	1448	21,500	20,776	2.5	9.411	11.911	2.0	1.0	9.411
Benzene.....	C ₆ H ₆	78.107	2.6920	3751	3601	18,210	17,480	7.5	28.232	35.732	6.0	3.0	28.232
Hydrogen Sulphide.	H ₂ S	34.076	1.1898	647	596	7,100	6,545	1.5	5.646	7.146	SO ₂ = 1.0	1.0	5.646
Sulphur Dioxide....	SO ₂	64.06	2.264

^(a)From "Fuel-Flue Gases"—American Gas Association—420 Lexington Ave., New York.

*For gases saturated with water at 60° F, 1.73% of the Btu value must be deducted.

centage of the total fuel volume, and then totaling the individual Btu values of the separate constituents. The heat of combustion for various dry elementary gases may be found in Table 3—II. For instance, the gross heating value of dry blast-furnace gas is 92.58 Btu per cu. ft. for the composition given below, calculated as follows:

Composition of Gas	Composition by Volume (%)	Gross Heating Value Btu per cu. ft. each Component	Gross Heating Value of each Fraction
CO ₂	12.76	0.0	0.0
CO	25.69	321.8	82.67
H ₂	3.05	325.0	9.91
N ₂	58.50	0.0	0.0
Total.....	92.58

In the calculation of the heating value of gases saturated with water vapor, the volume of water vapor must be deducted from the unit volume of gas. For instance, a cubic foot of dry CO gas has a heating value of 321.8 Btu, but when saturated with water vapor at 60° F and 30 in. Hg, a cubic foot has a heating value of only 316.2 Btu. The amount of water vapor present in saturated mixtures can be calculated from data in Table 3—III, as discussed later under "Gas Laws."

Thermal Capacity, Heat Capacity and Specific Heat—The thermal capacity or heat capacity of a substance is expressed as the amount of heat required to raise the temperature of a unit weight of the substance one degree. The British system uses Btu per lb. per ° F, while the metric system uses cal. per gm. per ° C. The specific heat of a substance is the ratio of the heat capacity of that substance to the heat capacity of water. Thus, specific heat is always a ratio, expressed as a number; for example, the specific heat of wrought iron is 0.115. There is no further designation, as this means that if it takes a certain number of Btu to heat a certain number of pounds of water a certain number of degrees F, it will take only 0.115 times as many Btu to heat the same number of pounds of wrought iron the same number of degrees F, and the same figure, 0.115, obviously applies if the metric system has been used. The amount of heat required to raise the temperature of equal masses of different substances to the same temperature level varies greatly; that is to say, the specific heat varies greatly; also the specific heat of the same substance varies at different temperatures. Usually, it is necessary to know the amount of heat required to raise the temperature of a substance some appreciable amount. For that purpose, formulae and tables are usually accessible in handbooks for supplying the mean specific heat between various temperature levels. Two values of specific heat for gases are usually given: (1) specific heat at constant pressure, and (2) specific heat at constant volume. The difference is due to the heat equivalent of the work of expansion caused by an increase of volume resulting from a temperature rise. Normal combustion practice with gases in steel plants deals with a constant pressure condition (or nearly so), and for this reason specific heat at constant pressure is used. The mean specific heat is the average value of the specific heat between two temperature levels. It is obtained by integrating the equations for instantaneous specific heat over the temperature limits desired, and dividing this quantity by the difference between the temperature limits.

Table 3—III. Water Vapor Pressure

Temp. (°F)	Pressure (In. Hg)
32	0.1804
35	0.2034
40	0.2477
45	0.3002
50	0.3625
55	0.4357
60	0.522
65	0.622
70	0.739
75	0.873
80	1.029
85	1.209
90	1.417
95	1.655
100	1.929
105	2.236
110	2.589
115	2.995
120	3.446
125	3.954
130	4.525
135	5.165
140	5.881
145	6.680
150	7.569
155	8.557
160	9.652
165	10.863
170	12.199
175	13.671
180	15.291
185	17.068
190	19.014
195	21.144
200	23.467
205	26.003
210	28.755
212	29.922

The heat content is the heat contained at a specified temperature above some fixed temperature. It is calculated by multiplying the weight of a substance by the mean specific heat times the temperature difference, or $H = \text{weight} \times \text{mean specific heat} \times (T_2 - T_1)$. For convenience in calculations with gases, the unit weight of the volume of a cubic foot of gas is often used.

Gas Laws—The volume of a gas varies in direct proportion to its absolute temperature and inversely as its absolute pressure (Charles' Law). Absolute temperature is the temperature above minus 459.6° F at 29.921 inches of mercury column in the English system, and the temperature above minus 273° C at 760 mm. of mercury in the metric system. In combustion calculations using the English system, 460° F and 30 in. Hg are assumed sufficiently accurate for all practical purposes. For instance, the volume of 40,000 cu. ft. of gas measured at 60° F and 30 in. Hg, when heated to 1800° F and 30 in. Hg, is equal to:

$$40,000 \times \frac{460 + 1800}{460 + 60} = 174,000 \text{ cu. ft.}$$

and the volume of 40,000 cu. ft. of fuel gas measured at 60° F and 8 in. Hg gage pressure is equal to 31,579 cu. ft. at standard conditions (60° F and 30 in. Hg), calculated as follows:

$$40,000 \times \frac{30}{30 + 8} = 31,579 \text{ cu. ft.}$$

The total pressure of any gas mixture is equal to the sum of the pressures of each component. Each component produces a partial pressure proportional to its concentration in the mixture. Therefore, in a mixture of water vapor and any other gas, each exerts a pressure proportional to its percentage by volume, and since water has a definite vapor pressure at various temperatures, as shown in Table 3—III, the concentration of water vapor in a gas is limited. When this limit of water vapor is reached, the gas is said to be **saturated**. Any drop in temperature or increase in pressure from that point will cause condensation of the water vapor; for instance, the water vapor in 1000 cu. ft. of saturated fuel gas measured at 60° F and 30 in. Hg is calculated as follows:

$$1000 \times \frac{0.522}{30} = 17.40 \text{ cu. ft.}$$

(0.522 is the partial pressure of water vapor in a saturated mixture at 60° F and 30 in. Hg—see Table 3—III). The amount of water vapor which will condense at various temperatures may be ascertained by the use of Table 3—III.

In many combustion calculations it is necessary to convert volumes to weights and vice versa. Such conversions may be made very conveniently by using the molar units, pound-mol. (abbreviated lb.-mol.) or gram-mol. (abbreviated gm.-mol.) A lb.-mol. (or gm.-mol.) is that quantity whose weight in pounds (or grams) is the same number as the number of the molecular weight. Thus, the molecular weight of oxygen is 32, so that a lb.-mol. of oxygen is 32 lbs. of oxygen (or a gm.-mol. of oxygen is 32 gm. of oxygen). In the English system, a lb.-mol. of any gas theoretically occupies 359 cu. ft. at 32° F and 29.92 in. Hg; or at 60° F and 30 in. Hg, the usual reference points for combustion problems in the steel industry, a lb.-mol. occupies 378.4 cu. ft. (**Avogadro's Principle**). Actually some gases deviate slightly from this figure, but for gases and air at pressures normally encountered in combustion practices, the ideal figure is entirely satisfactory. The simplicity of using the lb.-mol. for weight or volume conversions is shown by the following example. The weight of a cubic foot of dry air is:

$$0.21 \text{ (\% vol. of } O_2 \text{ in air)} \times \frac{32 \text{ (molecular wt. of oxygen)}}{378.4} = 6.72$$

$$0.79 \text{ (\% vol. of } N_2 \text{ in air)} \times \frac{28 \text{ (molecular wt. of nitrogen)}}{378.4} = 22.12$$

$$\text{Weight in lbs. of a lb.-mol. of air} = 28.84$$

$$\frac{28.84}{378.4} = 0.076 \text{ lbs. (wt. per cu. ft. of dry air at 60° F and 30 in. Hg)}$$

The volume of a pound of dry air at 60° F and 30 in. Hg is:

$$\frac{378.4}{28.84} = 13.1 \text{ cu. ft.}$$

The relation of an ideal gas to its volume and pressure is expressed by the formula:

$$PV = NRT$$

where:

- R = gas constant
- P = absolute pressure
- V = volume
- N = number of mols.
- T = absolute temperature of gas

The numerical value of R in the above equation depends upon what units (English or metric) are used to measure P, V, N and T. Values of R for various combinations of units for measuring the other quantities are as follows:

T	P	V	N	R
° F abs.	lb. per sq. in.	cu. ft.	lb.-mol.	10.72
° F abs.	inches of Hg	cu. ft.	lb.-mol.	21.85
° C abs.	mm. of Hg	liters	gram-mol.	62.37
° C abs.	atmospheres	liters	gram-mol.	0.08205

The application of the foregoing formula is shown by the following example. Calculate the volume occupied by 100 lbs. of natural gas having a composition of 80 per cent CH_4 , 18 per cent C_2H_6 , and 2 per cent N_2 by volume at a gage pressure of 8 in. Hg and a temperature of 100° F.

$$P = 30 + 8 = 38 \text{ in. Hg absolute}$$

The weight in lbs. of a lb.-mol. of the gas is:

$$\begin{aligned} CH_4 &= 0.80 \times 16 = 12.8 \\ C_2H_6 &= 0.18 \times 30 = 5.4 \\ N_2 &= 0.02 \times 28 = 0.56 \\ &\hline &18.76 \end{aligned}$$

$$N = \frac{100}{18.76} = 5.33$$

$$T = 100 + 460 = 560$$

Substituting these values in the equation for a perfect gas ($PV = NRT$):

$$\begin{aligned} 38 V &= 5.33 \times 21.85 \times 560 \\ V &= 1716 \text{ cu. ft.} \end{aligned}$$

Combustion Calculations—The combustion of fuels is carried out by chemical reaction with air, and occasionally with air enriched with oxygen, or with pure oxygen. **Dry air** is a mixture of the following gas volumes under average conditions:

N_2	= 78.03%
O_2	= 20.99%
Argon	= 0.94%
CO_2	= 0.03%
H_2	= 0.01%
Total	= 100.00%

In combustion calculations it is customary to include all elements in dry air (other than oxygen) with the nitrogen, as shown below:

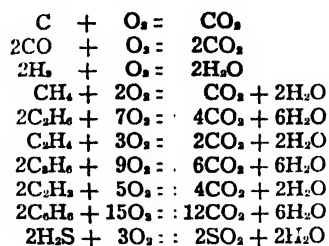
	% by Volume	% by Weight
Oxygen	20.99	23.11
Nitrogen	79.01	76.89

Only the oxygen in the air reacts with a fuel in combustion processes. The nitrogen acts as a diluent which must be heated up by the heat of the reaction between the oxygen and the fuel. It, therefore, reduces the temperature of the flame and reduces the velocity of combustion.

Water vapor which is present in air also acts as a diluent. The amount of moisture present in air is generally stated in terms of humidity. Air is capable of being saturated with water vapor the same as other gases as described under "Gas Laws." Air which is saturated completely with water vapor has a humidity of 100 per

cent; if only 50 per cent saturated, it has a humidity of 50 per cent (Table 3—IV).

The principal combustion reactions are:



The amount of oxygen required and consequently air, together with the amount of the resultant products of combustion, may be calculated by the use of molecular weights and the proper chemical equation. For instance, it will require $(32 \div 12)$ or 2.667 lbs. of O_2 to burn 1 lb. of C, and since dry air contains 23.11 per cent by weight of O_2 , the weight of dry air required to burn one pound of carbon will be $(2.667 \div 0.2311)$ or 11.540 lbs. The product of combustion, CO_2 , will amount to $[(12 \div 32) \div 12] = 3.667$ lbs.

Combustion calculations using gases are more conveniently made in volumetric units. For instance, to burn a cubic foot of CO completely to CO_2 requires $\frac{1}{2}$ cu. ft. of O_2 in accordance with the molecular relationship in the equation. The dry air required would be $(0.5 \div 0.209)$ or 2.382 cu. ft. For burning a cubic foot of methane, CH_4 , to CO_2 and H_2O , the air required would be $(2.0 \div 0.209)$ or 9.528 cu. ft.

Combustion calculations are necessary to determine the air requirements and the products of combustion for burning fuels of various compositions. The per cent of air

used above theoretical requirements is called per cent excess air; the per cent below, the per cent deficiency of air. Typical combustion data on a dry basis for burning gaseous fuels of the compositions stated are shown in Table 3—V. In making calculations to include the water vapor which may be present in a saturated or partially saturated gas and in air, the same general method may be used by adding water vapor to the fuel-gas composition, and by adding the volume of water vapor which is introduced through air in the products of combustion column, headed H_2O .

In order to maintain combustion, a fuel must, after it has been ignited, be able to impart sufficient heat to its air-gas mixture so that it will not drop below ignition temperature, the minimum point of self-ignition. Too lean or too rich a mixture of a fuel with air is unable to support combustion. An upper and lower limit of flammability exists for all gases. The limits of flammability, as well as ignition temperatures, for a number of gases are shown in Table 3—VI.

In the design of burners or in the selection of fuel for a specific purpose, consideration of velocity of combustion is of major importance. Since gaseous fuels are composed usually of a mixture of combustible gases, a knowledge of the relative combustion speed of each elementary gas will provide means for evaluating this factor in any gaseous mixture. The velocity of combustion, or rate of flame propagation, of a given fuel, is influenced by three factors: (1) degree to which the air and gas are mixed, (2) temperature of the air-gas mixture, and (3) contact of the air-gas mixture with a hot surface (catalyst). By intimately mixing air and gas, combustion may be accelerated and a shorter, sharper flame developed. In the case of a gas containing large amounts of hydrogen, intimate mixing will provide a combustion reaction of explosive velocity relative to that of a gas containing large amounts of methane. Inert

Table 3—IV.⁽¹⁾ Properties of Air

Temp.	Volume of One Pound Cu. Ft.	Weight of One Cu. Ft.	Grains* of Water Vapor per Pound of Dry Air for Percentage Humidities of			
			25%	50%	75%	100%
32	12.36	0.0809	6.6	13.2	19.9	26.5
40	12.56	0.0796	9.1	18.2	27.3	36.4
45	12.69	0.0788	11.0	22.1	33.2	44.2
50	12.81	0.0781	13.4	26.7	40.1	53.5
55	12.94	0.0773	16.1	32.2	48.3	64.4
60	13.063	0.07655	19.3	38.6	58.0	77.3
65	13.19	0.0758	23.1	46.3	69.5	92.6
70	13.31	0.0752	27.6	55.2	82.9	110.5
75	13.44	0.0745	32.8	65.7	98.6	131.4
80	13.57	0.0737	38.9	77.9	116.9	155.8
85	13.69	0.0730	46.1	92.2	138.3	184.4
90	13.82	0.0724	54.4	108.8	163.2	217.6
95	13.94	0.0718	64.1	128.1	192.2	256.3
100	14.07	0.0711	75.3	150.6	226.0	301.3
105	14.19	0.0705	88.5	177.	265.	354.
110	14.32	0.0699	104.	208.	311.	415.
115	14.44	0.0693	121.	243.	365.	486.
120	14.57	0.0686	142.	285.	427.	569.
125	14.70	0.0680	167.	333.	500.	667.
150	15.32	0.0653	371.	742.	1113.	1485.
175	15.95	0.0627	927.	1851.	2777.	3703.
200	16.58	0.0603	4016.	8033.	12049.	* 16065.

⁽¹⁾ From "Combustion"—American Gas Association.

* 7000 grains = 1 lb.

FUELS AND COMBUSTION

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Table 3-V. Combustion Data * for Blast-Furnace, Coke-Oven and Natural Gas

Gas Comp.	% by Volume	BLAST FURNACE GAS (ALL VOLUMES AT 60°F AND 30" Hg.)									
		10% Excess Air					50% Excess Air				
		Cu. Ft. Products of Combustion per Cu. Ft. of Fuel					Cu. Ft. of Fuel				
		CO ₂	H ₂ O	SO ₂	O ₂	N ₂	CO ₂	H ₂ O	SO ₂	O ₂	N ₂
CO ₂	11.5	.115									
N ₂	60.0										
CO	27.5										
H ₂	1.0										
Total	100.0										

Gas Comp.	% by Volume	COKE OVEN GAS (ALL VOLUMES AT 60°F AND 30" Hg.)									
		10% Excess Air					50% Excess Air				
		Cu. Ft. Products of Combustion per Cu. Ft. of Fuel					Cu. Ft. of Fuel				
		CO ₂	H ₂ O	SO ₂	O ₂	N ₂	CO ₂	H ₂ O	SO ₂	O ₂	N ₂
CO ₂	1.4	.0429									
H ₂ S	0.6	— .0190									
O ₂	0.4										
N ₂	4.3										
CO	5.6										
H ₂	28.4										
CH ₄	2.5										
C ₂ H ₄	0.8										
C ₂ H ₆	0.6										
Total	100.0										

Gas Comp.	% by Volume	NATURAL GAS (ALL VOLUMES AT 60°F AND 30" Hg.)									
		10% Excess Air					50% Excess Air				
		Cu. Ft. Products of Combustion per Cu. Ft. of Fuel					Cu. Ft. of Fuel				
		CO ₂	H ₂ O	SO ₂	O ₂	N ₂	CO ₂	H ₂ O	SO ₂	O ₂	N ₂
CO ₂	0.08	— .002									
O ₂	0.17										
N ₂	81.88										
CH ₄	16.85										
Total	100.00										

Gas Comp.	% by Volume	Cu. Ft. Products of Combustion per Cu. Ft. of Fuel									
		10% Excess Air					50% Excess Air				
		CO ₂	H ₂ O	SO ₂	O ₂	N ₂	CO ₂	H ₂ O	SO ₂	O ₂	N ₂
		From Excess Air	From Excess Air	From Excess Air	From Excess Air	From Excess Air	From Excess Air	From Excess Air	From Excess Air	From Excess Air	From Excess Air
CO ₂	0.08	.001					.001				
O ₂	0.17										
N ₂	81.88										
CH ₄	16.85										
Total	100.00										

*Dry Basis

Table 3—VI. Limits of Flammability and Ignition Temperature for Simple Gases and Compounds*

Simple Gases and Compounds	Limits of Flammability		Ignition Temperature (°F)
	Lower % by Volume Gas in Air	Upper % by Volume Gas in Air	
H ₂	4.1	74	1076-1094
CO	12.5	74	1191-1216
CH ₄	5.3	14.0	1200-1382
C ₂ H ₆	3.2	12.5	968-1166
C ₃ H ₈	2.4	9.5	965 approx.
C ₂ H ₄	3.3	34**	1000-1020
C ₃ H ₆	2.2	10	
C ₄ H ₈	1.7	9	
C ₂ H ₂	2.5	80	763-824
C ₆ H ₆	1.4	8.0	1364
C ₇ H ₈	1.3	6.75	1490

*From "Combustion"—American Gas Association.

**Maximum reported in Bulletin 279, Bureau of Mines.

gases, such as carbon dioxide and nitrogen, present in fuel gases or in a gas-air mixture, reduce combustion velocity. The proportion of nitrogen in a fuel gas-air mixture may be reduced by oxygen enrichment of air for combustion, and combustion speed may, by this means, be accelerated many fold. Such measures also will raise the flame temperature. The use of preheated air for combustion also accelerates combustion of gases. In order to burn large volumes of fuel in a small space, a mixture of air and gas is sometimes directed against a hot, incandescent surface. By increasing the velocity of combustion, higher temperatures are localized close to the burner point. This condition is desirable for some processes and highly undesirable for others. For instance, the scarfing process requires a highly intensive localized heat, while the heating of steel for rolling requires a soft, even distribution of heat over the full surface of the pieces being heated. In order to reduce combustion speed of a gaseous fuel, the air and gas streams may be stratified to produce slow mixing. Such a method creates a **diffusion flame**, a long flame of relatively uniform temperature.

Theoretical flame temperature is the temperature which would be attained by the products of combustion if the combustion of a fuel took place instantaneously, and there were no loss of heat to the surroundings. Such a condition never exists, but theoretical flame temperature represents another measure for comparing fuels. Fuels which develop a high flame temperature by combustion are more capable of producing a higher thermal efficiency in practice than those which develop low flame temperatures. The efficiency of heat utilization is the relation of the total heat absorbed by a substance to the heat supplied. Since the temperature level at which waste gases leave a furnace is usually fixed within a relatively narrow range, the higher the flame temperature the higher the potentiality for heat absorption by the substance to be heated. The theoretical flame temperature of a fuel may be calculated by balancing the sum of the net heating value of a given quantity of fuel and the sensible heat of the air-gas mixture against the heat content of the products of combustion. Theoretical

flame temperature so calculated should be corrected for dissociation of CO₂ and H₂O at temperatures in excess of 3000° F. The theoretical flame temperatures for a number of important gaseous fuels are given in Table 3—XVI. The reader is referred to "Combustion" published by the American Gas Association for a full explanation of the calculation of theoretical flame temperatures and the dissociation of gases at elevated temperatures.

There are a number of factors which determine the character, size and shape of a gas flame. Gases burned at very high combustion velocity will produce very little or no luminosity regardless of the kind of gas. The velocity and volume with which the air-gas stream leaves a burner or furnace port, the fuel-air ratio, and the amount of non-combustible material in the fuel will influence the length and shape of a flame. The kind of gas to be burned has a very great effect upon the character of the flame. Carbon monoxide and hydrogen burn with an invisible to a clear blue flame, while the hydrocarbon gases, methane, ethane, etc., are capable of developing highly luminous flames. The principal reason that these gases burn with a luminous flame is due to the thermal breakdown of the hydrocarbons into carbon and hydrogen, and under combustion conditions which permit this, the carbon particles are heated to incandescence thereby giving the flame its luminous appearance. The luminosity of a flame may be decreased or increased by varying the supply of air. A deficiency of air below theoretical requirements will increase luminosity and it also usually will lengthen the flame. An excess of air will decrease luminosity and shorten the flame with most burners or furnace ports. Increasing the temperature of preheat of the air for combustion will reduce luminosity, as is also the case when water vapor (steam), which may be introduced with the gas, air, or for atomization of liquid fuels, is increased. A luminous flame has a number of desirable qualities, the principal one being its greater ability to transfer heat by radiation from a fixed temperature level. However, it should be noted that a luminous flame is obtained usually at a lower temperature level than when the same fuel is burned with a lower degree of luminosity.

SECTION 4

HEAT FLOW

Heat flow is caused by a difference in temperature, and heat is transmitted in three ways, namely, by conduction, by convection, and by radiation.

Conduction is the transmission of heat through a solid body without visible motion of the body, as through a steel bar. The amount of heat transferred through a homogeneous solid by conduction is expressed by the formula:

$$Q = \frac{K A \Delta t}{L}$$

where Q = Btu transmitted per hour

K = conductivity, in Btu per hour per sq. ft. per °F per inch of thickness

A = area in sq. ft.

Δt = temperature difference in °F

L = length of heat transfer path in inches.

The flow of heat through a non-homogeneous solid body by conduction is expressed by the formula:

$$Q = \frac{\Delta t}{\frac{L_1}{K_1 A_1} + \frac{L_2}{K_2 A_2} + \frac{L_n}{K_n A_n}}$$

where:

L_1, L_2 and L_n = the respective length of heat transfer path through the various resistances.

K_1, K_2 and K_n = the corresponding conductivity factors of the various resistances expressed in Btu per hr. per sq. ft. per °F per inch of thickness.

A_1, A_2 and A_n = the corresponding areas expressed in square feet.

Convection—When heat is transmitted by the mechanical motion of gas or water currents in contact with a solid, or by gas currents in contact with a liquid, the transfer of heat is by convection. In the transfer of heat by convection, it is necessary to conduct heat through the relatively stationary film between the moving and stationary bodies. This film becomes thinner as the velocity of the currents parallel to its surface increases. The transfer of heat by convection is expressed by the formula:

$$Q = a_s A \Delta t$$

where:

Q = Btu transmitted per hour

a_s = film coefficient (Btu per sq. ft. per °F per hr.) dependent upon the velocity, specific gravity and viscosity of the moving fluid, and the conductivity of the film.

A = area in sq. ft.

Δt = temperature difference

Radiation refers to the transmission of heat through space without the help or intervention of matter. This is the means by which the heat of the sun reaches the

earth, and by which much of the heat of combustion of fuels is utilized in high-temperature processes in the steel industry. When radiant energy strikes any body a certain proportion of the total is reflected, while that absorbed is reconverted to heat energy. A perfectly black body is one that will not reflect radiations falling upon it but absorbs them all. The coefficient of reflectivity of a body receiving radiation is equal to one minus its black body coefficient. Emissivity refers to the rate at which a body radiates heat, and this rate depends upon the temperature of the body and the nature of its surface. Kirchhoff's Law shows that the absorptivity and emissivity of a given surface are numerically equal at the same temperature. The Stefan-Boltzmann Law states that the total energy of a black body is proportional to the fourth power of its absolute temperature. The net effect of heat transfer between two bodies is shown by the equation:

$$Q = 0.174 EA \left[\left(\frac{T_1}{100} \right)^4 - \left(\frac{T_2}{100} \right)^4 \right]$$

where:

Q = Btu transmitted per hour

0.174 = radiation factor for a perfect black body

E = emissivity or "black body" factor

A = sq. ft. of surface

T_1 = temperature of body giving off heat, in °F absolute

T_2 = temperature of body receiving heat, in °F absolute

The emissivity factors for various materials at specified temperatures are shown in Table 3—VII.

Table 3—VII. Emissivity Factors (A perfect absorber or radiator = 1)

Material	E
Polished aluminum at 445°F	0.039
Polished aluminum at 1075°F	0.056
Polished brass at 570°F	0.031
Polished nickel at 715°F	0.086
Polished nickel plated steel at 72°F	0.052
Bright tinned steel plate at 75°F	0.071
Polished mild steel	0.288
Cast iron—machined—at 72°F	0.437
Cast iron—liquid—at 2425°F	0.282
Cast iron—rough oxidized	0.97
Mild steel—dull oxidized—from 79° to 672°F ..	0.96
Firebrick glazed through use at 1830°F	0.75
Silica brick (rough)	0.81

In the generation of heat from fuels, the character of the flame and its proximity to the receptor of heat is particularly significant in the transfer of heat by radiation. The amount of heat transferred from a flame varies widely and in proportion to its degree of luminosity. The transfer of heat by radiation varies inversely with the square of the distance between the transmitter and receptor of radiant energy. For that reason, flames should be kept close to the substance to be heated where high heat transfer rates are desirable.

SECTION 5

SOLID FUELS AND THEIR UTILIZATION

The solid fuels have played a significant role in the evolution of our modern, industrial civilization. Coal in particular has been of far-reaching importance in that it has provided the prodigious amount of energy essential to the development of the iron and steel industries. Vast quantities of this energy source remain to be exploited but the rate of utilization far exceeds the rate at which coal is being formed. It follows that the efficient use of the remaining supply is desirable. Toward this end, modern coal research is directed.

The earliest-formed coals thus far encountered occur in the Silurian strata of Bohemia. It is not until Lower Carboniferous time (see Table 3—VIII), however, that the source materials of coal began to accumulate in significant quantities. Every continent, including Ant-

arctica, contains some coal and no system of rocks younger than the Silurian is devoid of this important substance. In North America major concentrations of source materials were accumulated during the Carboniferous, Cretaceous and Tertiary periods. A similar statement can be made for Europe but, in contrast, some of the most important Asiatic coals occur in Triassic and Jurassic rocks.

Coal Resources—The known coal deposits in the United States are greater than those of any other country. Based on U. S. Geological Survey estimates, the reserves of all grades of coal were in excess of 2,500 billion net tons in 1950. This would be enough to supply requirements for a long period in the future if all present coal reserves were available economically and of ac-

Table 3—VIII. Geologic Time Divisions.

ERAS	PERIODS		EPOCHS	MILLIONS OF YEARS
Cenozoic	Quaternary		Recent	<div> <div>70</div> <div>200</div> <div>500</div> <div>3,000+</div> </div>
			Pleistocene	
			Pliocene	
			Miocene	
			Oligocene	
			Eocene	
Mesozoic	Cretaceous			
	Jurassic			
	Triassic			
Paleozoic	Carboniferous	Permian		
		Pennsylvanian (Upper Carboniferous)		
		Mississippian (Lower Carboniferous)		
	Devonian			
	Silurian			
	Ordovician			
	Cambrian			
Proterozoic	Algonkian	Keweenawan		
		Huronian		
Archeozoic	Archean	Timiskamian		
		Keewatin		

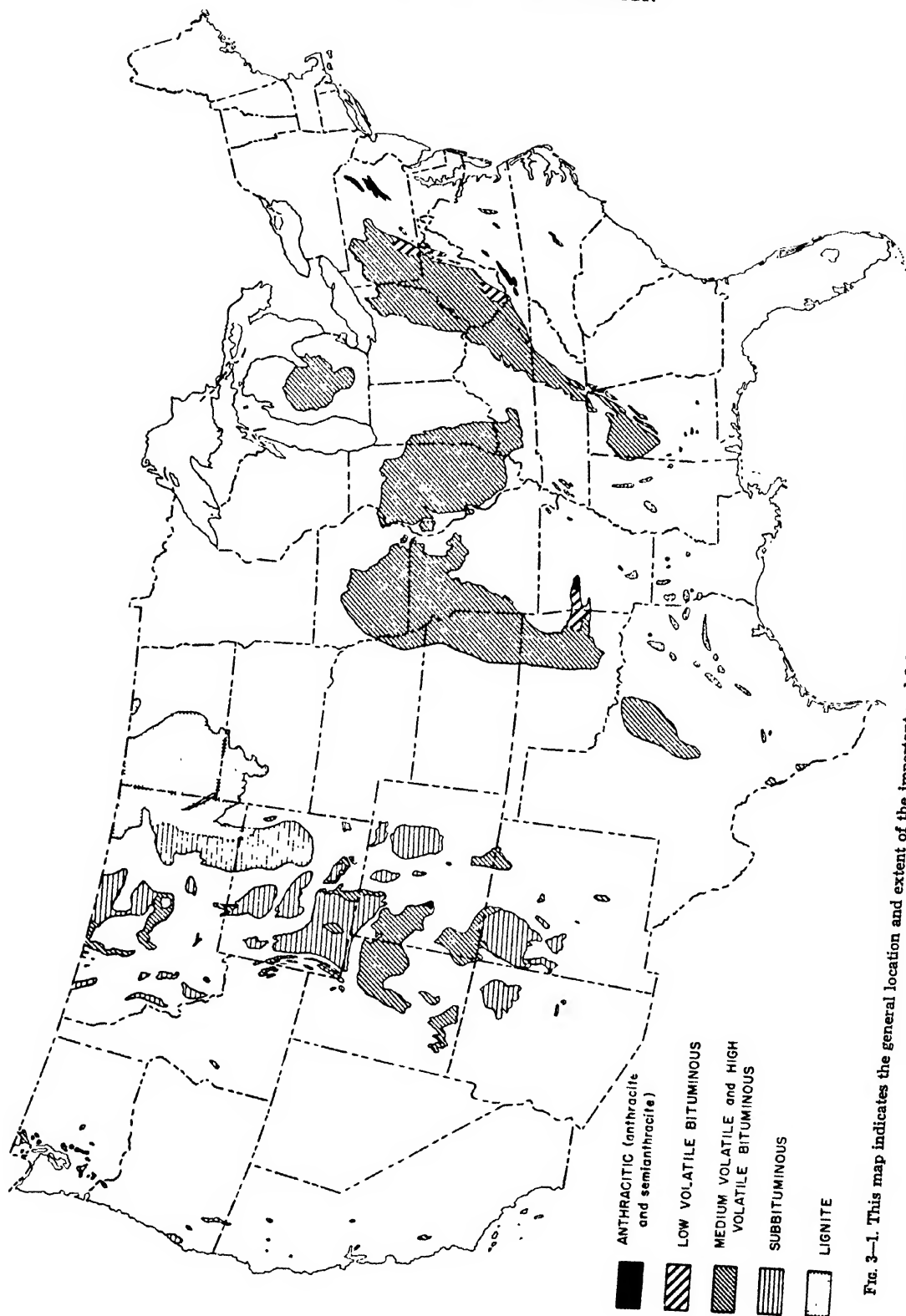


FIG. 3-1. This map indicates the general location and extent of the important coal fields of the United States. (Map prepared by United States Geological Survey.)

ceptable quality. A considerable quantity of the reserves of better quality coking coal have been utilized in the past and it is apparent that in the future it will be necessary to use coals requiring efficient extraction, cleaning, and other processing to assure proper utilization.

For obvious reasons, the steel industry has been striving to use coals which would produce metallurgical coke of optimum quality with a minimum of processing. Concentrations of coals of this class are found chiefly in the Appalachian area, although isolated deposits also exist in some Central and Western states. The preponderance of total coal reserve in the United States is in the form of lower-rank coals in the Great Plains, the Rocky Mountains, the Pacific Coast states and the Gulf region (see Figure 3-1). The manner in which this material can be used most effectively remains to be determined.

Origin and Composition of Coal—Coal is known to be a complex mixture of plant substances which have been altered in varying degrees by physical and chemical processes. Ordinarily, plant material, upon death, fails to accumulate because micro-organisms induce complete decomposition. Under certain circumstances, notably those associated with forested fresh-water swamps, the activities of bacteria and fungi are inhibited by antibiotic solutions which are common in this type of environment. As a result the rate of accumulation exceeds that of decomposition and dispersion. Under such conditions a brown fibrous deposit known as **peat** is formed. Peat is the first step in the formation of coal.

Peat deposits formed millions of years ago subsequently were submerged through vertical movements of the earth's crust, in which position they became covered by deposits of sedimentary rocks. Later movements of the earth's crust raised many of these deposits to various heights above sea level. In the meantime, the peat had been changed, through agencies of biological action, pressure, and heat, into coal. The better ranks of coal in this country were formed during the Carboniferous period, the geologic period when conditions were most favorable for plant accumulation and decomposition. Included in the present deposits that originated in that period are the coal fields of the Appalachian and Central states.

The rate at which peat forms depends upon the rapidity of plant growth and the manner in which tissue increment is related to the rate of decomposition. It has been

estimated that approximately one century is required to form a deposit of mature, compacted peat one foot in thickness. Certain studies of volatile matter relationships suggest that a three-foot-thick deposit of mature peat is required to produce a one-foot-thick layer of bituminous coal. These and other data indicate that a coal seam which is several feet thick may require a time span of thousands of years for its formation. If, in the course of time, the peat is subjected to the necessary conditions it becomes modified to **brown coal** and, when adequately consolidated, to **lignite**. From the lignitic stage, the material passes progressively through the sub-bituminous, bituminous, semi-anthracite and anthracite stages with a gradual change in the composition of the individual components of the complex mass. The elementary compositions of coal shown in Table 3-IX, illustrate the gradual concentration of carbon and loss of oxygen in the various stages of coal formation.

Peat varies in appearance from a light, brown-colored, fibrous material to a very black and dense, muck-like sediment. Lignite is usually brown in color and commonly shows a woody texture. It contains a large amount of moisture and usually disintegrates to a powdery mass as it dries on exposure to air. Sub-bituminous coal varies in color from very dark brown to black and fractures irregularly. Bituminous coal is black in color and usually exhibits a banded structure due to the alternate dull and vitreous layers of varying thickness. Coals of the high-volatile bituminous rank commonly burn with a smoky, yellow flame. Anthracite coal is black, hard and brittle and has a high luster. It ignites less easily than bituminous coal and burns with a short bluish-yellow flame producing little smoke. The characteristics of semi-anthracite coal are intermediate between those of bituminous coal and anthracite.

All of the solid natural fuels contain both combustible and non-combustible materials. The combustible material is composed mainly of carbon, hydrogen and, to a lesser extent, sulphur. The non-combustible constituents are water, nitrogen and oxygen, and a variety of mineral materials usually referred to as **ash**.

The bituminous coals are of greatest interest in view of the fact that essentially all coking coals fall in this category. The lustrous black bands which are conspicuous in a lump of bituminous coal are generally referred to as **vitrain** although some American coal petrographers

Table 3-IX. Typical Elementary Compositions of Raw Solid Fuels (Per Cent)

	Moisture Content (Sample Taken at Tipple)	Elementary Composition — (Dry Basis)					
		Carbon	Hydrogen	Oxygen	Nitrogen	Sulphur	Ash
Peat ⁽¹⁾	91.0	58.0	5.7	35.0	1.2	(4)	(3)
Lignite ⁽²⁾	40.0	65.6	4.5	20.2	1.1	1.0	7.6
(North Dakota)							
Sub-Bituminous ⁽²⁾ ...	17.1	73.8	5.1	15.2	1.6	1.0	3.3
(Wyoming)							
Bituminous ⁽²⁾	2.5	78.3	4.1	1.7	1.3	3.2	11.4
(Low-volatile B)							
Anthracite ⁽¹⁾	5.5 ⁽⁴⁾	85.6	2.0	1.4	0.8	0.6	9.6
(Northeastern Pa.)							

⁽¹⁾Johnson, A. J. and Auth, G. H., Fuels and Combustion Handbook. McGraw-Hill Book Co., Inc., New York (1951).

⁽²⁾Aresco, S. J. and Haller, C. P., Analyses of Tipple and Delivered Samples of Coal. Bureau of Mines Report of Investigation 4972, October, 1953.

⁽³⁾Composition of peat reported on ash-, sulphur-, and moisture-free basis.

⁽⁴⁾Moisture content of anthracite coal reported as average of 4 to 7 per cent.

employ the term **anthraxylon** in preference. Following U. S. Bureau of Mines terminology, the anthraxylon is derived from woody plant tissues and is surrounded by a dull ground mass made up of translucent attritus, opaque attritus and fusain. The attrital portion is composed of finely comminuted fragments of altered plant materials. Fusain is a friable, charcoal-like substance derived from woody tissues and is a term used universally without modification.

In addition to the readily recognizable bands of vitrain and fusain, European and Asiatic coal investigators have found it useful to identify silky, minutely striated layers within a coal as **clarain**. Layers of dull, compact coal are called **durain**. Thus, coal seams can be thought of as being composed, usually, of various mixtures of **vitrain**, **fusain**, **clarain** and **durain**, each occurring in the form of layers which are visually observable. Coals made up largely of vitrain and clarain are spoken of as **bright coals** whereas coals containing a high percentage of durain are called **splint coals**. Bright coals are generally better coking coals than splint coals, vitrain apparently playing an important part in the carbonization process. Fusain will not coke, but in small percentages it may actually increase coke strength. If present in concentrations greater than 18 to 20 per cent it begins to exert a deleterious effect. The fixed carbon content is higher and the volatile matter content is lower in fusain than in the other "banded ingredients."

Microscopic study has shown the banded components to be composed of identifiable plant entities called **phyterals**, but of greater significance is the fact that the vitrain, fusain, clarain and durain are made up of numerous components or **macerals** which can be defined by their physical and chemical properties. Durain, for example, may include several macerals (vitrinite, semifusinite, micrinite, cutinite, etc.) which are easily distinguished by their differing optical properties. It is probable that additional information regarding the nature and variability of these individual coal components will contribute materially to the effective and efficient utilization of all types of coal.

Chemical Composition and Coal Classification—There are two methods commonly employed to determine the chemical composition of coal; namely **ultimate analysis** and **proximate analysis**. An ultimate analysis determines the quantities of carbon, hydrogen, oxygen, nitrogen, sulphur, chlorine and ash in dry coal; a proximate analysis determines the fixed carbon, volatile matter, moisture and ash contents and, usually, the sulphur content. The proximate analysis which also includes determina-

tion of sulphur is used most commonly, since it furnishes most of the data required for normal commercial evaluations.

In order to provide a measure of the distillation products obtainable from coal used in coal-chemical and hydrogenation plants, another method of analysis called the **tube distillation assay test** (also called **progressive distillation**) is used. Table 3—X compares the composition of a bituminous coal as derived by the three methods mentioned.

Using data provided by chemical, physical or petrographic analyses, coals are classified according to **grade**, **rank** and **type**. The grade is determined by the amount, nature and fusibility of the ash and by the sulphur content. Classification according to rank is based upon the degree of metamorphism within the coal series from the level of lignite to that of anthracite coal. The American Society for Testing Materials ranks coals according to their fixed-carbon content on a dry basis, and the lower rank coals according to Btu content on a moist basis. The classification of coals by rank adopted by the American Society for Testing Materials (A.S.T.M. Specification D388-38), is shown in Table 3—XI.

In the United States, coals are also classified into **types** and such terms as **bright**, **semi-splint**, **splint**, **cannel** and **boghead** coal are applied. The data required are obtained from microscopic studies. The United States Bureau of Mines standards indicate that bright coal must have less than 20 per cent opaque matter, semi-splint must have between 20 and 30 per cent, and splint coal must be made up of more than 30 per cent of this ingredient. Cannel and boghead coals are **non-banded** and are characterized by a small percentage of anthraxylon (vitrain). Boghead possesses a high percentage of volatile oils and gases, and contains an abundance of algal material. Cannel, or candle coal, is so named because it can be ignited with a match or a candle flame and it burns with unusual brilliance. Cannel coal is non-coking, often contains large quantities of spore and pollen materials, and like boghead, has a high content of volatile oil and gas.

Mining of Coal—Seams of coals vary in thickness throughout the world from a fraction of an inch to over 250 feet. In this country the thickest seams are found in the sub-bituminous coals of the West, one of which approaches 100 feet. In the East, the Mammoth bed in the anthracite fields of Pennsylvania attains a thickness of 50 to 60 feet but is found to be quite variable when traced laterally. The Pittsburgh seam at the base of the Monongahela series in the Appalachian area is note-

Table 3—X. Comparison of Results of Common Methods for Analyzing a High-Volatile Bituminous Coal

Proximate Analysis Dry Basis (%)		Progressive Distillation Dry Basis (%)		Ultimate Analysis Dry Basis (%)	
Ash	7.16	Coke ^{aw}	70.36	Ash	7.16
Fixed Carbon (by diff.)	59.98	Tar	5.02	C	79.41
Volatile Matter	32.86				
Total	100.00	Total NH ₃	0.32	H	5.14
		Water	4.76	N	1.46
Total Sulphur, dry	1.02	Volatile Sulphur	0.31	O	5.81
Phosphorus, dry	0.005	Light Oil	0.35	S	1.02
Moisture	5.67	Gas	18.88		

^{aw}This comprises 63.20 per cent total carbonaceous residue containing a small amount (1.0 to 1.5 per cent) of non-volatitized volatile matter, and 7.16 per cent ash.

Table 3—XI. Classification of Coals by Rank^a

Legend: F.C.—Fixed Carbon.

V.M.—Volatile Matter.

Btu.—British thermal units

Class	Group	Limits of Fixed Carbon or Btu. Mineral-Matter-Free Basis ^a	Requisite Physical Properties
I. Anthracitic	1. Meta-anthracite	Dry F.C., 98 per cent or more (Dry V.M., 2 per cent or less)	Nonagglomerating ^b
	2. Anthracite	Dry F.C., 92 per cent or more and less than 98 per cent (Dry V.M., 8 per cent or less and more than 2 per cent)	
	3. Semianthracite	Dry F.C., 86 per cent or more and less than 92 per cent (Dry V.M., 14 per cent or less and more than 8 per cent)	
II. Bituminous ^d	1. Low volatile bituminous coal	Dry F.C., 78 per cent or more and less than 86 per cent (Dry V.M., 22 per cent or less and more than 14 per cent)	Either agglomerating or nonweathering ^f
	2. Medium volatile bituminous coal...	Dry F.C., 69 per cent or more and less than 78 per cent (Dry V.M., 31 per cent or less and more than 22 per cent)	
	3. High volatile A bituminous coal....	Dry F.C., less than 69 per cent (Dry V.M., more than 31 per cent); and moist ^e Btu., 14,000 ^g or more	
	4. High volatile B bituminous coal...	Moist ^e Btu., 13,000 or more and less than 14,000 ^g	
	5. High volatile C bituminous coal....	Moist Btu., 11,000 or more and less than 13,000 ^g	
III. Subbituminous	1. Subbituminous A coal.....	Moist Btu., 11,000 or more and less than 13,000 ^g	Both weathering and non-agglomerating
	2. Subbituminous B coal.....	Moist Btu., 9500 or more and less than 11,000 ^g	
	3. Subbituminous C coal.....	Moist Btu., 8300 or more and less than 9500 ^g	
IV. Lignite	1. Lignite	Moist Btu., less than 8300	Consolidated Unconsolidated
	2. Brown coal	Moist Btu., less than 8300	

^(a)This classification does not include a few coals which have unusual physical and chemical properties and which come within the limits of fixed carbon or Btu. of the high-volatile bituminous and subbituminous ranks. All of these coals either contain less than 48 per cent dry, mineral-matter-free fixed carbon or have more than 15,500 moist, mineral-matter-free Btu.

^(b)If agglomerating, classify in low-volatile group of the bituminous class.

^(c)Moist Btu. refers to coal containing its natural bed moisture but not including visible water on the surface of the coal.

^(d)It is recognized that there may be noncaking varieties in each group of the bituminous class.

^(e)Coals having 69 per cent or more fixed carbon on the dry, mineral-matter-free basis shall be classified according to fixed carbon, regardless of Btu.

^(f)There are three varieties of coal in the high-volatile C bituminous coal group, namely, Variety 1, agglomerating and nonweathering; Variety 2, agglomerating and weathering; Variety 3, nonagglomerating and nonweathering.

Computation of Mineral-Matter-Free Analyses

^(g)Mineral matter was taken as 1.1 times the ash. The values for fixed carbon and Btu. as given in the charts were calculated as follows:

$$\text{Moist fixed carbon} \times \frac{100}{100 - (\text{moisture} + 1.1 \text{ Ash})} = \text{dry, mineral-matter-free fixed carbon}$$

$$\text{Moist Btu.} \times \frac{100}{100 - 1.1 \text{ Ash}} = \text{moist, mineral-matter-free Btu.}$$

Moist, as used in the formulas, refers to the coal containing its natural bed moisture but not including visible moisture on the surface of the coal. For more accurate formulas that apply corrections for the sulphur in the coal, reference should be made to those for fixed carbon and Btu. as given in A.S.T.M. Standards on Coal and Coke.

worthy because of its exceptionally uniform thickness (approximately 7 feet) over thousands of square miles. Figure 3-2 shows the western portion of Pennsylvania in such a manner as to make clear the areal extent as well as the sub-surface relations of the coal-bearing formations of this region. Data are provided as to thickness of seams and distance between coals.

Coal seams may dip gently as shown in Figure 3-2, or they may be horizontal, or they may exist almost vertical with respect to the Earth's surface. Mining problems are often complicated by the fact that seams seldom remain in the same plane throughout their extent. Under present conditions, a coal bed must be at least 30 inches to 36 inches thick to be profitable for mining. Figure 3-2 shows, also, that coal seams vary in their distance from the Earth's surface. U. S. Geological Survey estimates of coal reserves do not include coal seams deeper than 3,000 feet from the surface, although in Great Britain and Europe coal seams at greater depth at present are being mined economically.

The mining of coal is performed by either one of two methods: (1) **Open work or stripping**, also called **contour mining**, or (2) **underground or closed work**. The first method involves removing the formation (overburden) above the seam by stripping with scrapers, bulldozers, or mechanically operated shovels, followed by removing the exposed coal. Stripping is applied to coal seams which are relatively close to the surface, particularly to thick seams underlying overburden not over 75 feet deep, although the development of larger equipment and improved techniques in recent years has justified removal of thicker layers of overburden than this. Production by strip mining has increased greatly since World War I due to reduced labor and material costs and a quicker return on capital investment compared to underground mining. In the United States, strip-mining accounted for nearly 24 per cent of the coal produced in 1950 and 22 per cent in 1951.

Underground mining is performed by either the **room-and-pillar** or the **longwall** method. The room-and-pillar method is in more common use in the United States, accounting for approximately 90 per cent of present underground mining. The longwall method is particularly adaptable to mining seams up to about $4\frac{1}{4}$ feet thick under conditions where the roof may be permitted to settle. It is used more extensively in the mines of the Middle West than in the East. There are a number of modifications applicable to each method. The room-and-pillar system consists essentially of working out rooms, chambers, or breasts in the coal seam from passages (entries) driven from the mine entrance. Entrance to an underground mine is by drift, shaft or slope. The rooms vary in width from about 12 feet to 40 feet, and from 150 feet to 300 feet in length, depending on such factors as weight and character of the overlying and underlying structure and thickness of seam. Pillars separating the rooms vary in width from 6 feet to 100 feet, depending on conditions and mining practice. These pillars are sometimes removed by retreat mining and the coal recovered.

In the longwall method, a continuous mining face is maintained in the coal seam. After mining, the roof is permitted to settle, thirty or forty feet from the mine working face. Waste rock is used to support the roof for maintenance of haulage roadways, which include main roads running diagonally like the spokes of a wheel from a central shaft and auxiliary roads running through the intervening areas.

Prior to the advent of mechanical mining, undercutting of the coal seam preparatory to blasting was done manually. Production per man was low by this method

and required a number of working faces in the mine to produce high mine tonnage. Hand loading of coal into mule-drawn cars was the prevailing practice for many years until development of machinery for both cutting and loading. Electric trolley-type locomotives capable of hauling longer underground trains of cars of increased capacity displaced mule-drawn trains as mine capacity increased.

In a modern underground mine the coal is emptied from the mine car by a rotary dumper which may have a capacity of as many as 37 cars. From the dumper the coal is fed by way of a conveyor or elevator to a shipping station or cleaning plant.

Continuous Mining—The cutting machines and loading machines characteristic of mechanical mining are both single-purpose units, and each performs essentially a single function of mining at the working face. After either unit has completed its work it must be withdrawn from the face to allow other units of the production set-up to move up to the face to carry out succeeding functions. To keep all operating units working at full efficiency, it is necessary to have additional working places near at hand so that the single-purpose machines can enter the places in rotation and carry out their respective functions without interference.

To eliminate some of the difficulties attendant upon the addition of extra working places, multi-purpose machines known as **continuous miners** have been developed and the operation carried out by such machines has been given the name **continuous mining**. Continuous miners combine in a single unit the actions of dislodging the coal from the solid seam and loading it into some unit of a transportation system. Such machines, therefore, combine in one operation the separate steps of cutting, drilling, blasting and loading common to modern mechanical mining methods.

There are several types of continuous miners in operation, one of which is the Konnerth mining machine. This machine is unique in that it employs electrically operated hammers to vibrate the coal at the working face so powerfully that it is shattered and falls to the floor to be picked up by the machine and loaded directly by conveyor into a shuttle car. The machine has two toothed-chain cutters which simultaneously make two horizontal cuts in the coal close to the floor. Two other cutters make vertical cuts in the coal to block out a section to be broken and loaded. After the section has been blocked out by the cutters, the two vibrating hammers of the machine, delivering 1800 blows per minute, are brought into contact with the face of the coal to break it. The cutter bits of the horizontal chain cutters pick up the shattered coal and move it continuously onto the conveyor of the machine.

In certain types of coal, light charges of explosives may be used to partially break the blocked-out coal before the hammers of this type of machine are brought into operation.

Continuous miners of some other types employ toothed-chain cutters to rip the coal from the face, or auger-type cutters that bore into the face, the cut coal in both cases being carried by a conveyor on the machine to a shuttle car or other means of transportation.

Coal Preparation—Specifically, the objective of coal cleaning (often called washing) is removal of solid foreign matter, such as rock and slate, from the coal prior to its use. Reduction of ash and sulphur contents; control of ash fusibility; increase of heating (calorific) value; and improvement of coking properties of the coal can be achieved by this practice. From a coal-cleaning standpoint, the impurities in coal are of two types; namely, those which cannot be separated from the coal,

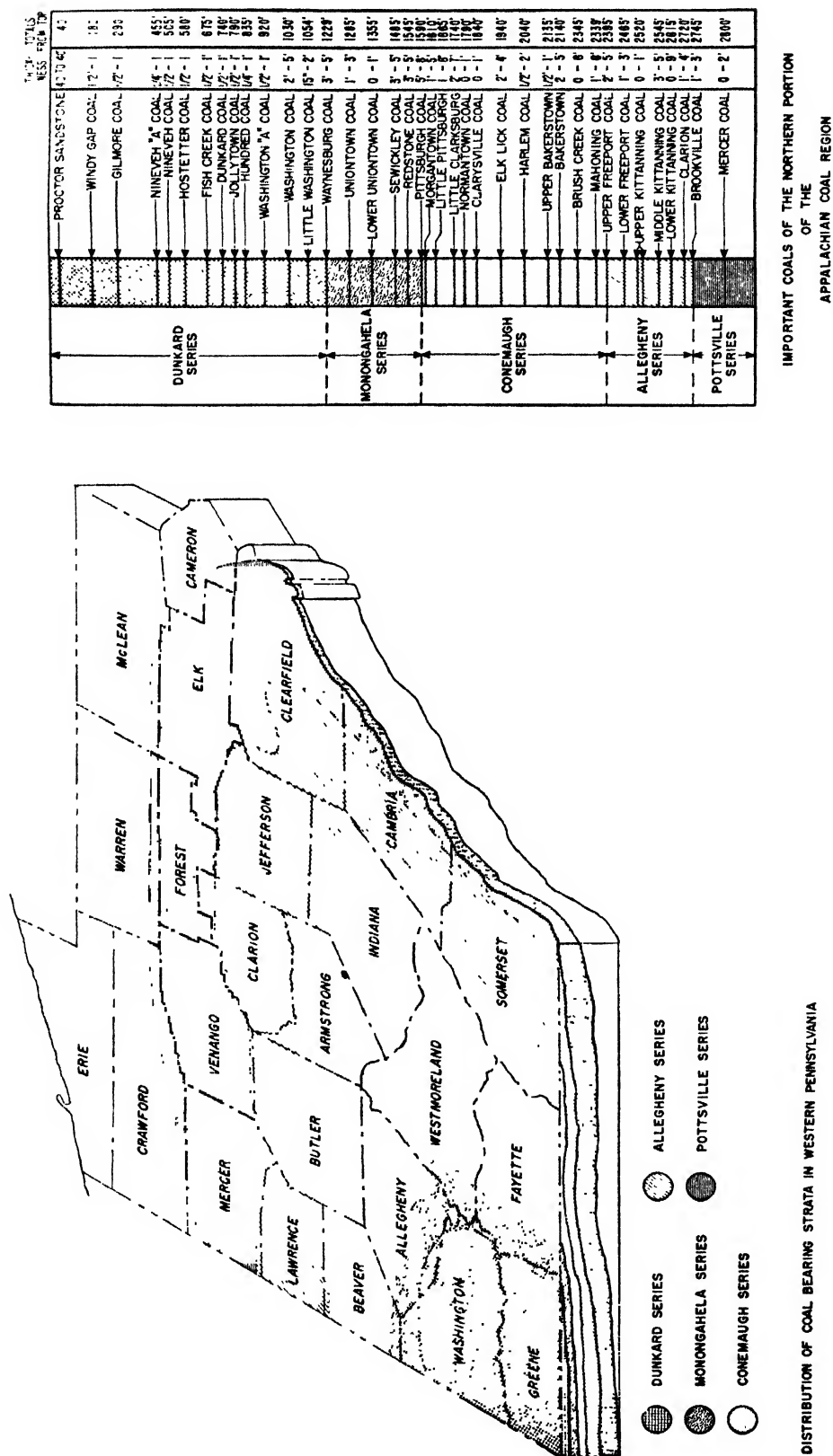


Fig. 3-2. (Left) Distribution of coal-bearing strata in Western Pennsylvania. (Right) Important coals of the northern portion of the Appalachian coal region.

usually called **fixed impurities**; and those which can be removed, herein referred to as **free impurities** or **refuse**. Altogether, these impurities are of eight types, named as follows: (1) residual inorganic matter of the coal-forming plants from which the coal was derived; (2) mineral matter washed or blown into the coal-forming mass during the periods of its formation; (3) pyrites (FeS_2) formed by the reaction of iron sulphate with coal-forming matter; (4) sedimentary deposits during the coal-forming periods which appear as partings, sometimes called "bone," that usually must be mined with the coal; (5) massive deposits formed through deposition on bedding planes; (6) saline deposits, somewhat rare in coal beds of the United States; (7) slate, shale, clay, etc. from the underlying and overlying strata accidentally included in mining; and (8) water or moisture, which includes that naturally carried by the coal in air-dry condition, and excess moisture producing a condition of wetness. Items (1), (2) and, for the most part, (3) form **fixed ash**, while (4), (5), (6), and (7) are partly **free ash-forming materials** that can be removed by hand-picking and suitable mechanical cleaning treatments. Item (8) involves drying operations differing from those required to separate mineral impurities, which is the primary objective of cleaning. Mechanical cleaning is possible because of the difference in specific gravity between the free impurities and the coal, the density of the former being 1.7 to 4.9, while pure coal has a density of about 1.3. Sulphur is present as pyrites, organic compounds, and sulphates, and only part of the pyrites can be removed by cleaning. Phosphorus is usually associated more with bony and impure coal than with clean coal and is, therefore, reduced by washing. Salts, particularly the alkali chlorides, lower the fusion point of the ash, affect coke-oven linings and are troublesome in waste liquors from coking operations.

The preparation of coal starts at the production face in the mine. If loading is done by hand, the miner is required to discard all rock and slate over 3-in. size. If loading is done mechanically (according to the U. S. Bureau of Mines, about 75 per cent of the total bituminous coal mined underground in the United States is being loaded entirely by mechanical means as this is written) no attempt is made to prepare the coal at the face other than to control the tonnage from various sections of the mine if sulphur content of the coal is known to be high or variable.

Since the cleaning problem becomes more difficult the finer the coal, modern practices in extraction and handling employ mining methods for production of the greatest quantity of coarse-size coal.

The cleaning qualities of a particular coal are determined by the **float and sink test**. This test consists in crushing coal to proper size and floating individual samples of it on liquids having densities of 1.30, 1.40, 1.50, 1.60, etc., to determine the weight and character of the material that floats and sinks in each liquid. The proportion of coal, and the ash and sulphur content of the different fractions, provides reasonably complete data on the washability characteristics of a tested coal.

There are two general types of coal cleaning processes: (1) gravity stratification, and (2) nongravity processes. Processes based upon the first type are generally used in this country; nongravity processes have been used extensively in Europe. The processes for cleaning coal may be classified as follows:

A. Gravity separation

1. Wet processes

a. Launder washers

b. Jigs

c. Rising-current classifiers

d. Tables

2. Dry processes

a. Jigs

b. Tables

B. Float-and-sink methods

1. Fine-solids and water flotation

a. Sand

b. Magnetite and other materials

2. Fine-solids and air flotation

C. Froth flotation

A complete description of each of the foregoing processes would be too lengthy for inclusion herein; hence only a brief review will be given of the principles of some of the more important types of processes in use at present. A reference list for further study of this subject is appended to this chapter.

Jigs were probably the earliest type of machine used in the mineral industry to separate materials of different densities. They consist essentially of a box with a perforated base into which the material is placed, and by alternate surges of water upward and downward through the perforations, materials of different specific gravities stratify. Materials having the highest specific gravities remain at the bottom while the lighter material rises. With proper mechanical facilities, a continuous separation is achieved. While jigs are not very efficient in cleaning a mixture of various sizes, they are capable of satisfying some market requirements, and capacities up to 500 tons per hour have been obtained.

The operating principle of a **launder** involves hydraulic stratification or the alluviation of materials of different gravity. The Rheolaveur launder consists of a downwardly inclined flume fitted with compartments spaced at intervals below the flume. Coal and flush water are fed at the high end, and the heavy-gravity material is withdrawn through the compartments. The water and coal having a top size not exceeding 6 inches are fed in at one end of the launder, the first section of which is steeply inclined to give the mixture a high initial velocity. This strong current causes the coal, middlings, and slate to stratify quickly, so that by the time the mixture reaches the flat section, the coal is carried forward while the refuse is definitely retarded. From this point the launder may again be steeply inclined to deliver the flow to a second flat section, finally to make an exit through a chute to dewatering screens. Successively, the heavy-gravity final rejects, including heavy and light middlings are withdrawn from the launder, the last two products being recirculated after withdrawal for the purpose of improving the washing efficiency. The end result is a refuse relatively free of recoverable coal and a coal relatively free of extraneous material. Where the coal has fallen into a middle-gravity classification due to its being laminated with slate or pyrites, it is drawn off, crushed, and freed of these impurities and then rewashed for recovery of the coal. The Rheolaveur launders are capable of cleaning coal of practically any size. Their capacity is variable depending upon the width of the launder and size of the coal to be cleaned. Coarse coal may be treated at rates ranging from 65 to 250 net tons an hour, and fine coal at rates as low as 20 net tons per hour.

With practically all wet-washing systems the water is recirculated. When the water passes through the dewatering screens it contains a considerable amount of small-size coal solids which must be recovered for efficiency reasons. Also, the circulating water must be clarified before it is returned to the cleaning unit. This

clarification is accomplished in various ways, the most important being by the use of the Dorr-type thickener. Settling cones and settling tanks are also used for this purpose. Where the Dorr thickener and settling tanks are used, it is customary to draw off the settlings in the form of a slurry containing 40 per cent to 60 per cent of solids and to further separate the slurry in a vacuum-type filter. The filters deliver a product with approximately 20 per cent moisture.

Upward-current cleaners are somewhat similar to jigs except that a constant current of upward-flowing water carries the coal particles to the discharge, while the falling particles of refuse settle downward through the rising water stream and are removed at a lower level. They are used principally for cleaning anthracite coal and for cleaning a nearly uniform size coal in a single operation. They are capable of cleaning as much as 150 tons of the coarser sizes per hour.

In dense-media classifiers, only a part of the power for separating coal and refuse is supplied by the upward flow of liquid, this separating power being supplemented by using a liquid medium which is heavier than water. The medium employed is a mixture of water and some finely divided solid material, such as sand, magnetite, or barite, which can be separated readily from the washed coal and reused. In the high-density suspension process, the upward flow is discarded entirely, the liquid medium consisting of a mixture which is just dense enough so that the coal floats in it, and the impurities sink. The size of coal has less significance in the efficiency of this process than of those previously described, and material ranging from 1/16 inch to 10 inch can be cleaned in one operation. However, difficulty is encountered in separating the solid material from coal of fine size. Capacities up to 600 tons per hour have been obtained with bituminous coal. The Chance method, which uses a mixture of sand and water, is the most widely used of the heavy-media classifiers in the United States. The Tromp and Barvoys processes, using magnetite and barite respectively as the solid material in the mixture, are used extensively in Europe. In these processes, the specific gravity of the mixture of solid material and water can be varied by changing their proportions to suit the optimum conditions in cleaning. Agitation in the separating cone is supplied by an upward current of water and by mechanical stirring.

A number of high-density solutions, such as those made up of calcium chloride or halogenated hydrocarbons to give the desired specific gravity to the separating medium for cleaning coal, have been used, but to date their application has been limited to the coarser sizes of coal due to the expense of the solution and losses in recovery.

Coal is often cleaned on tables similar to those used in ore dressing. Essentially these tables consist of a slightly inclined rectangular surface having a series of parallel grooves or cleats. The tables are mechanically agitated to permit stratification of the light and heavy material and to cause it to move with the long axis of the table. A current of water is introduced at one side of the table to wash the coal which has settled above the refuse to the discharge edge of the table. The refuse which settles underneath the coal moves longitudinally down the table and is discharged at the end. Tables have been used principally for cleaning coal of the smaller sizes, from dust up to 1-inch.

The disadvantages of water retention associated with wet cleaning have been eliminated by the pneumatic processes. These processes account for about 15 per cent of the coal washed in the United States. They are especially applicable to cleaning a relatively dry coal.

Tables, similar to those described for wet washing, are used where the sizes are under 3/4-inch, except that air currents, rising through a perforated table, are used as the stratification agent rather than water. Jigs and upward-flow processes are also employed. The Stump air-flow cleaner uses air rather than water for classification of material in equipment operating much the same as a jig. A mixture of sand with air, rather than water, to produce a heavy medium for separation is used in some upward-current types of classifiers. In some cases combination wet and dry plants are used to advantage. The restrictions imposed for safety reasons by the U. S. Bureau of Mines, requiring the use of water at underground mining faces to allay coal dust, and the fact that many coals treated need to be cleaned at specific gravities lower than those obtainable by dry air processes, have retarded extensive development in the use of pneumatic methods.

Froth flotation of coal involves agitating fine coal with a mixture of water and a relatively small quantity of some frothing agent. In this process, coal is buoyed to the surface by the froth and removed while refuse settles. Its application has been limited in this country.

The advent of full-seam mechanical coal mining and the increasing need for metallurgical coke of low and uniform ash and sulphur contents has focused attention on the needs for the most efficient types of washers, and also facilities for blending high and low quality coals to obtain the best over-all long-term results.

Carbonization of Coal—The most important use of coal in the steel industry is in the manufacture of coke, which is discussed in detail in Chapter 4. About one-sixth of the total bituminous coal mined in the United States is converted into coke, either in beehive or by-product coke ovens. Of the 72.7 million tons of coke produced in 1950, 66.9 million was by-product and 5.8 million beehive. In 1950, there were 56 active by-product coke plants connected with iron furnaces out of a total of 85 plants, producing 55,987,350 net tons of coke, or 83.8 per cent of the total national by-product coke production.

In recent years, on the average, metallurgical coke produced in by-product ovens has shown an increase in ash and sulphur content above that of former years. An increase in ash and sulphur content of coke affects pig-iron production and economies. The penalties for high and variable ash and sulphur contents in coke in blast-furnace operations are additional slag volume, increased coke consumption, decreased production, and difficulty of furnace control. In addition to the importance of chemical properties, the structural properties (i.e., resistance to shatter, size, crushing strength and bulk density) and possibly the reactivity or combustibility of coke are of paramount importance in coke quality.

The carbonization of coal in by-product ovens entails the production of large amounts of coke-oven gas and tar, important fuels in the steel industry, as well as light oils and various coal chemicals. The yields of gas and tar are largely a matter of the kind of coal used and the temperatures employed in coke manufacture.

Combustion of Solid Fuels—The principal combustion reactions of solid fuels have been given in Section 3 of this chapter, under "Principles of Combustion," and this present discussion will deal with operating factors pertinent to the combustion of solid fuels in steel plants.

The combustion of coke in blast furnaces has been studied by a number of investigators, each of whom has found that combustion takes place in a relatively small space directly in front of each tuyere, as discussed in Chapter 12 on "The Manufacture of Pig Iron."

Coke breeze, produced by screening coke at both the coke plant and blast furnaces, is utilized as a fuel in

steel-plant boiler houses to generate steam. This fuel is burned on chain-grate stokers. Of importance in the combustion of coke breeze on chain-grate stokers is the maintenance of a relatively uniform fuel bed on the grate, 8 to 12 inches thick, to prevent blowholes, and a balanced or slight positive pressure in the furnace at fuel-bed level. The operation of the grate should permit the normal combustion of about 30 pounds of coke breeze per square foot of effective grate area per hour. Chain-grate stokers are particularly adaptable to solid fuels with an ash of low fusion point. The design of front and back arches must take into consideration the fuel to be burned on chain-grate stokers. The arches are utilized to reflect heat and thereby aid ignition on the fuel bed.

Stokers for firing coal have generally been used in steel-plant boilers on units whose capacity is under 100,000 pounds of steam per hour and for units using exclusively a solid fuel. They are often used on boilers to provide flexibility for the adjustment of boiler output to the steam load in plants where there is an insufficient or fluctuating supply of gaseous by-product fuels. The advantage of stokers lies in their ability to control easily the rate of combustion of a solid fuel with efficient use of air. The combustion process on stoker-fired boilers consists essentially in first driving the volatile matter from a continuous supply of fuel, and then oxidizing the carbon in the residue on the stoker. The combustion of the coke-like residue on the grate produces CO_2 and CO . The CO and volatile matter are burned over the grate by secondary air admitted over the fuel bed. The temperature of the fuel bed is affected by the rate of firing and, at the top or hottest part of the bed, varies from about 2250°F at low to 2750°F at high rates. The amount of primary air supplied determines the capacity of stoker-fired furnaces and the effective use of secondary air determines the efficiency of combustion. In well operated and carefully sealed boilers, approximately 20 to 30 per cent excess air will permit combustion of the gases within seven or eight feet above the grate.

Stokers are classified in general according to the travel of the fuel. In an **overfired stoker** the fuel is fed on top of the bed, and in an **underfired** or a **retort stoker** the fuel is fed at the bottom or side of the bed. A **traveling-grate stoker** carries the bed horizontally on the flat upper surface of a conveyor as in a chain-grate stoker. There are a number of modifications of these stoker types. While the fuel bed of a stoker-fired boiler is relatively thin, usually from 4 to 12 inches, compared to a gas-producer bed, similar zones of reaction occur. In over-fired stokers the ash zone is immediately above the grate, followed by the oxidation, reduction and distillation zones. In underfired or retort stokers the distillation of the volatile matter takes place in an oxidizing atmosphere and the volatile products pass through the incandescent residue from combustion rather than through green coal, as in the case of overfired stokers. The normal combustion rates on coal-fired stokers amounts to 30 to 60 pounds of coal per square foot of effective grate area per hour.

Pulverized Coal—The cement industry was the first to use pulverized ("powdered") coal extensively as a fuel. Public utilities and the steel industry began applying pulverized coal on an experimental basis as a boiler fuel about 1917, and by 1935 practically all large boilers (above 100,000 lbs. of steam per hour) in public-utility power stations used this fuel, except for those stations located in the vicinity of oil and natural-gas fields where local fuels were more competitive than coal. Large modern boiler installations in integrated steel plants generally use pulverized coal, either as a standby or as an

auxiliary fuel in conjunction with blast-furnace gas for steam or power generation. Although pulverized coal has been used as a fuel for metallurgical purposes in steel plants, such as in open-hearth, reheating, forge and annealing furnaces, applications have been limited generally to periods of national fuel shortages, such as existed during the first and second World Wars, when the more desirable liquid or gaseous fuels were diverted to other uses and not available.

Pulverized coal offers important combustion advantages over solid fuels and an economic advantage over gaseous and liquid fuels in most sections of the country. Fine particles of coal burned in suspension are capable of developing a highly luminous, high-temperature flame. Coal in this form may be burned normally with less excess of air above theoretical requirements than with a solid fuel, and the rate of heat release from the combustion of pulverized coal is greater than that accomplished with the solid fuel. Coal, when pulverized to the degree common for boiler uses (70 per cent through a 200-mesh screen), has the control flexibility of gaseous and liquid fuels. Practically all ranks of coal, from anthracite to lignite, can be pulverized for combustion and each possesses specific combustion characteristics which largely influence the extent of pulverization.

The ash-disposal problem has been one of the principal deterrents to a more extended use of pulverized coal. In the cement kiln, coal ash is no problem as it is absorbed by the cement in the kiln without adverse effect on the final product. In boilers, the principal difficulty of clogged boiler tubes and deterioration of furnace walls has been overcome by the use of slagging-type furnaces in which the ash in molten form is granulated by water jets at the bottom of the furnace well. The problem of ash contamination resulting from burning fine particles of coal in suspension above a metallic liquid bath or mass of hot steel, damage to refractories from the chemical or physical action of ash, and the clogging of furnace checkers or recuperators from ash accumulation, as well as the normal availability of other fuels, has prevented widespread use of the fuel for metallurgical purposes in steel plants.

Pulverized coal for firing boilers is relatively more modern than stokers. This fuel is used generally on boiler units having a capacity in excess of 100,000 lbs. of steam per hour, or on practically any size of boiler using combination firing with oil or gas. Pulverized coal offers high boiler efficiency, and means for quick regulation of boiler load. The rank of coal pulverized and the extent of pulverization particularly determine the speed of combustion. A high-volatile coal will burn faster than anthracite coal, also one with a lower ash content will burn faster. The process of combustion with pulverized coal is similar to that of lump coal but is of much higher velocity due to the introduction of the particle in suspension in a high-temperature chamber, and the greater surface exposure relative to weight. The release of volatile matter in pulverized coal is practically instantaneous when blown into the furnace, and the speed of combustion of the resulting carbonized particle and volatile gases depends upon the thoroughness with which the pulverized coal has been mixed with air. High combustion temperatures, low ash losses, and low excess air needs (10-20 per cent), with resultant high boiler efficiencies (85 to 90 per cent with good practice), make pulverized coal an ideal boiler fuel. Air for combustion of pulverized fuel is generally preheated, with 10 to 50 per cent of that required introduced ahead of the pulverizer and the balance made up at a point near the burner. This method of introducing the air helps dry

the coal and maintains a non-explosive mix in the pulverized-coal transmission system. The combustion-chamber size for pulverized coal is generally proportioned for a heat-release range of from 20,000 to 100,000 Btu per cu. ft. of combustion space per hour. The differ-

ence in requirements is dependent upon whether pulverized coal is the sole fuel to be used in the chamber, the size of the coal particles, the rank of coal to be pulverized, the ash-slagging temperature of the coal, and the desired temperature for the combustion chamber.

SECTION 6

LIQUID FUELS AND THEIR UTILIZATION

Liquid fuels are essential to many parts of the American transportation system. The movement of passengers and freight by highway and air is dependent upon gasoline, a product of petroleum. The railways of the country are being equipped rapidly with Diesel locomotives powered by fuel oil. Nearly all ocean-going ships are driven by oil, as are the majority of lake and river craft. Liquid fuels have also become of major importance as a source of heat and power in manufacturing plants. The peculiar advantages of petroleum as a source of energy and the available supplies have brought about a phenomenal growth in the petroleum industry. In the period from 1918 to 1950 the consumption of all grades of oil in the United States rose from approximately 450 million to 2,400 million barrels. In 1951, the United States produced somewhat more than 50 per cent of the world's supply of crude petroleum.

Fuel oil, tar, pitch and pitch-tar mix are the principal liquid fuels used in the steel industry. Table 3—XII shows the consumption of fuel oil, tar and pitch for the year 1952.

The liquid fuels consumed annually amount to over 4 million Btu per ton of steel ingots and castings produced. The largest share of this fuel is consumed in melting furnaces. Since melting furnaces require from 3 to 5 million Btu per ton of ingots produced, the significance of liquid fuels in open-hearth practice is apparent.

Tar and pitch are by-products of the manufacture of coke. The virgin tar as it comes from the ovens contains valuable tar-liquor oils which can be extracted and the residue pitch used as a fuel. It is customary to mix virgin tar with this highly viscous residue to provide fluidity for facilitating handling and burning, or to utilize tar in which only the lighter products have been removed by a topping process by which sufficient fluidity is retained. Pitch-tar mixtures and topped tar make available for use as a fuel 78 to 83 per cent of the heat in the virgin tar recovered in the distillation process. The virgin tar produced in the United States for the years 1950 and 1951 amounted to 739,868,767 and 795,311,282 gallons, respectively. Approximately 40 per cent of this (31,300,000 gallons) was used in the steel industry as a fuel in 1950.

The reserves of crude petroleum in this country were estimated at over 25,000,000,000 barrels as of January 1, 1951, and during the prior year (1950) the annual pro-

duction was nearly 1,950,000,000 barrels. Since 1939 there has been a downward trend in new discoveries, and unless new national reserves are found in excess of consumption, it will be necessary to obtain supplies from foreign countries.

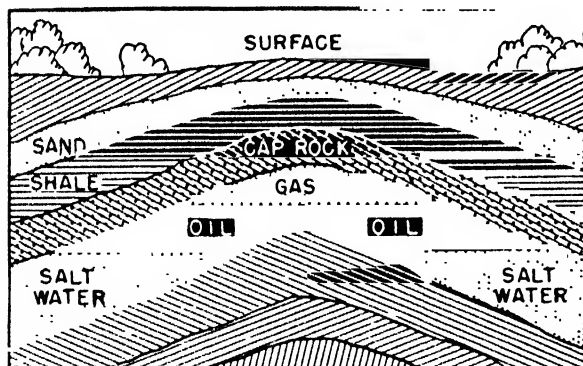
Origin, Composition and Distribution of Petroleum—Classified according to their origins, three main types of rocks make up the outer crust of the earth: igneous, sedimentary and metamorphic rocks. **Igneous rocks** are formed from **magma**, a molten (liquid or pasty) rock material originating at high pressures and temperatures within the earth. Lava is magma that reaches the surface in the liquid or pasty state. Very commonly, the magma cools and solidifies before reaching the surface. In any case, when the molten material cools sufficiently to become solid, igneous rocks are the result. If cooling is slow, the rocks will be crystalline (granite, for example); but if the cooling is rapid, the rocks will be not crystalline but glassy in nature (obsidian, for example). Because of the nature of their origin and their usually dense, non-porous structures, igneous rocks are never hosts to petroleum deposits.

Sedimentary rocks are formed from eroded particles of rocks and soil, carried away by wind or water (and sometimes glacial action) and deposited in seas, lakes, valleys and deltas in relatively even, sometimes very thick, beds or strata (sandstones and shales are formed from deposits of this type). Other types of stratified deposits may be formed by evaporation of land-locked seas (beds of rock salt), by accumulation of the mineral remains of animals (composed chiefly of calcium carbonate, which is the principal constituent of limestone), or by chemical precipitation (gypsum and some limestones originate in this manner). The beds of sand, silt, clay, calcium carbonate or whatever eventually are covered by other sedimentary deposits, sometimes to very great depths. With the passage of long periods of time, pressure of the overlying strata, heat, cementation by chemical means, earth movements, or a combination of these or other agencies, the strata are consolidated into sedimentary rocks, typified by the few mentioned parenthetically earlier. *Petroleum occurs almost entirely in sedimentary rock formations*, principally sandstones and limestones, under certain ideal conditions to be described later.

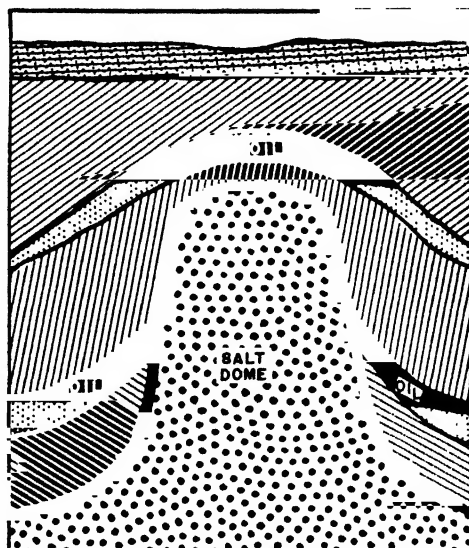
Metamorphic rocks originally were sedimentary or



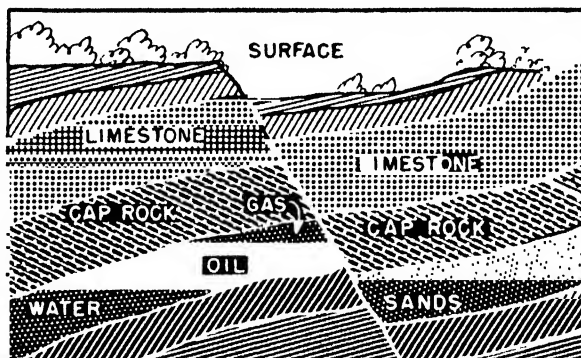
A. STRATIGRAPHIC TRAP.—In the stratigraphic trap, the producing formation gradually pinches out and disappears up the structure. An impervious layer is deposited on top of the sand, thus forming a cap rock. The solid black section represents petroleum accumulated below the cap rock.



B. ANTICLINES.—In an anticlinal structure, the rocks comprising the crust of the earth are folded upward. The oil and gas are usually found on the crest of an anticlinal structure. An impervious cap rock must be present to seal the reservoir and prevent the escape of the gas and oil into higher layers. This cap rock, in one form or another must be present in all reservoirs to contain the oil and gas within the structure.



C. SALT DOMES.—The salt dome is believed to be the result of the intrusion of large masses of salt into the sediments where they are found. This intrusion creates an upward pressure and results in the doming of the overlying sedimentary rocks. In this type of structure, petroleum accumulates within the upturned porous beds about the summit and flanks of the salt core, as indicated by the solid black sections.



D. FAULTS.—A fault is a structural closure caused by the fracturing of the crustal rocks during earth movements. In the process of folding, a reservoir for oil may be formed when a porous rock is brought into contact with an impervious layer, thus forming a trap.

FIG. 3—3. Schematic representation of four geologic structures associated with the underground accumulation of petroleum through natural agencies. From "Fundamentals of Petroleum," NAVPERS 10883, Superintendent of Documents, U. S. Government Printing Office, Washington, D.C.

igneous rocks. Their composition, constitution or structure have been changed through the single or combined action of natural forces such as heat, pressure, or other agencies. Marble, for example, is metamorphosed limestone.

The organic theory of the origin of petroleum, generally accepted by geologists, is that petroleum has been derived from either animal or vegetable matter, or both, by a process of slow distillation, after its burial under beds of sediments. There is evidence to indicate that the animal and vegetable matter was of marine origin; such evidence includes the association of brines with oil, the

visible oily coating on seaweeds found in certain localities, and the optical phenomenon of light polarization by oils similar to that of substances found in certain plants and animals and which is not shown by inorganically synthesized petroleum. The accumulation of the matter from which petroleum has been derived, its burial by sedimentary material, and the action of pressure and heat to cause distillation, has resulted in petroleum formation in many parts of the world. Geological studies indicate that petroleum was not formed in the pools in which it is found, but that the action of water pressing against oil formations caused the petroleum to flow, over

a period of many years, through porous beds or strata to points of accumulation. Pools of oil occur in "traps" in sedimentary rocks such as sandstone or limestone. These traps may be formed in various ways, a few of which are illustrated schematically in Figure 3-3. Essentially, such traps are formed by an impervious layer which prevents upward migration of the petroleum to any further extent. The oil is obtained by drilling wells into these zones of accumulation. The well is encased in a steel pipe through which it is often customary to run a number of smaller pipes to bring the product to the surface.

Crude petroleum is a liquid containing a complex mixture of solid, liquid and gaseous hydrocarbons. The solid hydrocarbons are in solution and the liquid is at least partly saturated with gases (methane, ethane, etc.). The elementary composition of American crude oils from representative fields covers the following ranges:

Carbon	84 to 87
Hydrogen	11.5 to 14.0
Sulphur	0.05 to 1.75
Nitrogen	0.10 to 1.70

Ordinary crude petroleum is brownish-green to black in color with a specific gravity from about 0.810 to 0.985, and an ash content of 0.01 to 0.05 per cent.

The principal constituents in crude oil are the paraffin (C_nH_{2n+2}), naphthene (C_nH_{2n}), and aromatic (C_nH_{2n-6}) series of hydrocarbons, and asphaltic compounds. In paraffin-base crudes, such as found in Pennsylvania, the asphaltic content is low, only traces of sulphur and nitrogen are found, and the specific gravity averages about 0.810. In mixed-base crudes which have a lower content of paraffins and a higher content of naphthenes than the paraffin-base crudes, the content of asphaltic compounds is higher, the sulphur content usually is under 0.4 per cent and the paraffin-wax content is generally high. Mixed-base crudes occur in the mid-continent region. The naphthene-base crudes contain a high percentage of naphthenes, very little paraffin wax, and have a relatively high specific gravity. They occur in the central, south-central and south-western areas of the United States. Light naphthene-base crudes contain a low proportion of asphalt, compared to reverse proportions in heavy naphthene-base crudes. The sulphur content varies widely. The aromatic crudes, which occur chiefly in California, generally have a high asphaltic-compound content, sulphur content varying from 0.1 to 4.13 per cent and a relatively high nitrogen content. The specific gravity is often fairly low and the presence of wax widespread, although some crudes of this class are free of wax.

Crude oil is delivered to the refineries by rail, inland and intercoastal waterways, in specially constructed tanks, and by pipeline. In 1915 there were less than 20,000 miles of crude-oil trunk lines, which had been expanded in 1950 to nearly 65,000 miles to handle more than 86 billion ton-miles of crude oil and nearly 14,000,000 ton-miles of refined oil.

Grades of Petroleum Used as Fuels—Fuel oils may be classified generally as: (1) raw or natural crude petroleum, (2) distillate fuel oils, (3) residual fuel oils, and (4) blended oils. The increasing demand for gasoline and other petroleum products makes it very undesirable that crude petroleum as obtained from the wells be used for fuel. Besides, gasoline in a fuel oil is dangerous on account of the increased danger of explosions its presence entails. By the older methods of refining, the products from many of the oil refineries west of the Mississippi River were gasoline, naphtha, kerosene and fuel

oil, while eastern refineries usually carried the fractionation of oil much further, their output being such products as gasoline, benzene, naphtha, kerosene, light machine oil, automobile oils, cylinder oils, paraffin wax and tar, pitch, or coke. Recent improvements in thermal cracking at both high and low pressure and the use of catalytic conversion processes have enabled refiners to convert more of the petroleum to gasoline and to produce lubricants from western petroleum relatively high in asphalt.

Distillate fuel oils consist of the fractions distilled intermediate between kerosene and lubricating oils. Residual fuel oils are the viscous residual products remaining after the more volatile hydrocarbons have been driven off in the refining process. Blended oils are mixtures of any or all of the other three classes of fuel oils.

Properties and Specifications of Liquid Fuels—Before discussing the more important properties and specifications of fuel oil, some of the common terms will be reviewed.

Specific gravity is the ratio of the weight of a volume of a body to the weight of an equal volume of some standard substance. In the case of liquids, the standard is water. **Baumé gravity** is an arbitrary scale for measuring the density of a liquid, the unit being called "Baumé degree." Its relation to specific gravity is shown by the formula:

$$Be^{\circ} = \frac{140}{Sp. Gr.} - 130 \text{ (for liquids lighter than water)}$$

For example, the Baumé hydrometer will read 10° Bé in pure water, when the specific gravity scale reads 1.000. The American Petroleum Institute (API) Gravity is a modification of the Baumé scale for light liquids. API gravities are always reported at 60° F. The relation between API gravity and specific gravity is:

$$^{\circ}API = \frac{141.5}{Sp. Gr.} - 131.5$$

The greater the degrees Baumé or API, the lighter the fluid. There are about 90 API degrees between the heaviest and lightest oils which, therefore, makes this scale valuable for determining differences between the density of various oils.

Flash point is the lowest temperature at which, under definite specified conditions, a liquid fuel vaporizes rapidly enough to form above its surface an air and vapor mixture which gives a flash or slight explosion when ignited by a small flame. It is an indication of the ease of combustion or of the fire hazard in handling or using oil.

Pour point is the lowest temperature at which oil will pour or flow when chilled without disturbance under specified conditions.

Viscosity is the property of liquids that causes them to resist instantaneous change of shape or the arrangement of their parts due to internal friction. Since this property has a direct relation to resistance of flow in fuel-oil pipe systems and to atomization, it is an important specification. **Absolute viscosity** is a measure of internal fluid friction. It is expressed in the English system as pounds per second per foot and is the force which will move one square foot of a plane surface with a speed of one foot per second relative to another plane surface from which it is separated by a layer of the liquid one foot thick. The viscosity of a fluid relative to water at 68° F is the ratio of its absolute viscosity to that of water at 68° F (both absolute viscosities must be in the same units). This relation is called **relative viscosity**. The absolute viscosity of water at 68° F is 0.000672 lbs. per sec. per ft. or 0.0100 gms. per sec. per cm. which is equal to

Table 3—XIII. Properties of Typical Liquid Fuels

Fuel	Ultimate Analysis of Fuel (%)							Undetermined
	H ₂ O	C	H ₂	N ₂	O ₂	S	Ash	
Pitch-Tar (Dry).....		90.78	5.35	1.39	1.65	0.61	0.22	
Pitch-Tar (Natural Basis)....	1.33	89.57	5.28	1.37	1.63	0.60	0.22	
Bunker "C" Fuel Oil (Dry)*..		88.60	10.50	0.30	0.00	0.55	0.05	

Fuel	Specific Gravity at 60°F.	Weight (Lbs./Gal.)	Calorific Value (Btu/Lb.)		Dry Air Required for Combustion (Cu.Ft./Lb.)	Theoretical Flame Temp. (°F)
			Gross	Net		
Pitch-Tar (Dry).....						
Pitch-Tar (Natural Basis)	1.199	9.9855	16,155	15,674	158.13	3195
Bunker "C" Fuel Oil (Dry)*.....	0.9529	7.935	18,890	17,820	180	3800±

*Courtesy of Sun Oil Co.—(Typical Analysis).

0.0100 poises or 1.00 centipoise. The viscosity of all liquids decreases with increasing temperature and ASTM viscosity determinations are made at oil temperatures of 100° F, 122° F, 130° F and 210° F, and are often expressed as Saybolt Universal at 100° F or Saybolt Furol at 122° F. The terms "Saybolt Universal" and "Saybolt Furol" represent the type of instrument used in making the viscosity determinations. Viscosity measurements made by either may be interconverted by the use of tables.

Reid Vapor Pressure is a test for the vapor pressure of gasoline at 100° F. It shows the tendency of gasoline to generate vapor bubbles and is expressed in pounds per square inch absolute.

Octane number is the anti-knock rating of gasoline. The rating is made by matching the fuel in a test engine with a mixture of normal heptane, which detonates very easily and has an octane rating of zero, and iso-octane, which has exceptionally high anti-knock characteristics and is rated at 100. A fuel knock that matches a mixture of say 60 per cent octane and 40 per cent heptane would have an octane rating or number of 60. **Cetane number** is used to show the ignition quality of Diesel oils. The rating is based on a scale resembling those of octane numbers by matching the ignition delay of the fuel against blends of cetane, a fast-burning paraffin, and methyl naphthalene, a slow-burning aromatic material.

The A.S.T.M. has developed a table for grading fuel oils, consisting of six grades. According to this classification, heating oils generally used for domestic and small industrial heating furnaces comprise Grades 1, 2 and 3. Grades 5 and 6 correspond to **Bunker "B"** and **Bunker "C"** fuel oils, respectively, which are used extensively in the steel industry. **Bunker "B"** fuel oil is usually cracking-still tar and Grade 6, or **Bunker "C"** fuel oil, a straight-run or cracked residual, or a mixture of residual and cracking-still tar blended to reduce the viscosity to that required by the consumer.

All grades of fuel oil are normally sold to meet specifications mutually satisfactory to buyer and seller. A typical specification of **Bunker "C"** fuel oil for use in open-hearth furnaces is as follows:

Gravity—API (60° F)13.0 to 16.0
Flash point (closed cup)150° F minimum
Btu per gallon150,000 minimum
Sulphur0.75 to 1.00 maximum
Viscosity at 122° F100 secs. Saybolt Furol
Water and sediment (% by vol.) .1% maximum

The yield of tar produced in by-product coke ovens by

high-temperature distillation between 1832 and 2012° F (1000 to 1100° C) differs within very wide limits according to the kind of bituminous coal coked, and to the temperature, coking time, and design of oven employed in the process. Virgin tar as produced in the by-product ovens consists essentially of tar acids, neutral oils which are principally aromatic hydrocarbons, and a residuo pitch.

The residue pitch from the distillation of tar is highly viscous or brittle. Pitch contains a substantial percentage of free carbon and some high-boiling and complex organic chemicals. The properties of a typical pitch-tar mix and **Bunker "C"** fuel oil are shown in Table 3—XIII.

The viscosity of liquid fuels decreases with temperature increase. The range of typical samples of virgin tar, pitch-tar mixtures and topped tar is as follows:

Fuel	Test Temp. (°F)	Viscosity in Sec., Saybolt Universal
Virgin Tar	175	{ 189.4 max. 73.3 min. 109.4 avg.
Pitch-Tar Mix	175	{ 1940 max. 181 min. 946.1 avg.
Pitch-Tar Mix	210	{ 687 max. 97 min. 561.7 avg.
Topped Tar	200	{ 700 max. 550 min. 600 avg.

Combustion of Liquid Fuels—The combustion of liquid fuel usually is obtained by atomizing the fuel. Atomization breaks up the fuel into fine, mist-like globules, thus permitting an increased area for intimate contact between the air supplied for combustion and the fuel. The chemistry of combustion of liquid fuels is complex. The small particles of fuel either vaporize to form gaseous hydrocarbons which burn to CO₂ and H₂O through a chain of reactions, or the fuel cracks to form carbon (soot) and hydrogen which also burn with complete combustion to CO₂ and H₂O. Both of these condi-

tions normally occur in the combustion of liquid fuels. The first condition predominates with good atomization and proper mixing with sufficient air. A deficiency of air or poor atomization will cause smoke. For large furnaces, such as open hearths and heating furnaces, the atomizing agent is usually steam at a pressure anywhere between 60 and 125 pounds per square inch gage. The steam consumed in atomization varies from 0.3 to 0.7 pounds per pound of fuel. When liquid fuels are used in smaller furnaces, atomization usually is procured by compressed air or by mechanical action. The character of a liquid-fuel flame, that is, its shape, size and luminosity, may be altered with a fixed burner design by changing the degree of atomization which is controlled by the steam pressure. Liquid fuels normally are burned in steel plants to produce a highly luminous flame at an intensity of flame propagation intermediate between that generally secured with coke-oven gas and that with natural gas.

Liquid fuels usually are preferred above all other fuels for use in open-hearth furnaces because they permit better control of flame direction and, because of their high calorific value, control of flame temperature and luminosity.

The amount of air required to burn liquid fuels depends upon the chemical composition of the particular fuel. Bunker "C" fuel oil requires approximately 180 cu. ft. of dry air per pound of oil for perfect combustion, and tar-pitch approximately 158 cu. ft. From the ultimate analysis of a liquid fuel, the theoretical air requirements and products of combustion may be calculated, as explained in Section 3 under "Combustion Calculations." The composition and properties of tar-pitch are given in Table 3—XIII, shown earlier.

Liquid-Fuel Burners—There are many different designs of burners for liquid fuels. Burners designed for atomization by steam or air may be classified into two general types, the inside mixing and the outside mixing. In the inside-mixing type the fuel and atomizing agent are mixed inside the burner or burner system, while in

the outside-mixing type the two fluids meet immediately outside the burner tip. In open-hearth furnaces and large reheating furnaces the inside-mixing type is used. The inside-mixing type is sometimes classified as an emulsion type or a nozzle-mix type of burner. In the emulsion type the mixing is performed at a point several feet from the burner tip, while in the nozzle-mix type the two fluids meet inside the burner but very close to the burner tip. In the latter type, mixing probably takes place both inside the burner and as the stream enters the furnace.

Liquid-fuel burners used in open-hearth furnaces are water cooled due to port end design; those in reheating, forge and annealing furnaces seldom require water cooling.

The handling of liquid fuels at consuming plants requires a system for their transportation, storage and conditioning. Where liquid fuels are received by tank car, a system of receiving basins, unloading pumps, strainers and storage tanks generally is required. The storage tanks must be of ample size to meet fuel demands between deliveries and should be provided with heaters to maintain proper fluidity for flow through pipe lines to the system pressure pumps. Pressure pumps are used to deliver the liquid fuel through a pipe system to the point of consumption. Where there are a number of consuming units being served from a common fuel-storage system, the pipe feeder line is designed in the form of a loop through which the fuel flows at constant pressure and temperature. The various units tap into this loop. The fuel-oil lines are lagged and provided with tracer steam lines to maintain uniform fluidity throughout the system and to provide fuel at the burners at the proper viscosity for atomization. The temperature at which liquid fuel is delivered to the burners varies with the character of the fuel and burner design. Where pitch is used, a temperature as high as 300° F in the lines is sometimes required. A temperature level usually somewhere between 200° F and 250° F is maintained for pitch-tar mixtures, and 150° F to 200° F for Bunker "C" fuel oil.

SECTION 7

GASEOUS FUELS AND THEIR UTILIZATION

The availability of natural gas in so many sections of this country has had a profound influence upon our industrial progress and comfort. It was first used as an illuminating gas at Fredonia, New York, in 1824. The discovery of new fields and the installation of pipe lines to consuming centers led to increasing demands, as the convenience, cleanliness, and general utility of this form of fuel became better known. The initial use of natural gas for steel manufacture was at a rolling mill plant at Leechburg, Pa., in 1874. A well in this area permitted exclusive use of natural gas for puddling, heating, and steam generation for a period of six months. Since 1932 there has been an accelerated demand for natural gas.

In 1950, for example, 6,281 billion cubic feet of natural gas were produced. To meet this demand, the gas industry had installed some 265,000 miles of natural-gas pipe lines by the close of 1950.

Producer gas was the first gaseous fuel successfully utilized by the iron and steel industry. This gas permitted the early experimentation in regeneration, and the utilization of this principle started a new era of steel manufacturing. The advantages of preheated gas and air were so clearly indicated in 1861 that producer gas rapidly became the major fuel utilized by open-hearth furnaces

and maintained its position for almost sixty years, or until about 1920, when by-product coke plants, supplying coke-oven gas and tar, began to challenge this leadership.

Blast-furnace gas utilization by the iron and steel industry probably should rank first historically, although its adoption by the industry was slower than in the case of producer gas. The sensible heat in the blast-furnace top gases was first utilized in 1832 to transfer heat to the cold blast. Originally, this heat exchanger was mounted on the furnace top. In 1845, the first attempts were made to make use of its heat of combustion, but history indicates that the burning of blast-furnace gas was not successful until 1857. It is probable that progress in the utilization of blast-furnace gas was delayed by its dust content, the problems of cleaning and handling, and the low cost of solid fuel. Increasing cost of other fuels and competition forced its use, and by the turn of this century, blast-furnace gas had become one of the major fuels of the iron and steel industry. In 1951, the blast-furnace gas produced and consumed amounted to nearly 28 million tons of coal equivalent.

The initial use of by-product coke-oven gas in the iron and steel industry was at the Cambria Steel Company,

Johnstown, Pa., in 1894. This installation was followed by only a few by-product coke-plant additions until a shortage of transportation facilities and the rising price of coal and natural gas during the first World War accelerated installations throughout the steel industry. The utilization of coke-oven gas has been very profitable as it reduced the purchase of outside fuels. It is estimated that plants operating steel-making furnaces used 371,476,712,000 cubic feet of coke-oven gas as fuel in 1950.

Natural Gas—Natural gas and petroleum are related closely in their chemical composition and in geographical distribution. Both are made up predominately of hydrocarbons. Petroleum rarely is free of natural gas, and the same fields usually produce both fuels. When natural gas exists indigenous to an oil stratum and its production is incidental to that of oil, it is called **casinghead gas**. Gas found in a field is usually under pressure which diminishes with extended use or, sometimes, from the presence of too many other wells. The life of a well varies from a few months to twenty years. The depth of wells varies from 250 to 4000 or more feet. Rocks bearing gas are sandstones, limestones, conglomerates, and shales—never igneous rocks. Natural gas is derived from the remains of marine animal and plant life—in theory, the same as described previously for petroleum.

Natural gas as found is usually of singular purity and is composed principally of the lower gaseous hydrocarbons of the paraffin series, methane and ethane, some of the heavier liquefiable hydrocarbons (which are recovered as **casinghead gasoline** or sold in bottled form as butane, propane, pentane, etc.) and a small amount of nitrogen or carbon dioxide. Some natural gases contain small quantities of helium. Occasionally, wells are found in which the gas contains hydrogen sulphide and organic sulphur vapors. **Sour gas** is defined as a natural gas which contains in excess of $1\frac{1}{2}$ grains of hydrogen sulphide or 30 grains of total sulphur per 100 cu. ft. It is fortunate, however, that by far the greater part of natural gas available in this country is practically sulphur-free.

There are a number of great gas fields in this country; viz., the Appalachian, the Lima-Indiana, Illinois, Mid-Continent, Gulf, and California. Due to the increasing customer demand, exhaustion of wells, and the inability of the older local fields to adequately meet the demand, a number of the fields have been linked by pipe lines. Gas from the extensive Texas oil fields, which previously

had been allowed to escape on account of lack of pipe lines to carry the excess to more distant consuming centers, now is being utilized.

The principal constituent of natural gas is methane, CH_4 . Since natural gas contains from 60 to 100 per cent of CH_4 by volume, the characteristics of methane gas, which were shown in Section 3, largely dominate the parent gas. Comparing methane with the other principal combustible gases, it will be noted that it has a low rate of flame propagation, a high ignition temperature, and a narrow explosive range. Methane, as well as all other hydrocarbons (of which it is the lowest member), burns with a luminous flame.

Typical compositions of natural gas are presented in Table 3—XIV.

The iron and steel industry consumed 208,989 million cubic feet of natural gas in 1952, 206,797 million cubic feet in 1951, and 203,852 million cubic feet in 1950.

Manufactured Gases—The four most important manufactured gases are **producer gas**, **water gas**, **oil gas**, and so-called **bottled gases**. All are used domestically and industrially.

Producer gas may be made from coke, anthracite coal, or bituminous coal. When made from bituminous coal, the raw, uncleaned gas contains tarry deposits which render it unsuitable for pipe-line transmission. Hot, raw producer gas has wide successful application in industrial plants, principally those of the steel, ceramic, and glass industries, but is never used for domestic service. Cleaned, cooled and washed producer gas is produced by public utilities for both domestic and industrial use.

Water gas is made usually from coke or anthracite coal. It is relatively clean compared to producer gas and is used much more extensively for domestic purposes. It has gained great importance in the chemical industry in the past decade as a base for the production of synthetic organic chemicals. Water gas is often called **blue gas** and when enriched with cracked fuel oil it is called **carburetted water gas**.

Oil gas is a combination of cracked petroleum and water gas made by passing oil and steam through hot refractory checker work. Oil gas is commercially important in localities where coal or coke is expensive and oil cheap.

Bottled gases are liquefied petroleum gases. They have become commercially important because of the concentration of fuel energy in liquid form which may be

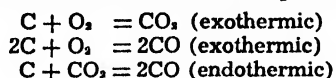
Table 3—XIV. Typical Composition of Natural Gas in Various Districts

Constituents	Districts				
	Birmingham	Pittsburgh	So. California	Los Angeles	Kansas City
CO_2			0.7	6.5	0.8
O_2					
N_2	5.0	0.8	0.5		8.4
CO					
H_2					
CH_4	90.0	83.4	84.0	77.5	84.1
C_2H_6	5.0	15.8	14.8	16.0	6.7
Total	100.0	100.0	100.0	100.0	100.0
Specific Gravity	0.60	0.61	0.64	0.70	0.63
Gross Heating Value— Btu/cu. ft.	1002	1129	1116	1073	974
Net Heating Value— Btu/cu. ft.	904	1021	1009	971	879

converted easily into a gas. They are distributed for household use in steel cylinders called "bottles" and in tank cars or trucks for industrial purposes. They are sometimes sold under various trade names but are composed mainly of butane, propane, and pentane. A steel cylinder of propane as sold for domestic purposes contains approximately 21,640 Btu per pound of liquid gas.

Manufacture of Producer Gas—Producer gas is manufactured by blowing an insufficient supply of air for complete combustion, with or without the admixture of steam, through a thick, hot, solid-fuel bed. A large proportion of the original heating value of the solid fuel is recovered in the potential heat of carbon monoxide, hydrogen, tarry vapors, and some hydrocarbons, and in the sensible heat of the composite gas which also contains carbon dioxide and nitrogen. When the gas is cleaned, the sensible heat of the gases and the potential heat of the tar vapors is lost. Originally, gas producers were blown only with air, and the gas produced had a heating value of only about 110 Btu per cu. ft. There was trouble with excessive clinker formation due to the exothermic reaction between the oxygen in the air and the carbon in the coal, and a large sensible-heat loss occurred in cleaning.

The basic reactions in an air-blown producer are:



Theoretically, air-blown producers may have 70 per cent gasification efficiency if all the carbon is converted to carbon monoxide (CO). The other 30 per cent is lost when the hot gases are cooled. If all the carbon is converted to CO, the gas would have a heating value of 112 Btu per cu. ft. The introduction of steam with the air, now the general practice, permits generating a gas of higher Btu value (130 to 190), reduces the temperature of the hot zones in the producer on account of the endothermic reaction between steam and hot carbon, and reduces the temperature of the gases leaving the producer.

The thickness and temperature of the zones, the velocity at which the gases move through them, the relation of the volume and ratio of the air and steam in the blast to the coal consumed, gas channeling, and the character of the coal, determine the gas composition and the optimum producer efficiency.

Two typical modern gas producers are shown in Figures 3-4 and 3-5. Figure 3-4 is a cross section of a modern Morgan Construction Company gas producer. The coal feed is regulated by temperature of the outlet gas or by a combination of the temperature reading and the rate of flow of air to the producer, which is metered for that purpose. Temperature of air blast is controlled automatically to maintain a constant weight of steam and air ratio entering the producer in order to maintain uniform firing conditions.

Figure 3-5 is a cross section of a modern Wellman mechanical gas producer equipped with bell-type feed. The producer is designed for application of modern controls for quantity and temperature of gas delivery. An important feature is its mechanically driven, oscillating poker. Air is delivered to the producer by a steam-driven turbo blower.

As shown by Figures 3-4 and 3-5, producers are constructed of steel plates and castings, the parts exposed to high temperatures being lined with refractories or water-cooled. They are made in different sizes and dimensions are varied accordingly. Each producer consists essentially of two separate parts, a combustion chamber and ash pan which are mounted upon rollers

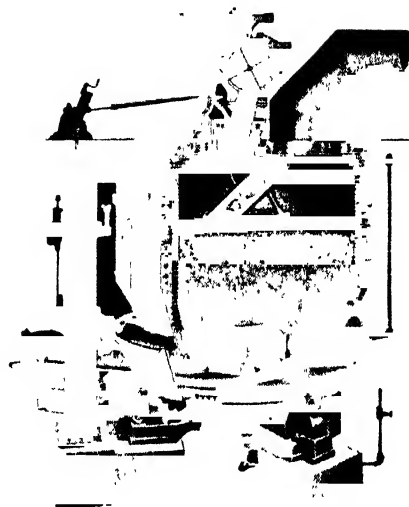


FIG. 3-4. Cross-section of a modern Morgan Construction Company gas producer.

so as to move at a speed of about 1/10 r.p.m., and a top which is stationary and held in place by steel supports. The body portion is lined with best quality firebrick and is constructed to provide water seals at both top and bottom, the top seal to prevent escape of gas and the bottom seal to provide a means of removing the ash formed. Removal of the ash is accomplished by constructing the bottom parts so as to deflect the ash toward the outer periphery of the pan and by providing a scoop or plow to lift the ash out of the pan at certain intervals as required. The top of the machine is water-cooled and contains the openings for the introduction of the fuel, the gas outlet, two or more observation holes, and an opening for the insertion of a poker or a fuel leveler. Some means of keeping the fuel bed at a uniform depth and of distributing or leveling the raw coal is essential, but de-

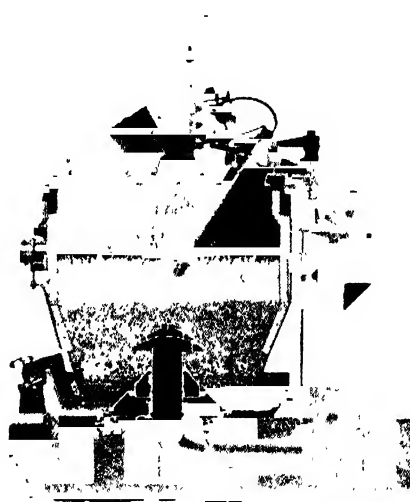


FIG. 3-5. Cross-section of a modern Wellman mechanical gas producer with bell-type feed.

sign differs as to the method, some using agitation with a poker and others merely a leveler that leaves the fuel bed undisturbed. An essential part of the producer is the blower which is built into the bottom. It serves as a means for introducing the air and steam required for the partial combustion of the fuel and for distributing these so as to obtain uniform combustion over the entire bottom of the fuel bed. There are two types of blowers: the steam-jet or injector type, and the turbo blower. In the former, steam is the only propelling agent, while in the latter, a small steam turbine is directly connected to a blower which forces the air and exhaust steam from the turbine into the producer. The steam-jet type is the cheapest to operate, but formerly it gave poor control because of condensation. Improvements in jets have overcome many of their drawbacks. Supplementary equipment required for efficient operation of the machine include: (1) a boiler to supply steam; (2) electric motors to rotate the machine, operate the fuel feed, and the poker, if the machine is so equipped; (3) a bin located directly above the producer for the storage of the fuel; and (4) a dust catcher and shut-off valve, built adjacent to the machine, to collect dust and soot carried out of the producer with the gas.

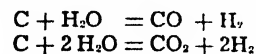
The gasification rate of producers varies. Hand-poked machines gasify 10 to 15 lbs. of coal per sq. ft. of fuel bed area per hour. Modern mechanically-poked machines will gasify 50 to 75 lbs. per sq. ft. per hour with good bituminous coals, and rates as high as 90 lbs. have been obtained. The gasification rate with coke breeze is about half that of bituminous coal. The air blown per pound of fuel varies from 45 to 53 cu. ft. depending upon the grade of fuel; about 52 cu. ft. is blown for low-ash high-volatile coal. The steam used per pound of fuel varies from 0.17 to 0.40 lb. depending upon the amount and fusion point of the ash, and is about 0.20 lb. for a high-volatile coal of low ash content with a high ash-fusion point. The efficiency of conversion, per cent total calorific value of fuel recovered, is 88 to 90 per cent when raw gas is consumed hot, and 65 to 80 per cent when the gas is cooled and cleaned.

Table 3—XV gives the composition of clean producer gas made from various fuels in a well-operated updraft producer.

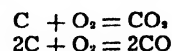
The gross heating value of raw producer gas, including tar, made from a high-volatile coal, 8 per cent ash, is 170 to 190 Btu per cubic foot.

Producer gas has a very low rate of flame propagation due to the relatively large amount of inert gases, N_2 and CO_2 , it contains. The hot gas, containing tar, burns with a luminous flame; the cold gas is only slightly luminous, while it is non-luminous if made from anthracite coal or coke. Producer gas is a relatively heavy gas and has a wide explosive range. The theoretical flame temperature is low, approximately $3175^\circ F$, and the gas generally is preheated when utilized in steel-plant processes.

Manufacture of Water Gas—Water gas or blue gas is generated by blowing steam through an incandescent bed of carbon. The gas forming reactions are primarily:



Since both of these reactions are endothermic, the temperature of the bed of carbon through which the steam is blown would be lowered quickly to a point where no reaction would occur, if proper control was not provided. The temperature of a bed of hot carbon may be increased by blowing air through it, causing the two basic exothermic reactions:



By controlling the time during which air is blown, the temperature level of the bed can be controlled and the effect of the second reaction, which consumes more carbon and generates less heat, is minimized. The air blow raises the temperature level of the bed, and the gases formed during this part of the process pass through a stack to the outside air. The steam run lowers the temperature level of the bed and generates water gas which is collected by passing the gases through outlets to a gas holder, the reservoir for distribution. The character of the gas formed and the amount of steam decomposed during the steam run are largely functions of the bed temperature and thickness, and velocity of gases through the bed. In order to maintain a practical limitation to the rise and fall of the temperature level, the length of each cycle is of short duration, usually from 2 to 6 minutes. The thickness of the fuel bed is maintained by charging fresh fuel periodically and removing ash from fuel spent in the process. Charging usually is done every 45 to 75 minutes and the ash removed every 6 to 8 hours. The velocity of the gases is controlled by the rate of steam input and fuel sizing. In order to maintain as nearly uni-

Table 3—XV. Composition of Clean Producer Gas^u

Constituent	Solid Fuel Feed				
	Anthracite Coal	Coke		Bituminous Coal	
		1 to 5" Lump	Breeze	A	B
CO_2	6.3%	9.2%	8.7%	3.4%	9.2%
Illuminants	0.0	0.1	0.0	0.8	0.4
O_2	0.0	0.0	0.0	0.0	0.0
CO	25.0	21.9	23.3	25.3	20.9
H_2	14.2	11.1	12.8	9.2	15.6
CH_4	0.5	0.2	0.4	3.1	1.9
N_2	54.0	57.5	54.8	58.2	52.0
Total	100.0%	100.0%	100.0%	100.0%	100.0%
Gross Heating Value, Btu per cu. ft.	132	121	131	155	156

^uU. S. Bureau of Mines, Bulletin 301.

form temperature as possible in the fuel bed (about 1650° F) and thereby to secure the optimum gas yield and least trouble from ash clinkers, the steam run during which water gas is produced, also called the gas run, is divided into three distinct periods, known as the up-run, the down-run, and the final up-run. These terms designate the direction of the steam-gas movement. In the up-run the steam is admitted below the fuel bed and taken off at outlets above the bed. The down-run is the exact opposite. A typical four-minute water-gas cycle is as follows:

Air Blow		90 seconds—13,000 cu. ft. of air per min. (100–150 c.f.m. per sq. ft. of grate area)
Steam Run	Up-run	44 seconds—175 lb. steam per minute
	Down-run	90 seconds—200 lb. steam per minute
	Up-run	16 seconds—175 lb. steam per minute
	Total	240 seconds

During the air blow period of each cycle, a hot, lean producer gas is formed. A part of this gas sometimes is permitted to dilute the steam-run gas. The latter part of the air blow generates a gas generally high in CO, and is the most efficient period in which to collect this gas. Normally, however, all of the gas made during the air blow is passed through a separate chamber containing refractory checker work and the CO in the gas burned with air diverted from the generator to this chamber. The checkers absorb some of the heat of the spent gases as they pass to the stack. The down-run steam sometimes is passed through the checker chamber before entering the generator to recover the stored heat. Heat in a similar chamber is used for cracking fuel oil when carburetted water gas is made.

The division of each cycle into definite periods requires an elaborate system of valving to direct properly the flow of air, steam and gases, the mixing of which would lead to dilution and, in some cases, to serious explosions. The sequence and time in which the large number of valves must be opened and closed are controlled automatically, the valves themselves being operated either electrically or hydraulically in response to the controls. Since cycles are frequently changed as conditions warrant, the controls are designed for easy alterations to the timing.

While coke generally is used as the fuel in the production of water gas because of its high carbon content and cleanliness, anthracite and bituminous coal and mixtures of coal and coke also have been used successfully, but with some sacrifice in over-all operating efficiency.

Water gas burns with a clear blue flame; hence, the name "blue gas." It is used in a number of chemical processes to supply a basic gas for synthetic processes, but it is not suitable for distribution as a domestic fuel unless it has been enriched with cracked fuel oil, when it is called carburetted water gas.

Water gas made from coke burns with a non-luminous flame. Carburetted water gas burns with a highly luminous flame. Both gases have a high rate of flame propagation. The speed of combustion for water gas exceeds that of any other extensively used fuel gas; that for carburetted water gas is practically the same as for coke-oven gas. Water gas has a slightly lower specific gravity than natural gas, but is somewhat heavier than coke-oven gas. Carburetted water gas is heavier than natural gas but lighter than producer gas. The theoretical flame temperature of both blue and carburetted water gas is very high, respectively about 3670 and 3725° F, exceeding that of all other industrial fuel gases commonly used. Both gases have a relatively wide explosive range.

Special Gas Processes—Studies and investigations in

the art of gasification have been under way in the United States for a number of years. The German demand for self-sufficiency prior to and during the second World War led to developments of the water-gas machine which first made possible continuous gasification, and later the manufacture of a gas containing methane. Technical advances in the manufacture of cheap oxygen contributed largely to these developments. The first step towards improvements in fuel-gas manufacture eliminated the disadvantage of cyclic operation and the second, the necessity for carbureting water gas. Continuous gasification was accomplished in the **Winkler Process** by passing the proper mixture of steam and oxygen through a fuel bed to compensate for the endothermic reaction, $C + H_2O$, with the exothermic reaction, $C + O_2$, thereby maintaining a fixed temperature in the fuel bed.

The **Lurgi Process**, developed in 1936, produces a rich water gas containing CH₄. A mixture of steam and oxygen at 20 atmospheres pressure (steam at 900° F) is introduced through the hollow shaft of a rotating grate and passes through the ash bed into a fixed fuel bed, which is at a temperature of 2150° F, where the fuel is gasified. Fuel is charged intermittently into a hopper immediately above the generator proper. A foot valve between the hopper and generator locks the high pressure in the generator during charging. After filling the hopper, fuel is fed continuously into the generator chamber. A plow directs ash through the grate into a small compartment in the lower portion of the generator, and from there the ash is removed continuously by a revolving scraper into an outside ash hopper.

By-Product Gaseous Fuels—The two major by-product gaseous fuels are blast-furnace and coke-oven gases. A number of other unavoidable gaseous fuels are created by regular manufacturing processes. Some of these are of minor economic consequence, but the majority are useful and generally utilized at the plant where they are produced. An exception is oil-refinery gas which is sometimes piped and marketed to industries adjacent to refineries. The calorific value and flame characteristics of by-product gases have wide ranges. Blast-furnace gas has probably the lowest heat content of any, and oil refinery gas the highest, respectively 90 and 1850 Btu per cubic foot, although both vary from these values.

Blast-furnace gas is a by-product of the iron blast furnace. The paramount objective in blast-furnace operation is to produce iron of a specified quality, economically; the fact that usable gas issues from the top of the furnace is merely a fortunate attendant circumstance. When air enters the tuyeres (see Chapter 12 on "The Manufacture of Pig Iron") its oxygen reacts with the coke. The resulting gas passes up through the shaft of the furnace which has been charged with coke, ore, and limestone, and after a number of chemical reactions and a travel of some 80 feet, issues as a heated, dust-laden, lean, combustible gas. The annual volume production of this gas is greater than that of any other gaseous fuel. Four to five tons of blast-furnace gas are generated per ton of pig iron produced. While the purpose of the gases generated by the partial combustion of carbon is to reduce iron ore, the value of a blast furnace as a gas producer is evident from the relation just noted. The essential reactions by which blast-furnace gas is produced are shown in Chapter 12.

The percentage of CO and CO₂ in blast-furnace gas is directly related to the amount of carbon in the coke and the amount of CO₂ in the limestone charged per ton of iron produced. The rate of carbon consumption depends principally upon the kind of iron to be made, the physical and chemical characteristics of the charged material, the distribution of the material in the furnace stack, the

furnace lines, and the temperature of the hot blast. The total CO + CO₂ content of the top gas is about 40 per cent by volume, and when producing ordinary grades of iron the ratio of CO to CO₂ will vary from 1.7 to 2.5, to 1. The hydrogen content of the gas varies from 3 to 4 per cent. The remaining percentage is made up of nitrogen, except for about 0.2 per cent CH₄. The efficiency of the blast furnace as a gas producer on a cold basis is a little less than 70 per cent; and on a hot basis, about 90 per cent.

Blast-furnace gas leaves the furnace at a temperature of 250° to 700° F, and at a pressure of 15 to 60 inches w.g., carrying with it 10 to 50 grains of water vapor and 8 to 15 grains of dust per cubic foot. The particles of dust vary from 1/4 to 0.00001 inch in diameter. In early days of blast-furnace operation, the gas was used as it came from the furnace without cleaning, causing a great deal of trouble with flues, combustion chambers, and stoves due to clogging. The gas now is cleaned almost universally, the degree depending upon the use.

The outstanding characteristics of blast-furnace gas as a fuel are: (1) very low calorific value—approximately 90 Btu per cu. ft., (2) low theoretical flame temperature—approximately 2650° F, (3) low rate of flame propagation—relatively lower than any other common gaseous fuel, (4) high specific gravity—highest of all common gaseous fuels, and (5) burns with a non-luminous flame.

Coke-Oven Gas—The steel industry, which uses about 85 per cent of the total coke-oven gas generated in the United States, generally classifies coke-oven gas as a by-product of coke manufacture. This undoubtedly is due to the former waste of coke-oven gas and other coal products for so many years in the beehive-coke process. Actually, the production of coke-oven gas and other coal chemicals is a part of an important manufacturing process in which large sums have been expended for their recovery, as they have a value almost equal to that of the coke. Coke-oven gas is produced during the carbonization or destructive distillation of bituminous coal in the absence of air. A description of the carbonization of coal and the recovery of coke-oven gas and coal chemicals is given in Chapters 4 and 5.

The composition of coke-oven gas varies in accordance with grade and density of coal and operating practices. Typical ranges for the constituents of dry coke-oven gas by volume are as follows:

CO ₂ *	1.3 - 2.4
O ₂	0.2 - 0.9
N ₂	2.0 - 9.6
CO	4.5 - 6.9
H ₂	46.5 - 57.9
CH ₄	26.7 - 32.1
Illuminants	3.1 - 4.0
Specific Gravity	0.36-0.44
Heating Value, Btu per cu. ft. (gross) ..	537-580
Heating Value, Btu per cu. ft. (net) ...	480-523

*Includes H₂S

Coke-oven gas contains hydrogen sulphide, H₂S. About 40 per cent of the sulphur in coal, not removed in the washing process, is evolved with the distillation products. Much of this remains in the gas. Carbonization of coals containing 1.20 per cent sulphur evolves a gas containing about 424 grains of sulphur per 100 cu. ft., and those containing 1.60 per cent sulphur about 600 grains per 100 cu. ft. Commercial coals in the eastern part of the United States usually run from 0.5 to 1.5 per cent sulphur. Gases high in sulphur content are very undesirable for metallurgical purposes.

Coke-oven gas normally is saturated with water vapor.

In distribution systems, means must be provided for draining off the condensation due to any temperature change.

Coke-oven gas burns with a non-luminous to semi-luminous flame, depending upon the degree of mixing air and gas. Its rate of flame propagation is high—considerably higher than natural, producer, or blast-furnace gas. It has a low specific gravity—lowest of any of the gaseous fuels commonly utilized by the steel industry. It has a high theoretical flame temperature—about 3600° F, a little higher than that of natural gas. The explosive range is about twice that of natural gas.

Use of Various Gaseous Fuels in the Steel Industry—Gaseous fuels are ideal for many steel-plant applications. Below are the more important applications where gaseous fuels either must be used on account of the nature of the work or facility, or where they are preferred over a liquid or solid fuel:

- Coke-Oven Heating
- Blast-Furnace Stoves
- Gas Engines for Blowing or Power Generation
- Soaking Pits
- Reheating Furnaces
- Forge and Blacksmith Furnaces
- Normalizing and Annealing Furnaces
- Controlled-Cooling Pits
- Foundry Core and Bessemer Bottom Ovens
- Blast Furnace and Steel Ladle Drying
- Drying of Blast-Furnace Runners and
- Open-Hearth Tapping Spouts
- Hot-Top Drying

To these may be added the gas turbine for blowing or power generation when further development permits a more extended use of this machine.

The choice of the most desirable fuel for each of the many facilities in a steel plant is not always possible, but by judicious planning the most efficient fuel or combination can be selected from those available. The general characteristics of each gas govern, wherever possible, its selection for a specific purpose in a steel plant. An outline of the important applications of the major gaseous fuels follows.

Uses for Blast-Furnace Gas—For many years, the use of blast-furnace gas for purposes other than for the firing of stoves and boilers was not economical. A number of factors have contributed, however, to the enlarged use of blast-furnace gas, the more important of which are: (1) rising cost of purchased fuel; (2) technical progress in gas cleaning, in the use of regeneration and recuperation, and in the mixing of gaseous fuels; (3) the economic advantage of using pulverized coal in boiler houses to substitute for blast-furnace gas, thereby permitting its substitution elsewhere for the more expensive liquid and gaseous fuels; and (4) seasonal shortages in the availability of purchased liquid and gaseous fuels.

In certain applications, in addition to preheating the air, the gas itself may be preheated to provide higher temperature potential. For the facilities listed below, blast-furnace gas may be utilized successfully without preheat:

- Blast Furnace Stoves
- Normalizing and Annealing Furnaces
- Foundry Core and Bessemer Bottom Ovens
- Gas Engines for Blowing or Power Generation

The thermal advantage of using blast-furnace gas in blowing engines and gas electric generators must overcome the heavy investment and maintenance expense of this equipment. The modern boiler house utilizing

high steam pressure and temperature with efficient turboblowers and generators has sufficiently reduced the thermal advantage of gas engines so that their use is difficult to justify.

Preheated blast-furnace gas burned with preheated air has been used successfully in the following applications.

Coke-Oven Heating Soaking Pits Reheating Furnaces

When blast-furnace gas is preheated, it should have a minimum cleanliness of 0.01 grains per cu. ft.; and in all cases where this gas is used, extra precautions must be taken to prevent the escape of fuel or unburned gas into attendable surroundings since it contains a large percentage of toxic CO gas. Blast-furnace gas is used for many applications in the steel plant and, in addition, is used frequently for heating coke ovens and sometimes is mixed with other gases as an open-hearth fuel.

Use of Coke-Oven Gas—Coke-oven gas has had a more extended use than blast-furnace gas because of: (1) relatively low distribution costs due to its low specific gravity, high calorific value, and cleanliness; (2) its ability to develop extremely high temperatures by combustion; and (3) the high rate at which it can release heat, thereby eliminating excessively large combustion chambers. Important applications for coke-oven gas include open-hearth furnaces in addition to those previously listed for gaseous fuels. The low specific gravity of coke-oven gas is a disadvantage in the open hearth, and for this reason, it is supplemented wherever possible with a driven liquid fuel in this service. In addition, the sulphur (in the form of H_2S) present in coke-oven gas is a distinct disadvantage, particularly when used in making low-sulphur heats in the open hearth and in heating certain grades of alloy steel for rolling. Its presence also requires the use of materials resistant to sulphur attack in pipe lines, valves, and burners.

There are a number of fuel applications in a steel plant where neither blast-furnace gas nor coke-oven gas, when burned alone, develop the desired flame characteristics or temperature level for optimum results. By mixing two fuels of such great variance in characteristics, a more ideal fuel can be obtained for specific applications.

The speed of combustion is very high for coke-oven gas and very low for blast-furnace gas. The desired speed can be attained through the proper proportioning of the two fuels. The speed also can be modified to a limited extent when necessary by suitable combustion technique. Mixed blast-furnace and coke-oven gas is particularly suitable for application to soaking pits and reheating furnaces.

Use of Natural Gas—Due to plant balances requiring the purchase of outside gaseous fuels, mixtures of coke-oven gas and natural gas are often utilized. While the temperature-developing characteristics of these two gases are nearly identical, they have differences in other characteristics, notably in the rate of flame propagation and in luminosity. By proper proportioning, the advantage of a short, intensive cutting flame or a long, lumi-

nous, soft flame may be had to suit the applications.

Use of Producer Gas—Raw, hot, producer gas has been used extensively in steel-plant operations for open-hearth furnaces, soaking pits, and reheating furnaces. It has been customary to preheat this gas regeneratively when used in open hearths and soaking pits, and also in batch-type reheating furnaces. In continuous-type reheating furnaces, the fuel seldom is preheated. With good gas making, producer gas develops a soft, heavy, long, luminous flame desirable for reheating steel and in working an open-hearth heat. The use of this gas largely has been superseded in many plants by by-product gaseous and liquid fuels.

Combustion of Various Gaseous Fuels—The major combustion reactions of the components of gaseous fuels with air and a table of essential gas combustion constants were given in Section 3 of this chapter. From chemical equations, the quantity of air required to provide perfect combustion and the resultant products may be calculated for any given gaseous fuel. Table 3—XVI shows the air requirements, products of combustion, and pertinent characteristics of several gaseous fuels. The degree of mixing of air with a gaseous fuel, and the degree of excess or deficiency of air to the theoretical requirements are pertinent combustion problems. The degree of mixing is controlled by burner design. Burners have been developed to produce short, intense flames or long, slow-burning flames. The short, intense flame is usually non-luminous or semi-luminous, while the long flame is luminous. This relation is not always the case, however, since a gas must contain hydrocarbons to develop luminosity. Burners capable of producing short, intense flames will liberate a large amount of heat in a small space. Some gases, due primarily to the constituents of which they are composed, are capable of a high rate of heat release; others, of a very low rate of heat release. The two extremes are evident in two common steel-plant fuels—coke-oven gas and blast-furnace gas. There is also a limit to the length of flame which can be produced. It is determined by the ability of the flame to provide enough heat to propagate itself. If the short, intense-flame type burner is used with coke-oven or natural gas, combustion will be so intense that no flame will be visible, and heat can be liberated at rates up to several million Btu per cubic foot of combustion space per hour; while the long, slow-burning-flame burner firing the same gases is capable of developing a visible flame twenty or thirty feet long with a heat liberation of 15,000 to 20,000 Btu per cu. ft. per hour. Both types of flames are desirable for specific steel-plant applications. It is obvious that burner selection based on degree of mixing is important. Carrying an excess or deficiency of air for combustion is practiced usually to control scale formation, but this is done sometimes in order to control flame characteristics. An excess of air tends to shorten, while a deficiency lengthens, a flame. An excess of air above theoretical requirements causes higher heat losses as any extra air absorbs its share of the heat of combustion. When there is a deficiency of air, potential heat is lost. In problems of design and fuel conservation, the air requirements and volume and constituents of the products of combustion must be known.

SECTION 8

FUEL ECONOMY

Since fuel represents the largest single item of expense in assembling raw materials for the manufacture of iron and steel, the subject of fuel economy is of consequence

to both the producer and consumer of steel products. The steel industry consumes annually during normal times about 100 million net tons of fuel in coal equivalent.

Table 3—XVI. Properties of Typical Gaseous Fuels^(a)

Fuel Gas	Constituents of Fuel Gas Per Cent by Volume (Dry Basis)									Specific Gravity	Cu. Ft. of Air Required for Combustion of Cu. Ft. of Gas	Heating Value Btu per Cu. Ft. of Gas	
	CO ₂	O ₂	N ₂	CO	H ₂	CH ₄	C ₂ H ₆	Illuminants					
								C ₂ H ₄	C ₃ H ₈				
Natural Gas.....			0.8			83.4	15.8			0.61	10.58	1129	1021
Reformed Natural Gas	1.4	0.2	2.9	9.7	46.6	37.1		1.3	0.8	0.41	5.22	599	536
Coke-Oven Gas.....	2.2	0.8	8.1	6.3	46.5	32.1		3.5	0.5	0.44	4.99	574	514
Water Gas (Coke)....	5.4	0.7	8.3	37.0	47.3	1.3				0.57	2.10	287	262
Carburetted Water Gas.....	3.0	0.5	2.9	34.0	40.5	10.2		6.1	2.8	0.63	4.60	550	508
Oil Gas (Pacific Coast)	4.7	0.3	3.6	12.7	48.6	26.3		2.7	1.1	0.47	4.73	551	496
Producer Gas (Bituminous Coal)...	4.5	0.6	50.9	27.0	14.0	3.0				0.86	1.23	163	153
Blast Furnace Gas....	11.5		60.0	27.5	1.0					1.02	0.68	92	92
Butane (Commercial)				(C ₄ H ₁₀ —93.0)	(C ₃ H ₈ —7.0)					1.95	30.47	3225	2977
Propane (Commercial)				(C ₃ H ₈ —100.0)						1.52	23.82	2572	2371

Table 3—XVI. (Continued)

Fuel Gas	Products of Combustion in Cu. Ft. per Cu. Ft. of Fuel				Ulti- mate % CO ₂	Net Btu per Cu. Ft. of Products of Com- bustion	Theor. Flame Temp. No Excess Air °F
	H ₂ O	CO ₂	N ₂	Total			
Natural Gas	2.22	1.15	8.37	11.73	12.1	87.0	3562
Reformed Natural Gas	1.30	0.53	4.16	5.99	11.3	89.6	3615
Coke-Oven Gas.....	1.25	0.51	4.02	5.78	11.2	87.0	3610
Water Gas(Coke)....	0.53	0.44	1.74	2.71	20.1	96.6	3670
Carburetted Water Gas.....	0.87	0.76	3.66	5.29	17.2	96.2	3725
Oil Gas(Pacific Coast)	1.15	0.56	3.77	5.48	12.9	90.5	3630
Producer Gas (Bituminous Coal)	0.23	0.35	1.48	2.06	18.9	74.6	3175
Blast Furnace Gas ..	0.02	0.39	1.14	1.54	25.5	59.5	2650
Butane (Commercial) ..	4.93	3.93	24.07	32.93	14.0	90.5	3640
Propane (Commercial)	4.17	3.00	18.82	25.99	13.7	91.2	3660

^(a)From "Combustion"—American Gas Association.

The efficient utilization of this large quantity of fuel is also pertinent to the conservation of our fuel resources. The history of the steel industry shows great progress has been made in reducing the amount of fuel required to produce a ton of steel. During the Revolutionary War, iron making required large quantities of charcoal, as the source of carbon, to reduce the ore. If a substitute had not been found for charcoal, our forests would have disappeared many years ago and our industrial progress arrested. In the past one hundred years, which really represents the modern era of steelmaking, a number of important developments have taken place to reduce the fuel requirements in producing steel. Some of these developments could be listed by historical sequence, while others are of such a nature that they cannot be designated by any period of time. The major contributions to fuel economy in steel plants have been:

- (1) Development of the Bessemer converter.
- (2) Development of the Siemens-Martin regenerator.
- (3) Development of the hot blast.
- (4) Utilization of blast-furnace gas.
- (5) Installation of by-product coke plants and utilization of by-product fuels.
- (6) Integration of steel plants.
- (7) Electric drives for rolling mills.
- (8) Improved efficiency of steam-generating equipment and steam prime movers.
- (9) Large producing units.
- (10) Balancing of producing units.
- (11) Recovery of waste heat by recuperators, boilers and other forms of heat exchangers.
- (12) Development and utilization of instruments and control equipment.
- (13) Insulation of high-temperature facilities.
- (14) Utilization of the optimum fuel for specific facilities.
- (15) Improvements in manufacturing technique and production control.
- (16) More highly skilled operators.

The results of the above contributions now have made it possible to produce a ton of steel ingots utilizing less than 1½ tons of coal, instead of several tons as required a hundred years ago. The consumption of primary fuels in the iron and steel industry for three recent years is shown in Table 3—XVII.

In addition to these outstanding contributions to fuel economy in steel mills, the importance of the effect that rate of operations has on fuel economy should be stressed. Historically, the iron and steel industry follows the general business level maintained in the country, but its rate of operations often fluctuates more than that of many other industries. During peak production, optimum fuel economy is the natural result of operating the facilities which require fuel under the conditions for which they were designed to operate most economically. During periods of low production, fuel consumption undergoes a severe increase per unit of output; careful scheduling of production and facilities are required during this period to maintain minimum fuel losses.

The effectiveness with which by-product fuels are used in steel plants is of major significance in reducing the quantity of primary or purchased fuel required to produce a ton of steel. The consumption of by-product fuels (blast-furnace gas, coke-oven gas, pitch-tar and coke breeze) for three recent years is shown in Table 3—XVIII.

Of the total gas produced by blast furnaces and coke

ovens, only a portion is available for replacement of purchased liquid or gaseous fuels. Blast furnaces consume 35 to 45 per cent of the top gas produced to operate their own facilities. Fuel is required in stoves to heat the blast, and to provide steam for driving blowers, pumps and electric generators to satisfy furnace demands. By-product coke ovens use the equivalent of 30 to 40 per cent of the gas they produce to heat the ovens in addition to fuel requirements for generating steam for pumping water, process steam and the electric power necessary to produce coke and coal chemicals.

While the primary fuel requirements of recent years represent progress over former years, future utilization of lower grade raw materials (higher ash and sulphur in coal and ore of lower iron content) will require extra diligence and development work to continue the past rate of progress in fuel economy. The present national fuel figure of about 1.3 tons of coal per ton of ingots will be reduced considerably as modern facilities replace those which become obsolete or are of no further economic value, provided the disadvantage of lower grade raw materials is matched by developments in their use. Figure 3—6 illustrates graphically existing potentialities for fuel economy in a steel plant. With modern facilities and efficient practices in a well-balanced and integrated plant, it is possible, as shown in the flow diagram, to produce a ton of ingots with one ton of coal.

Efficiency of Heat Utilization in Steel Plants—1. Blast Furnaces—Blast furnaces require much more fuel than any of the other producing units in a steel plant. However, about half of this fuel is recovered in the form of blast-furnace gas. The efficiency of fuel utilization of the other half required for smelting iron in the blast furnace is very high and no large reduction, such as has been procured during the past twenty years in other processes in the steel plant, can be foreseen. However, any appreciable reduction in blast-furnace coke rates which can be effected will have a beneficial effect upon steel-plant fuel economy as a whole. During the period of World War II, a number of furnaces were increased in size. Practice and tests on some of these revealed potential fuel economies are possible over those of smaller size. Tests with beneficiated ore also have disclosed marked improvement in production and in fuel economy.

The developments of the past twenty years in blast-furnace fuel economy have centered especially on means for procuring better distribution of the gas through the stock and on employment of higher hot-blast temperatures. Better gas flow has been accomplished through improved charging facilities, improvements in the structure and size of coke, in the beneficiation of ore, higher furnace top pressure, and improved furnace lines and working volume. The installation of larger and more adequate stoves has made possible the use of higher hot-blast temperatures.

2. Blast-Furnace Stoves—The fuel efficiency of furnace stoves of older design probably averaged little better than 60 per cent. A comparison of an approximate heat balance of older with modern stoves of 80 to 85 per cent efficiency is as follows:

Item	Former Stoves (%)	Modern Stoves (%)
Heat absorbed by blast.	50 to 70	80 to 85
Sensible and latent heat in stack gases.	20 to 40	10 to 15
Potential heat in stack gases.	2 to 5	0 to 2
Radiation and unaccounted losses.	10 to 25	5 to 10

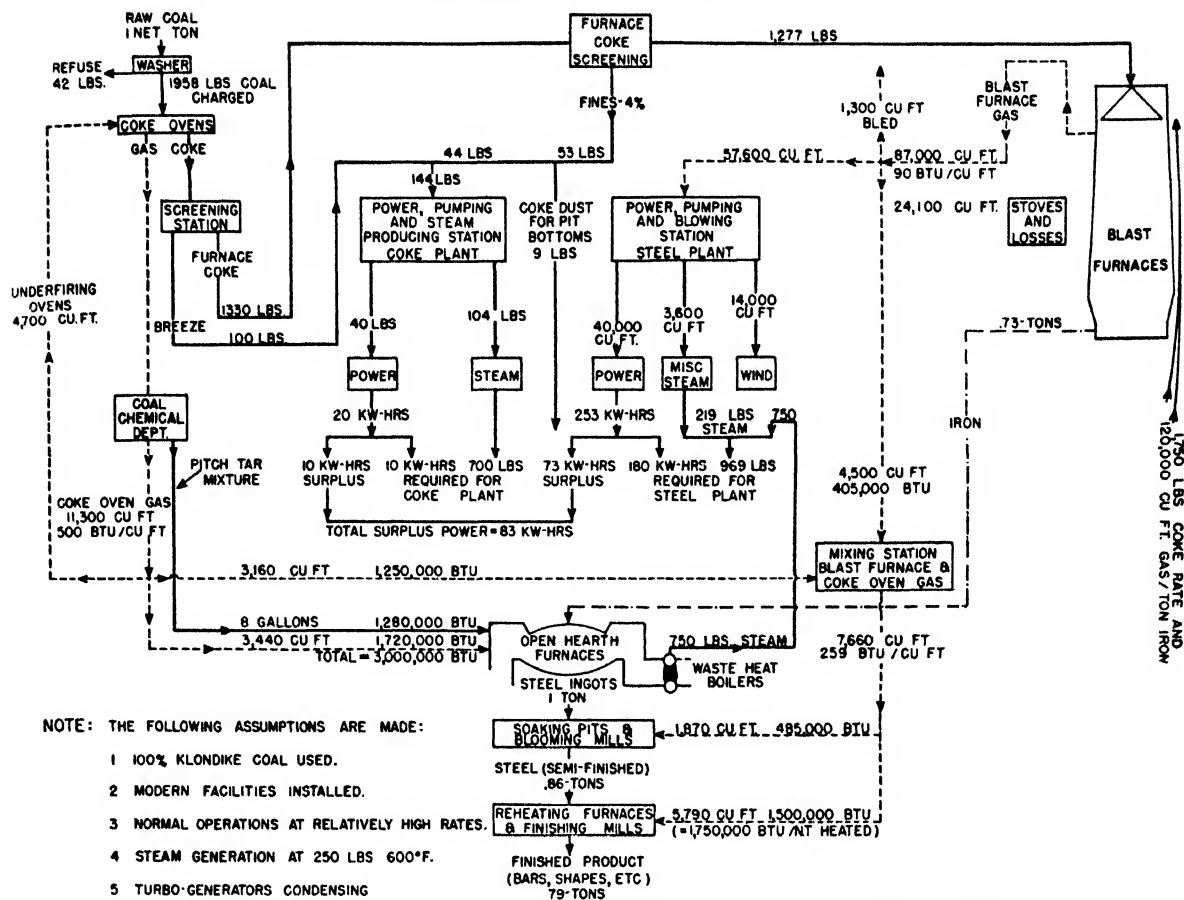


FIG. 3—6. Graphic representation of existing potentialities for fuel economy in a steel plant.

3. Blast-Furnace Blowers—Modern blast furnaces generally are equipped with steam-turbine driven blowers consuming about 18,000,000 to 20,000,000 Btu in steam per blast unit. A "blast unit" is a measure of energy applied to blast-furnace blowing. It is the energy required to compress 1,606,140 cu. ft. of dry air at 60° F and 30 in. Hg adiabatically to 15 lbs. per sq. in. gage pressure, and is equal to 1000 Kwh., or 3,415,000 Btu. Turbo-blowers have replaced many of the older reciprocating-engine-driven compressing tubs or cylinders in which the air was raised to the proper pressure level, an operation which consumed from 30,000,000 to 50,000,000 Btu in steam per blast unit. The gas-engine-driven blowing-tub consumes only about 18,000,000 Btu in gas per blast unit, but its thermal advantage over the turbo-blower is largely overcome by relatively high initial and maintenance costs. Modern turbo-blowers, complete with auxiliaries and efficient boilers (80 to 85 per cent) for producing steam, require about 15 per cent of the total blast-furnace gas produced.

4. Basic Open-Hearth Furnaces—Accomplishments in fuel economy in open-hearth furnaces between 1928 and 1952 outrank those of any other major metallurgical facility in the steel plant, and although available national records are insufficient, it is safe to assume a reduction of 20 to 25 per cent has been made in this interim. In 1926, very few open-hearth shops could boast of consistently producing steel ingots for less than 5,000,000 net Btu per net ton of steel, while today only a few shops require that amount, and the majority of modern shops

seldom exceed 3,500,000 net Btu. The fuel economy effected in basic open-hearth furnaces (largely in the last 20 years) is primarily the result of the development and intelligent use of controls and the gradual change to larger furnaces. Faster charging, reduced bottom delays, minimized hot-metal and pouring delays, better fuels, more effectively designed checkers, ports and burners, and many other factors also have contributed to fuel economy. In addition to the reduction in Btu per ton, large fuel savings, which are not shown by the figures quoted, have been effected by converting the heat in waste gases into steam by the use of waste-heat boilers.

The open hearth, because of the high working temperature required, has very high stack and radiation losses.

The approximate heat balance of a large, modern, open-hearth furnace equipped with waste-heat boilers is shown schematically in Figure 3-7 (see page 85).

Recent technical developments in the use of compressed air or oxygen to hasten bath reactions, atomization of liquid fuels with air rather than steam, enriched air for combustion and better open-hearth refractories, have opened the way for future improvements in the fuel economy of open-hearth furnaces.

5. **Soaking Pits**—Prior to 1930, the majority of soaking pits were of the regenerative type, consuming from 800,000 to 2,500,000 net Btu per ton of ingots, the difference in fuel requirements depending largely upon the temperature of the steel charged into the pits. Several new types which have been developed in recent years, to-

Table 3—XVII. Primary Fuels Consumed by the Iron and Steel Industry^m

Year	Net Tons of Coal		Net Tons of Coal Equivalent			Total Net Tons Coal Equivalent Consumed	Steel Production (Net Tons)		Net Tons Coal Equivalent	
	Anthracite	Bituminous	Fuel Oil*	Natural Gas**	Purchased Electric Power***		Ingots	Hot Rolled Product	Per Net Ton of Steel Ingots	Per Net Ton of Hot Rolled Steel
1952	538,407	86,732,570	13,440,750	8,777,538	9,492,450	118,981,715	93,108,039	71,348,528	1.277	1.668
1951	572,946	99,625,580	15,142,260	8,685,474	10,103,500	134,123,760	105,199,848	81,911,320	1.275	1.637
1950	543,046	91,990,551	13,765,224	8,561,784	8,628,400	123,489,005	96,836,075	75,190,581	1.275	1.642

^mBased on statistics obtained from "Annual Statistical Reports"—Iron & Steel Institute and "Minerals Yearbook"—Bureau of Mines.

Based on coal equivalents (ton of coal = 25,000,000 Btu) as follows:

*1 Gallon of fuel oil = 0.006 tons of coal

**1 Million cu. ft. of natural gas = 42 tons of coal

***1 Kwh. = 1.1 lbs. of coal (national utility average—1952)

Table 3—XVIII. By-Product Fuels Consumed by the Iron and Steel Industry^m
(In Net Tons of Coal Equivalent)

Year	Blast-Furnace Gas ⁽²⁾	Coke-Oven Gas ⁽³⁾	Pitch-Tar ⁽⁴⁾	Coke Breeze ⁽⁵⁾	Total By-Product Fuels
1952	24,280,000	13,612,000	1,682,000	2,555,000	42,129,000
1951	27,829,000	15,698,000	1,820,000	2,884,000	48,231,000
1950	25,576,000	14,343,000	1,769,000	2,879,000	44,567,000

^mBased on statistics obtained from "Annual Statistical Reports"—Iron & Steel Institute and "Minerals Yearbook"—Bureau of Mines.

⁽²⁾Based on 90 percent utilization and 11,000,000 Btu in gas produced per net ton of pig iron.

⁽³⁾Based on 20 tons coal equivalent per million cubic feet of gas.

⁽⁴⁾Based on 0.0064 tons coal equivalent per gallon of pitch-tar.

⁽⁵⁾Based on the proportion of 10,500:12,500 to coal equivalent (0.840).

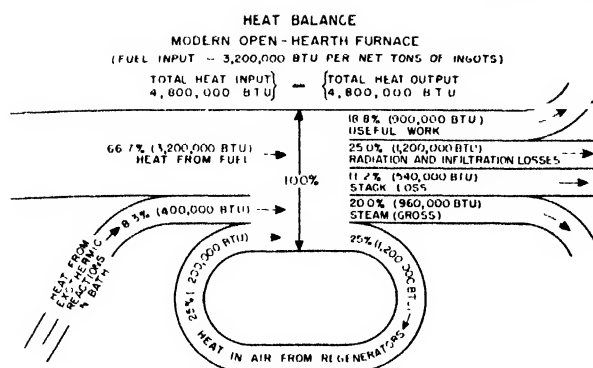


Fig. 3—7. Schematic approximate heat balance of a large, modern, open-hearth furnace equipped with waste-heat boilers.

gether with the development and use of automatic controls, have resulted in a reduction of 300,000 to 1,200,000 Btu per net ton of ingots in the fuel consumption for heating ingots.

6. Reheating Furnaces—The fuel consumption of reheating furnaces varies considerably depending upon the temperature of the steel to be reheated, the size and grade of steel, operating schedules and practices, and furnace design. Table 3—XIX shows the fuel consumption generally realized in modern reheating furnaces.

The principal loss which occurs in reheating furnaces is the sensible heat lost in the stack gases. A reduction of this loss has been accomplished by the installation of regenerators, recuperators or waste-heat boilers.

7. Steam and Power Generation—Boilers in steel plants generate steam for driving blowers, electric-power generators, service water pumps, and a multitude of miscellaneous equipment, as well as provide steam for the heating of buildings, shops and offices, and for general process work. About 1928, reciprocating steam engines were the principal source of power in all the older steel plants. Steam lines from several boilers in large plants provided a labyrinth of feeder lines to various furnace and mill facilities. Leakage of steam and condensation losses in lines were high. Steam pressure and temperatures seldom exceeded 150 lbs. per sq. in. gage and 50° F of superheat. Boiler efficiencies usually were in the 50 to 60 per cent range.

A history of the growth of electrification in steel mills is given in Chapter 24, on "Rolling Mill Drives." This growth has had a continued influence since 1906 on national fuel economics. As existing old mill drives and obsolete steam equipment are replaced with modern electrical facilities, further economies will result. Converting steam power to electrical power will in itself use up power, but once the electrical power is obtained, it can be used very efficiently. While a number of modern

boiler houses now installed are 80 to 85 per cent efficient, the national average for steel plants probably does not exceed 70 per cent. The difference offers a major possible unit reduction in future steel-plant fuel requirements.

The efficient utilization of by-product fuels provides a surplus, in well-integrated plants, for the generation of electric power. A modern fully-integrated plant is capable of generating all of its own power requirements from available surplus by-product fuel. Table 3—XX shows the electrical energy consumed annually by the steel industry in 1950, 1951, and 1952. It also shows the amount of electrical energy purchased from public utilities and that generated within the industry. At least 30 per cent of the electric power requirements in these years was generated by the steel industry itself.

Table 3—XX. Consumption of Electrical Energy in Steel Industry (In Millions of Kwh.)^(a)

Year	Generated	Purchased	Consumed
1952	8,686	17,259	25,945
1951	9,241	18,370	27,611
1950	8,976	15,688	24,664

^(a)"Annual Statistical Reports," American Iron & Steel Institute.

Many of the existing turbines used for blowers, generators, service water pumps, and other facilities installed prior to 1930 were designed for steam pressures under 250 lbs. and 550° F temperature. Future replacement of these facilities will provide further fuel economies, as estimated in Table 3—XXI.

Means Employed for Heat Conservation—The heat from the combustion of fuel which is not utilized in steel-plant metallurgical and service facilities represents an appreciable part of the total supplied. The amount lost differs with various processes. In general, the processes having the higher temperature levels have the greater thermal losses, and, therefore, offer the best opportunity for heat recovery. The largest losses usually are contained in the waste flue gases and in radiation from the furnace walls. The recovery of heat from waste flue gases of high-temperature processes has been practiced for nearly a hundred years. Since that time, improvements in design of the originally conceived regenerators, and the development and use of recuperators and waste-heat boilers, have made possible substantial recovery of heat losses. The reduction of radiation losses by the use of insulating material has been practiced in some processes for possibly fifty years; in others for relatively only a few years. The utilization of higher steam pressures and temperatures and more efficient facilities for steam generation has progressed slowly but definitely in the past two decades.

Waste flue gases contain both sensible heat and the

Table 3—XIX. Fuel Consumption in Reheating Furnaces

Furnace Type	Material Heated	Net Btu per Net Ton of Steel	
		Minimum	Maximum
Batch	Hot blooms and billets	300,000	1,500,000
Continuous	Cold slabs and medium-sized blooms	1,600,000	2,800,000
Batch	Cold slabs and blooms	1,800,000	4,200,000
Continuous	Cold small billets	1,000,000	2,400,000

Table 3—XXI. Estimated Fuel Economics Possible with Modern Steam Turbines, Utilizing Higher Pressures and Temperatures

Steam Pressure (lbs. per sq. in. gage).....	250	450	650	850	1000
Total steam temperature (°F).....	550	750	750	850	900
Total heat per lb. of steam (Btu)....	1288.5	1386.5	1377.0	1427.5	1453.0
Heat in liquid at 300°F (Btu).....	269.5	269.5	269.5	269.5	269.5
Heat supplied by boiler (Btu).....	1019.0	1117.0	1107.5	1158.0	1183.5
Theoretical water rates (lbs. steam/kwh.)...	8.48	7.08	6.83	6.29	6.03
Expected over-all fuel requirements (allowing for 80% efficient boilers, 78% rotary-converter efficiency, and 10% for auxiliaries) Btu/kwh.	15,400	14,050	13,450	12,950	12,700
Over-all thermal efficiency (%).....	22.2	24.3	25.4	26.4	26.9

latent heat of vaporization of water and sometimes potential heat (unburned fuel gases). The recovery of the heat of vaporization of water is not practicable, and the elimination of potential heat in waste flue gas is controlled by providing sufficient air for combustion at the burners. The amount of sensible heat in waste flue gases is the product of the heat content per cubic foot or pound of gas, times the volume or weight of gases.

The total loss of heat in waste flue gases is minimized by providing only sufficient air for combustion, and by preventing air infiltration. The temperature of furnace exit gases is lowered by observance of heat-transfer principles. Waste flue-gas temperature is reduced and heat recovered by heat exchangers such as regenerators, recuperators and waste-heat boilers. The heat content of the products of combustion of coke-oven gas and natural gas at different temperatures for various amounts of excess air is shown respectively in Figures 3-8 and 3-9.

Fuel-Air Proportioning—In modern steel-plant furnaces, and in boilers, the amount of air supplied for combustion is maintained only a little above theoretical requirements by automatically controlling its flow in chosen proportion to the supply of fuel. In open-hearth and some other regenerative furnaces in earlier times, air for combustion was drawn into and through the regenerators solely by the draft created by the furnace stack. Crude attempts were made to control the supply of air by opening or closing the stack dampers. Where

preheated gas was used as fuel, the problem was still more complicated.

In modern furnaces of this type, all possible points of leakage in the furnace system have been sealed to prevent air infiltration, and controlled quantities of air, in selected proportion to the amount of fuel supplied, are blown into and through the regenerators and thence into the furnace to mix with the fuel to provide the controlled combustion. In other types of furnaces, such as in heat-treating furnaces, special pressure regulators and valves accurately proportion the amounts of fuel and air fed to the burners and effect the same result.

Accurate and automatic proportioning of fuel and air has contributed to improvements in fuel economy, furnace efficiency and process control in the relatively recent past. The control of the rate of firing has also made important contributions and will be covered under "Automatic Temperature Control."

Oxygen Enrichment of Combustion Air—Air is composed of only about 20.9 per cent oxygen, with the remainder consisting of inert nitrogen plus a small amount of several other inert gases. When combustion takes place, the oxygen combines with the carbon and hydrogen to the fuel and liberates heat. The inert gases of the air absorb heat from the combustion and carry it out of the furnace, and it is lost so far as the furnace process is concerned. They reduce flame temperature by absorbing heat, thus reducing rate of heat transfer to the work.

It has been known for a long time that if the inert

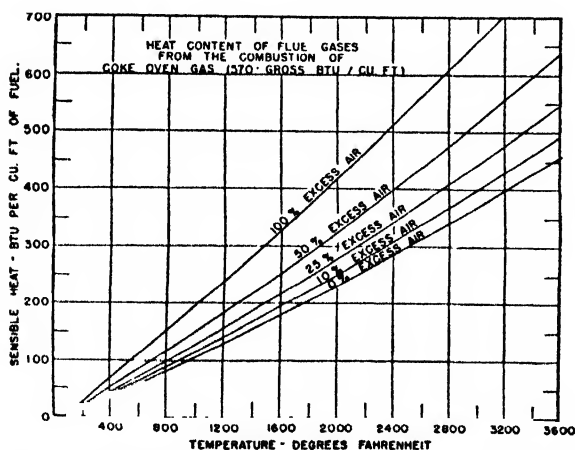


FIG. 3-8. Heat content of products of combustion of coke-oven gas at different temperatures for various amounts of excess air.

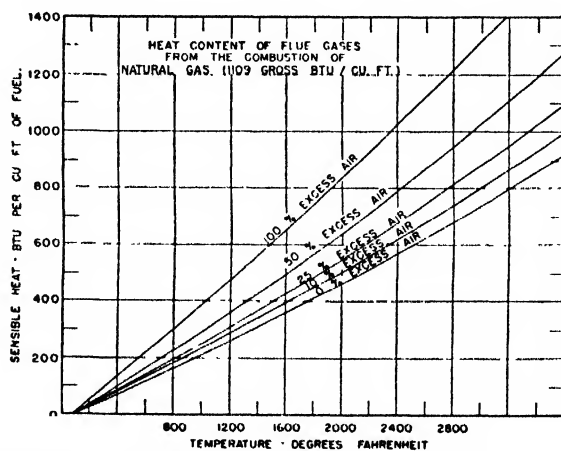


FIG. 3-9. Heat content of products of combustion of natural gas at different temperatures for various amounts of excess air.

content of air could be diminished, much more efficient combustion could be attained. Recent technical developments that have lowered the production cost of oxygen of commercial purity have made large-scale use of this gas economical for some industrial processes. Consequently, many plants have experimented with the addition of oxygen to ordinary air used for combustion, with generally good results. In effect, increasing the oxygen content lowers the inert content of the air; consequently when a given amount of fuel is burned with oxygen-enriched air, the volume of waste gas is less than if ordinary air is used. If the temperature of the waste gas is not increased, the sensible heat loss in the flue gas will be decreased, due to the smaller heat capacity of the smaller volume. In furnaces operated at a high thermal head, a diminution of the inerts usually results in a decrease in waste-gas temperature. With the same fuel input, enriched air for combustion raises flame temperature of a given fuel, thereby improving heat-transfer rate and increasing production; alternatively, the fuel input may be decreased when enriched air is used, to maintain the same production rate as obtained with more fuel using ordinary air. Increased production rates almost always reduce the heat losses per ton of product in any furnace employing a high thermal head.

Furnace Pressure Control—If the pressure of the gases in the heating chamber of a furnace is below atmospheric, cold outside air will be drawn into the furnace through any openings that exist. If the interior pressure is above that of the outside air, the hot gases will be forced out of the furnace through these same openings, and if too much higher will, in addition, tend to penetrate the refractories with, in some cases, damaging effect. It is desirable generally to operate a furnace with a slight positive pressure in the heating chamber (i.e., furnace pressure slightly higher than atmospheric). It should be noted that the pressure from top to bottom of the heating chamber is not uniform, due to the stack effect of the hot gases. Control, therefore, is aimed at maintaining the desired pressure at hearth level.

Air drawn into a furnace operating under negative pressure upsets the fuel-air ratio which is controlled automatically or by valve settings. In some furnaces, such as reheating or heat treating, this air aggravates the problem of oxidation (or scaling) of the work because of the oxygen present.

If the pressure in the furnace at hearth level is equal to atmospheric or slightly positive, better heating conditions are obtained by improving heat transfer, and by better control of temperature uniformity. This is especially so in furnaces where most of the heating of the work takes place through heat transfer by radiation from the flame to the bath or work. The positive pressure must be controlled to prevent excessive sting-out of flame from furnace openings (a small pressure imparts a high velocity to hot gases), as well as to avoid the build-up of excessive back pressure that would interfere with the proper flow of fuel (if gaseous) and combustion air. Positive pressures maintained at hearth level in practical work are quite low, ranging only up to a few hundredths of an inch of water. Furnace pressure is controlled by adjusting the opening in the stack damper.

Positioning of the damper can be done manually, using the flame sting-out as an indication of the existence of a positive pressure, but it is difficult to adjust the opening for the frequent changes in furnace conditions. The development, about the year 1928, of industrial-type instruments with sufficient sensitivity to measure differential gas pressures with an accuracy of ± 0.0025 inch

of water, made possible the use of automatic control of furnace pressure. Such instruments employ a diaphragm (or equivalent) to measure the difference between furnace and atmospheric pressure, and, through electrical relays or other devices that operate motors or by hydraulic systems, move the stack damper automatically to maintain the desired pressure in the furnace.

Automatic furnace-pressure control has been provided for the majority of steel-plant furnaces, and has been a principal factor in the improvements in fuel economy and efficiency of melting and reheating furnaces in the steel industry during the years following its adoption.

Automatic Temperature Control—By eliminating as nearly as possible human error in judging temperatures by senses, automatic instruments for measuring temperature have contributed largely to the improvement and economy of many steel-plant operations. The use of excessive amounts of fuel is wasteful of the fuel itself and results in high furnace exit gas temperatures and damage to refractories. In some processes, high fuel rates not only do not hasten transfer of heat to the material being processed, but also may cause actual damage to it. The use of insufficient fuel reduces the rate of heat transfer and prolongs process time, thereby increasing thermal losses. The optimum fuel rate for protection either of the material being heated or the furnace refractories, and often for control of heating or production rate, is maintained in many types of furnaces by automatic temperature-measuring instruments which control the fuel rate by actuating electrical relays or other units which control the operation of motors, hydraulic systems, or other means for regulating valves which control the fuel rate. It would be impossible within the scope of these pages to discuss even briefly the many types of instruments and auxiliaries used for the automatic control of temperature; however, it may be stated that the instruments employed for measuring high temperatures in the steel industry operate on three main principles: (1) by measuring the intensity of radiation emitted by the hot furnace or object; (2) by measuring the minute electric current generated in a circuit composed of two wires of dissimilar metals, joined end to end, when one of the joints is heated (this is the principle of the thermocouple), and (3) by measuring the change in electrical resistance of conductors when heated to the temperature in question.

Regenerators and Recuperators—Regenerators are used alternately to absorb heat from one fluid and then transfer it to another fluid; recuperators are used to transfer heat continuously from one fluid to another. The fluids referred to in these two definitions are: (1) hot, gaseous products of combustion which give up heat during passage through the regenerator or recuperator and (2) fuel gas or air for combustion which undergo heating while passing through the regenerator or recuperator. Regenerators are applied usually to furnaces which can be fired alternately from the ends, the flow of gases through the furnace and regenerators being reversed by predetermined time and/or temperature cycles. Open-hearth furnaces, and many soaking-pit and batch-type reheating furnaces, are equipped with regenerators. Blast-furnace stoves also use the regenerative principle but operate over a much longer cycle and in a somewhat different manner than that practiced in other installations. In a blast-furnace stove, the checker brick is heated by burning a fuel exclusively for the purpose of heating the regenerator brick while in the open-hearth and other furnace installations the checkers are heated by waste gases. In both cases, the heat stored in the regenerators is used to preheat air for the combustion of fuel in the furnace they serve.

Recuperators have been applied in many cases to modern pit-, batch- and continuous-type reheating furnaces, and to steam boilers. When applied to steam boilers, they are commonly called "air preheaters." Recuperators are of three general types, classified according to the direction of flow of the waste gases and air, as follows:

1. Counter-flow
2. Parallel or co-current flow
3. Cross flow

Counter-flow is used to attain maximum air-preheat temperatures; cross-current flow to secure optimum heat-transfer rates (Btu per sq. ft. of recuperator surface per degree F temperature difference per hour). Parallel flow is used where it is desirable, such as in metallic recuperators, to maintain the temperature of the division wall between the two fluids as uniform as possible throughout its length and to hold the temperature of the hot end to a minimum. Generally, a combination of the first and third types is applied to many steel-mill furnace applications where a refractory material is used to divide the two fluids. A combination of the two types is accomplished by baffling the flow of one of the fluids. In such designs, the general direction of flow of fluids exchanging heat is countercurrent and the flow in each baffled section is cross-current. Where the temperature of the waste-gas from which heat is to be extracted is relatively low, say under 1800° F, metallic tubes are generally used since they possess an advantage against leakage. High-temperature recuperators are generally constructed of a clay or silicon-carbide material.

Waste-Heat Boilers—These units are used to obtain heat recovery when the practical limit of recovery has been obtained by regenerators or recuperators and there is still sufficient heat left in the waste gases to justify expenditures for waste-heat boilers. In modern large, stationary open hearths, waste gases leave the regenerators at temperatures of approximately 1200° F, and from this level waste-heat boilers recover 30 to 33 per cent of the fuel used, converted into steam. Waste-heat boilers also are used sometimes in place of regenerators or recuperators, depending on conditions such as where preheated air is undesirable or where the generation of steam solves the problem of fuel conservation more satisfactorily.

Waste-heat boilers are most applicable to high-temperature, continuous processes and have been used principally in the steel plant in connection with open hearths and, to a lesser degree, with reheating furnaces and soaking pits. Fire-tube and water-tube boiler types have been installed, the former being the preferred type, generally of horizontal, single-pass design. Approximately 35 sq. ft. of boiler heating surface per ton of stationary open-hearth capacity, designed for high mass velocity of waste gases through the tubes to develop a scouring action for keeping the tubes clean and to provide high heat transfer, is used normally. Waste-heat boilers usually are provided with superheaters and sometimes with economizers.

Insulation—Thermal insulating materials have been used in steel plants for a great many years. There are many different kinds of insulating materials, each being most suitable for a specific temperature level and for the degree of insulation desired. Early applications were made to enable a facility to function more satisfactorily. Insulating material, such as Kieselguhr, was used to line the shell of a blast-furnace stove to aid in the retention of heat for later use by the blast furnace. Steam lines were covered with asbestos or other material to

prevent condensation of steam and consequent loss of power or trouble with blowing or mill engines. The value of insulating material to conserve fuel, to afford safer and more comfortable working conditions, to protect materials susceptible to damage from heat and thermal strain, and to speed up furnace operations became evident and the progressive application of insulating material to practically all facilities which employ heat has proven beneficial to steel-plant economy. The high thermal heads at which many steel-plant operations are carried out are particularly conducive to the use of insulation for preventing large losses from radiation. In the modern steel plant, the use of insulation is justified in terms of fuel saved in boilers; stoves; open-hearth regenerators; reheating, pit, forge and annealing furnaces; steam, hot-blast and preheated-air lines; and many other miscellaneous facilities. In some operations, such as in the open hearth, the use of insulation is restricted by the temperature the refractories will stand; consequently, radiation losses are still very high. Heat losses have been reduced in such cases by sealing cracks and openings in a furnace wall to prevent heat loss from radiation, exfiltration of gases and infiltration of cold air. Further progress in the development of high-temperature refractories will permit further insulation. The development of insulating firebrick in recent years is particularly significant to the fuel economy of furnaces which are operated intermittently. Many of the older furnaces are constructed of heavy refractory walls which must be heated up before the furnace is capable of producing at normal rates. These heavy furnace walls soak up considerable heat, which must be supplied by fuel. The replacement of these furnace walls with much lighter insulating firebrick permits heating up more rapidly because of less heat absorption, and permits increased production.

Instruments, for measuring temperature, pressure, volume, weight, electrical energy, etc., are used extensively in the steel industry as operating guides. They benefit fuel economy directly or indirectly.

Chapter 4

METALLURGICAL COKE PRODUCTION

SECTION 1

INTRODUCTORY

Selection and Preparation of Coal for Coking—Coke is the term used to describe the residue from the destructive distillation of bituminous coal. Structurally it is a cellular, porous compound which also is heterogeneous in both physical and chemical properties. The physical properties of metallurgical coke, as well as its composition, depend largely upon the coal used and the temperature at which it is carbonized. Not all bituminous coals will form coke, and not all coking coals will give the same firm, cellular mass characteristic of coke suitable for metallurgical purposes. Some coals will produce an acceptable coke without blending with other coals, while others are usable only as constituents of blends. Chapter 3 already has described how preparation of coals prior to coking is an important element in metallurgical-coke production. The type and method of operation of coking facilities also exert a profound influence on the quality and yield of coke for the blast furnaces.

Kinds of Coke—There are three principal kinds of coke, classified according to the methods by which they are manufactured: low-, medium- and high-temperature coke. All of the coke used for metallurgical purposes must be processed in the higher ranges of temperature if the product is to have satisfactory physical properties. Even with good coking coal, the product obtained by low-temperature carbonization (900° to 1400° F) is unacceptable for metallurgical purposes.

It is generally agreed that the most desirable blast-furnace coke is made from mixtures of high-volatile and low-volatile coals, pulverized and blended and then coked in ovens capable of heating the mass to a uniformly high temperature.

At the majority of coke plants, high-volatile coking coals are blended with low-volatile coals in varying percentages, depending on the particular coals, the purpose being to produce coke of high quality and yield. In the earlier days of the coke industry, some of the coking coals used were superior to those currently being carbonized. Some progress has been made in the utilization of coals which expand and exert pressures on oven walls during coking. In addition, it is generally conceded that higher ash and sulphur coals are acceptable currently for coking purposes than in use in former years. In all probability the practice of blending coals has been one of the greatest single factors in the extension of coal reserves usable for coking purposes. At present, laboratory scale tests are available which are capable of evaluating both the coking and expansion characteristics of individual coals or blends. A few plants retain the practice of using only a single coal for coking purposes while others use mixtures of high-, medium- and low-volatile coals for coke production.

The practices followed in this respect are determined largely by economic considerations.

The acceptability of a coal for metallurgical coke production depends on various factors relating to its chemical and physical characteristics as well as its economic availability. While laboratory tests can be used to develop data to permit evaluations in this respect, final appraisal can be determined best by actual full-scale plant test under exact conditions of their use.

Factors Controlling Properties of Metallurgical Coke—Coke for blast-furnace consumption must be sufficiently firm and strong to resist shattering by handling, and crushing by the pressure exerted by the heavy blast-furnace burden. It should be free of dust and fines, and in pieces not too large for optimum speed of combustion. With a good coking coal, these physical properties can be controlled only moderately by the coking process. As the coal is heated, it becomes plastic at 660° to 890° F, forming a fused mass irrespective of its form when charged into the retort, and through this range of temperature, volatile matter is given off, rapidly at first, then more slowly up to about 1740° F. The coals making up a blend, so far as possible, should have about the same plastic range. Slow heating through the plastic range increases slightly the hardness of the coke. The size of the lumps of coke depends largely upon the thickness of the coal charge and whether or not it is heated from one or both sides. As to chemical composition, a good metallurgical coke will contain very little volatile matter—not over 2 per cent—and 85 to 90 per cent fixed carbon. The remainder is ash, sulphur and phosphorus. The phosphorus content, 0.018 to 0.040 per cent for making Bessemer iron, preferably should be low also for basic iron. Sulphur varies from 0.6 per cent to 1.5 per cent, but is desired as low as possible because coke is the chief source of sulphur in the pig iron produced. Standard specifications for foundry coke call for a volatile-matter content of 2 per cent, a maximum sulphur of 1 per cent, a maximum moisture of 3 per cent, and a minimum fixed carbon of 86 per cent. Shatter and tumbler tests are also specified, but no standard for combustibility has been adopted. These requirements are controlled through selection of the coal, which should be low in sulphur, free from slate or removable refuse, and give an ash which has a moderately-high fusion point in a reducing atmosphere. The question of why coals coke is not fully understood.

Methods of Manufacturing Metallurgical Coke—There are two methods for manufacturing metallurgical coke, known as the beehive, and the by-product or retort process. In the beehive process, air is admitted to the coking chamber in controlled amounts for the purpose of burning therein the volatile products of the coal to

generate heat for further distillation. In the by-product method, air is excluded from the coking chambers, and the necessary heat for distillation is supplied from external combustion of some of the gas recovered from the coking process. With modern by-product ovens, properly operated, all the volatile products liberated during coking are recovered as gas and coal chemicals, and about 40 per cent of the gas produced is returned to the ovens for heating purposes. While the beehive process was the leading method for the manufacture of coke up to 1918, it now has been replaced largely by the by-product process. There are still places, however,

where beehive ovens serve a useful purpose. This is due to the fact that the beehive oven is more flexible in operation and capable of peak coke production more economically than the by-product type oven. As far as can be determined, the processes of manufacture have little effect on the quality of the coke for blast-furnace use. There is a difference in coking temperature of the two processes, that of the by-product being somewhat lower than the beehive. Beehive coke is usually larger, though not as uniform in size. In general, properly carbonized beehive and by-product coke are silvery gray in appearance when quenched with fresh water.

SECTION 2

THE BEEHIVE PROCESS FOR CARBONIZING COAL

Construction of Ovens—As shown in Figure 4—1, the name beehive is literally descriptive of the form and construction of the beehive coke oven. Beehive coke oven plants constructed in the past have followed three general arrangements, as follows: (1) the bank system, in which the ovens are built in single rows against a bank of earth, natural or artificial, thus making it necessary to build but one retaining wall along the front of the ovens; (2) the single-block system, which consists

of a single row of ovens with retaining walls at both the front and back; and (3) the double-block system, in which the ovens, in a double row, are built back to back or staggered with a retaining wall extending along the front of each row. Figure 4—2 illustrates the double-block system.

Waste-Heat System—Early in this century, an occasional beehive coke-oven battery, usually of the banked ovens, was arranged for utilizing the waste heat

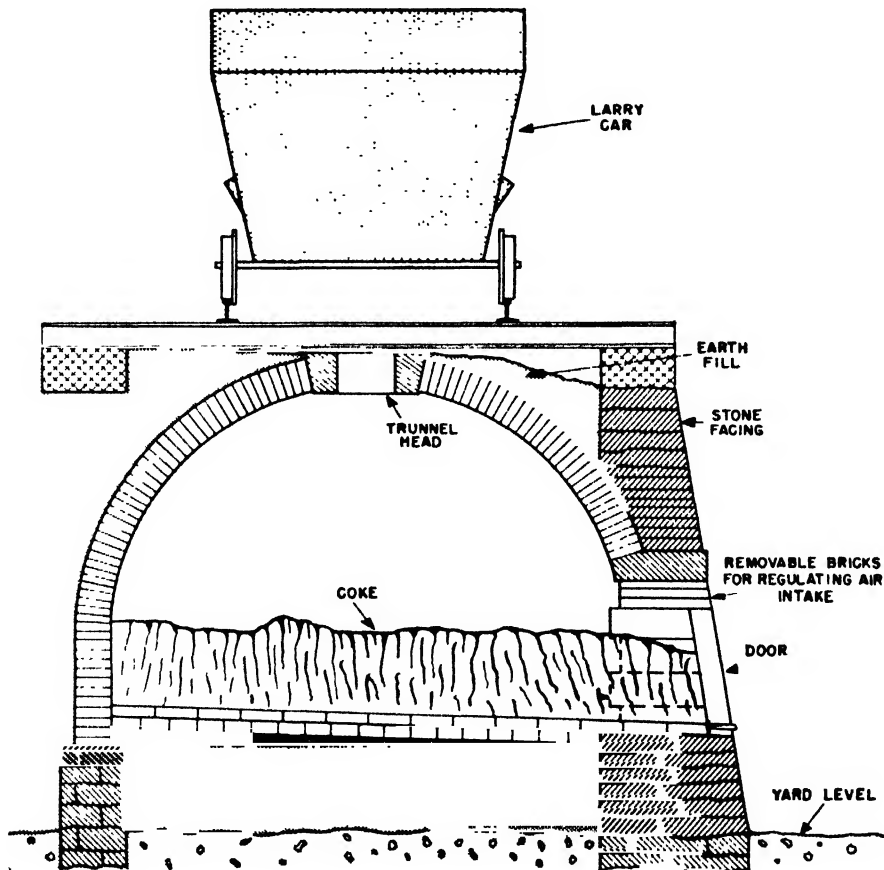


FIG. 4—1. Ideal section of beehive coke oven in a single-block battery, showing refractory brick lining, the clay and earth fill, the arched door through which the coke is watered and drawn, and the trunnel head at the top through which the coal is charged and the volatile products escape. Vertical lines in the coke bed indicate the fissures that develop during coking, giving beehive coke its characteristic columnar structure.

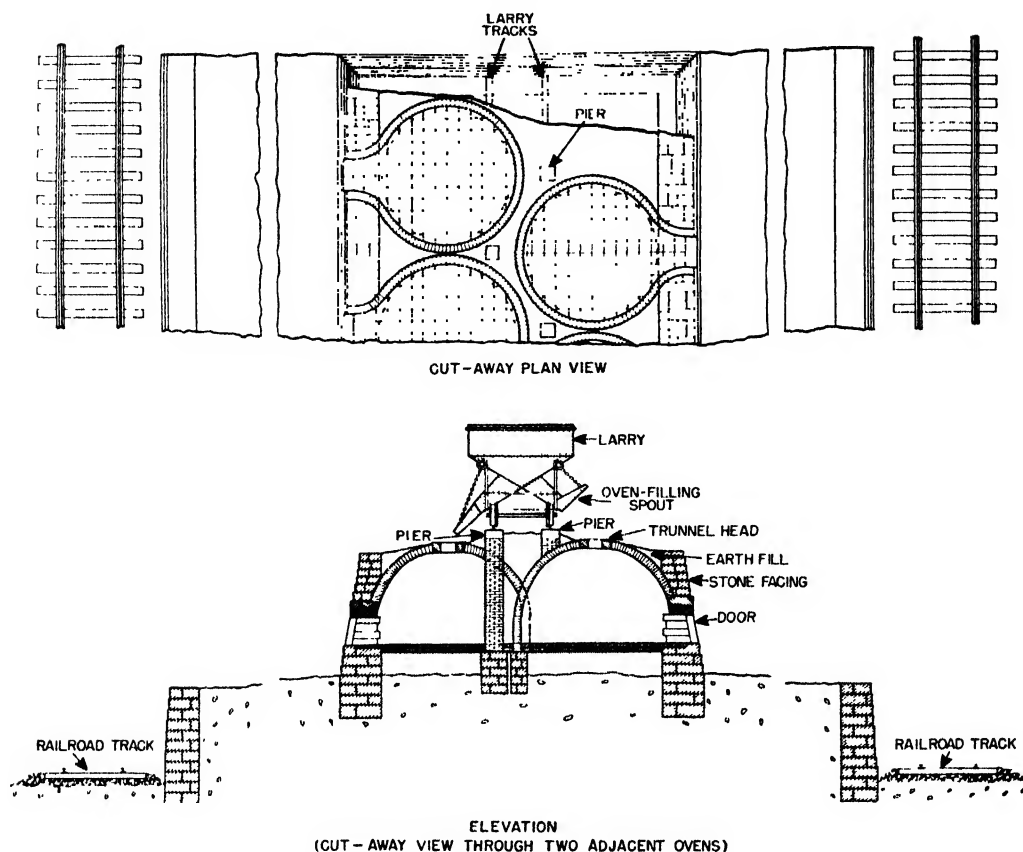


FIG. 4-2. Schematic diagram showing arrangement of beehive ovens built according to the double-block system.

from the products of combustion to generate steam. A large tunnel was constructed in the bank some 10 feet back of the ovens and parallel to the battery. This tunnel was connected to each oven by a flue which conducted the hot gases out of the oven from an opening sufficiently above the side wall to prevent its being closed by the largest charge of coal. Each flue was provided with a damper for closing off the draft during the period when the oven was being watered, drawn, and charged. From the battery, the tunnel passed to the boiler house, where branches conducted the hot gases through the fireboxes and flues of the boilers to a common stack. The stacks used were about 100 feet in height to produce proper draft. During the coking period, the coal-charging hole on the ovens necessarily was kept tightly closed. Owing to the increased draft, these ovens were inclined to run at a little higher temperature than ordinary beehive ovens, causing the temperature in the tunnel to be high, sometimes reaching 2730° F. A maximum of about 20 horsepower per oven was generated from the waste heat from a battery. This method did not achieve widespread popularity in this country and was used only at a few plants.

Charging—The ovens are charged as soon as practicable after drawing, in order that stored-up heat from the previous charge will be sufficient to start the coking process. New ovens must be heated up gradually to the coking temperature by wood and coal fires, after which small charges of coal for coking are used until the ovens reach normal working conditions. With the oven in

readiness for charging, the door is partially bricked up and the charge is dropped through the trunnel head from the larry car above, leaving the coal in a cone-shaped pile in the oven. In order to secure uniform coking of the coal, this pile must be leveled so that the coal will lie in a bed of uniform depth over the entire bottom of the oven. This leveling may be done by machine or by hand. In works not equipped with a machine, the leveling is accomplished by a large long-handled scraper, operated through the door of the oven, which is purposely bricked up to only two-thirds of its height at the time of charging. After leveling the coal, the door opening is then bricked up to within about 1½ inches of the top.

Coking Process—The coking process begins very soon after leveling is completed, as the ovens retain enough heat in the brick of the walls and the loam backing to start liberation of the volatile matter from the coal. As more heat is absorbed by the coal charge, the temperature of the oven soon reaches the "kindling" (ignition) point of volatile gases, which, in the presence of the air admitted to the oven, ignite with a slight explosion at first, and then continue to burn quietly in the crown of the ovens, or as small candlelike flames at the surface of the coking mass, thus supplying heat to continue the process. Coking proceeds from the top of the coal downward, so that the coking time depends mainly upon the depth of the coal. The generation of gas thus rapidly approaches a maximum, which is maintained for a period, then declines to practically nothing. The

burning of volatile matter during this period must be regulated by gradually closing up the opening at the top of the door for admission of air. This regulation is necessary to maintain the temperature at a maximum, and conserve coke, as an excess of air at the beginning of the coking period tends to cool the oven, and later consumes the carbon of the coke. The yield is also reduced by improper leveling. If the coal is not of uniform depth in the oven, the thin portions coke through before the thick, and some of the coke of the thin sections is consumed while the coking of the thick portions is being completed. On the other hand, if the process is stopped when the thin areas have coked through, there will be a loss due to uncoked butts on the thick areas. In the coking of bituminous coals in beehive ovens, coking proceeds downward from the top of the charge in which the coal, at increasing depths, passes through a plastic state as the temperature rises. This produces expansion and contraction of the charge with the result that the cake is ramified by a great number of irregular vertical fissures, thus giving it a long columnar structure. These very irregular columns extend from the top to the bottom of the cake. This structure affords a means by which beehive coke can be distinguished from by-product coke.

Watering and Drawing—At the end of the coking time, the brickwork closing the door is torn out, and the coke is watered out. Usually, this watering is accomplished by a self-propelled spraying device. It consists of a tube or pipe a few inches shorter than the diameter of the oven, pivoted at the center to a feed pipe and perforated by two rows of holes on opposite sides, starting from the center. The holes are arranged to throw jets of water horizontally, which causes the pipe to revolve. Where this device is not provided, the ovens are watered by spraying with a stream of water

through the door of the oven. After watering, the coke may be drawn either by hand or machine. As the work is arduous, a machine known as the Covington coke-drawing machine is commonly employed. It is provided with a long arm fitted with a head, flat on the bottom, but inclined on the top, and a pair of hinged ears, or drawing lugs. Upon being pushed by motor into the oven, the head moves in advance of the drawing lugs, which lie flat, and raises the coke from the bottom of the oven. Upon the return, the lugs engage this loosened coke and force it through the door in advance of the head to fall upon a conveyor system that carries it to a screen, from which it passes to railroad cars. It is impossible to remove all the coke with the machine, and what remains must be drawn out by hand upon the conveyor. In straight hand-drawing, the coke is drawn out into the yard and forked into barrows, which are used to wheel the coke into railroad cars. The forking and screening leave the coke free of smaller particles called breeze.

Present Status of Beehive Process—The beehive process still holds an advantage for certain peak requirements where the high investment cost of a by-product plant cannot be justified because of long inoperative periods. Beehive coke is usually made near the mine that supplies the coal, which in turn is determined largely by the availability of coal which can be coked successfully in this type oven.

Due to adaptability the coals used formerly were seldom pre-treated or blended with other coals. Because of depletion of coals in these areas and due to the rather limited range of individual coals that are suitable for beehive-coke making, it is possible that this method will find decreasing future use unless blended coals can be used economically or some process developed for recovery of waste gases.

SECTION 3

THE BY-PRODUCT PROCESS FOR CARBONIZING COAL

The by-product process, being a true distillation process, involves the use of retort ovens. While there are many modifications, these ovens consist essentially of three main parts; namely, the coking chambers, the heating chambers, and the regenerative chambers—all constructed of refractory brick. The coking chambers are rectangular in section, varying in general from 30 to 42.6 feet in length, from 6 to 14 feet in height, and 12 to 22 inches in width. From 10 to 100 ovens constitute a battery of ovens, in which coking chambers alternate with heating chambers so that, in effect, there is a heating chamber on each side of each coking chamber. The regenerative chambers are underneath the heating and coking chambers. Separating walls, between regenerators, also serve as foundation walls for the heating and coking chambers. The entire structure is supported either from the ground or by columns under a structural-steel base. The coal is charged through openings in the top of the oven, and the coke is pushed out from one end by a power-driven ram, or pusher, acting through the other end. All quenching or watering of hot coke is done outside of the oven. During the coking period, the ends of the oven are closed by refractory-lined doors, which must be constructed so as to effect complete sealing of the oven. The ovens first constructed in the industry provided a space between the door and the jamb which was filled with a special luting mixture prior to charging. Later several types of self-sealing

doors were developed, which seal the opening when put in place and require no luting. To permit the escape of the volatile matter, which must undergo several different treatments to separate the various coal chemicals, an opening is provided through the top and at one or both ends of the oven. This opening is fitted with an offtake pipe, which in turn connects with the gas-collecting main for the battery.

The combustion chambers consist of a great number of flues in order to promote uniformity of heating throughout the entire length of the oven. Ovens with both horizontal and vertical heating flues have been built but the latter has largely replaced the former in present installations. While some of the older ovens employed the recuperative principle for preheating the air for combustion, modern practice demands the use of regenerative chambers, because the heat is better conserved and less gas is thereby required to heat the ovens. In the arrangement of these regenerators, two plans have been employed. By the first plan, the regenerative chambers, two in number, are placed longitudinally beneath a whole battery of ovens; with this arrangement, the ovens of a battery are at right angles to the regenerators. Each end of each oven is connected to one of the regenerators. The flow of gases, obviously, must be reversed simultaneously for all ovens in a battery when changing from one regenerator to the other. This precluded, to a great extent, precise heat regulation for an individual or selected

group of ovens in a battery. This type oven did not achieve widespread popularity in this country. In the second plan, individual regenerators are placed under each oven. This plan permits control of the preheated-air supply for combustion to individual vertical heating flues, and makes closer control of heating possible to improve uniformity of heat distribution, in conjunction with other refinements made practicable by this system. Another advantage of using individual regenerators is that each oven is thus made more nearly an independent unit, and the operation of the whole battery is not liable to be influenced by one or two ovens that may be shut down for repair or other reasons.

Heating of individual ovens is controlled so that the temperature at the base of the flues in which gas is being burned does not exceed 2600° to 2700° F, which is considered the safe maximum temperature range to which coke-oven refractories should be subjected. With the flues operating within this temperature range, coking time depends upon the width of the oven, the nature of the coals being coked, and other factors. In general, a coking time is selected that will produce a uniform "skin temperature" of the block of coke in the oven of from 1900° to 2000° F at the time the charge has been coked all the way through to the center. The "skin temperature" referred to above applies to the coke adjacent to the walls of the oven. The time required for coking coal under the above operating conditions will vary from 16 to 20 hours, depending upon the factors already stated. Average time is about 17 to 18 hours.

Modern Types of By-Product Ovens in the United States—In the steel industry in the United States, the transition from beehive to by-product ovens was accelerated with the start of World War I when the construction of by-product coke ovens was started in many

locations in the eastern and central states. While some of the ovens erected at that time are still in operation, a large number either have been rebuilt "in kind" or replaced with ovens of different design. The present popular designs in the United States, largely named after either the designer or builder, are the Koppers, Koppers-Becker, Wilputte and Semet-Solvay. The chief differences in design of these four types involve the heating systems employed. Figure 4-3 illustrates schematically, by simplified sketches, how each type is heated. More detail is given in the text and in subsequent illustrations. Ovens of these four designs comprise about 97 per cent of the by-product coking capacity in this country.

Metallurgical coke, while still the primary product, no longer commands the almost exclusive consideration in oven design as in former years since the coal chemicals recoverable in the by-product process are in constantly increasing demand.

The Koppers Oven—This type of by-product oven, more technically referred to as a regenerative, single-divided oven, was the most prominent around 1916 and many are still in operation. In a typical oven of this period (Figures 4-3, 4-4, 4-5 and 4-6), all parts except the foundation and battery top are constructed almost entirely of the best grade of silica brick.

The majority of this type oven initially constructed in the United States had a coking volume of 500 cubic feet. The dimensions of these ovens were: length, 37 feet from face to face of the doors; height, 9 feet, 10 inches from floor to roof; and width, tapering from 17 inches at the pusher end to $19\frac{1}{2}$ inches at the discharge end. Usually four charging holes were provided in the top for admitting the coal charge, while a separate opening at one end provided an outlet for volatile matter. The

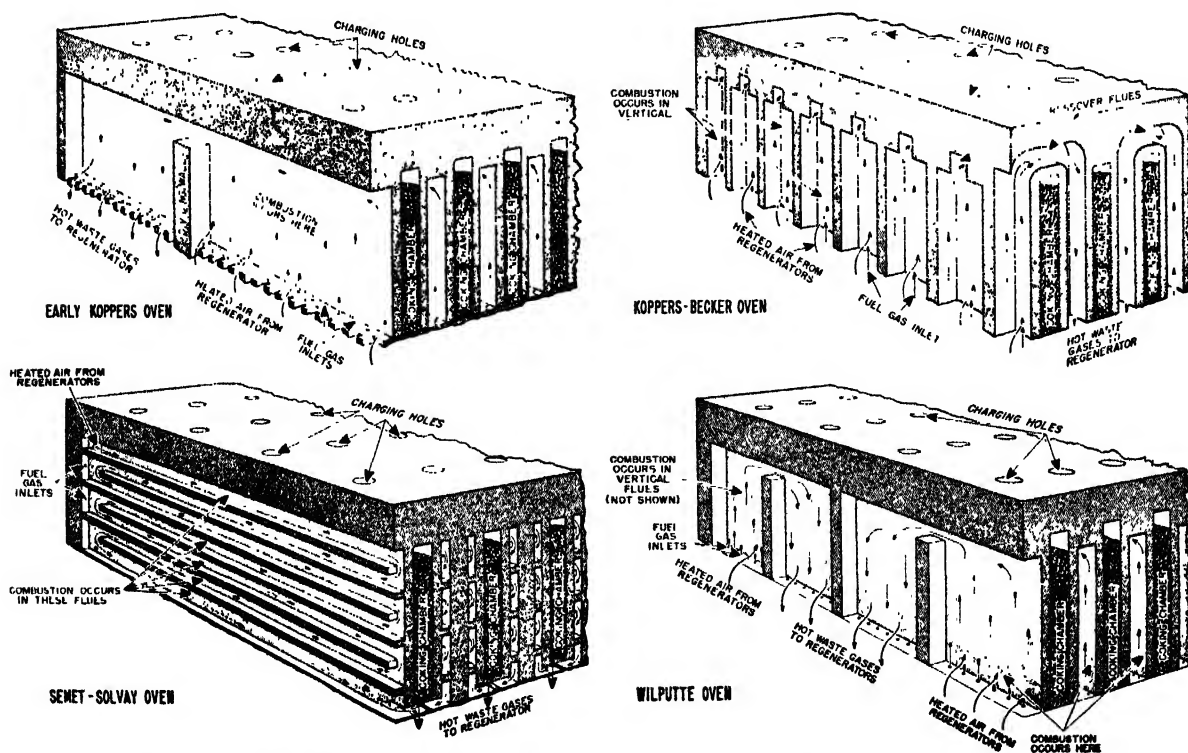


FIG. 4-3. Schematic representation of the differences in firing methods employed in the four most common types of coke ovens. Individual flues are not shown, except for the Semet-Solvay oven. The firing procedures shown are for a single phase of heating which is reversed at the end of a specified period.

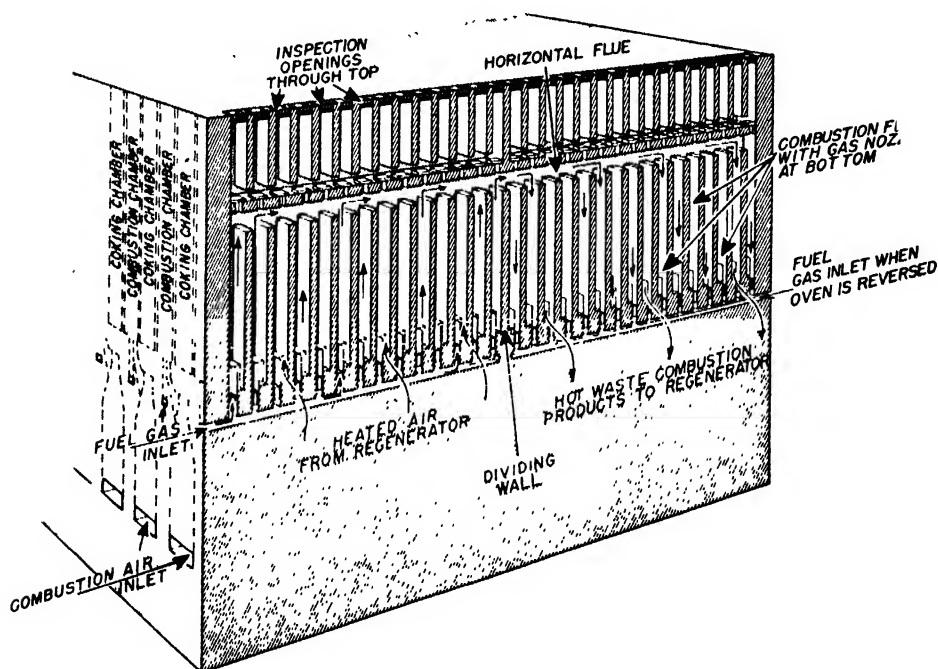
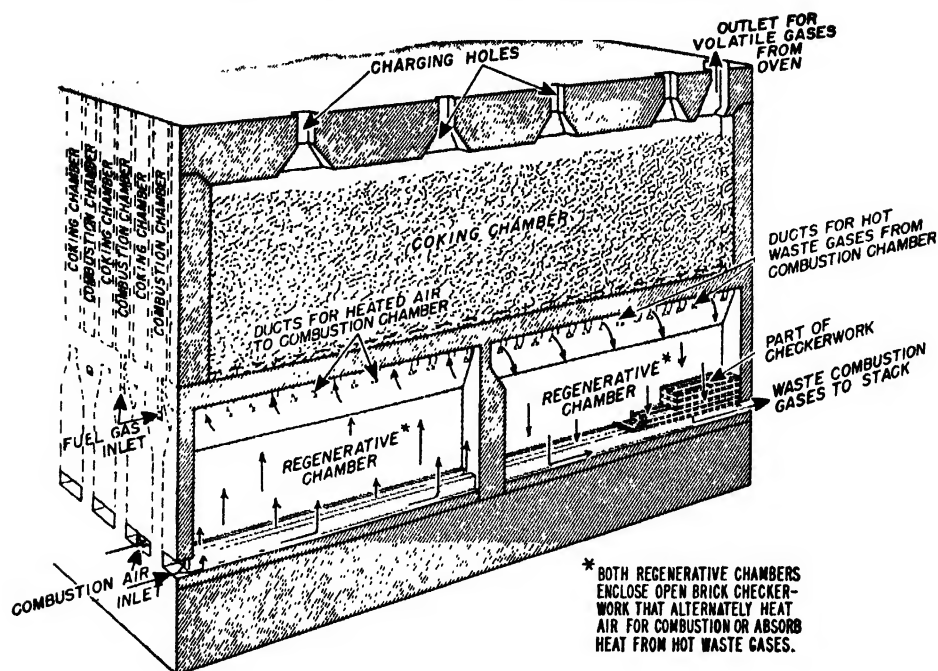


FIG. 4-4. (Above) Schematic section through coking chamber and regenerators and (below) section through combustion chambers, showing the paths of air, fuel gas and products of combustion in a Koppers 500-cubic foot coke oven. (See also Figures 4-3, 4-5 and 4-6).

oven is of the vertical-flue type with individual regenerative chambers (Figure 4-6). The heating chamber has a total of thirty vertical flues (Figure 4-5). They are provided with openings to the regenerative chambers, the fuel gas mains, and to a large horizontal flue on a level a little below the top of the coking chamber. A dividing wall near the middle of the oven separates

the heating chamber, except the horizontal flue, into two parts with sixteen vertical flues on the narrower end of the oven and fourteen on the wider end. Each end, approximating half of the oven, may thus be heated alternately, and in practice the reversals are made automatically every half hour for each battery of ovens by a reversing mechanism controlled by an automatic timing

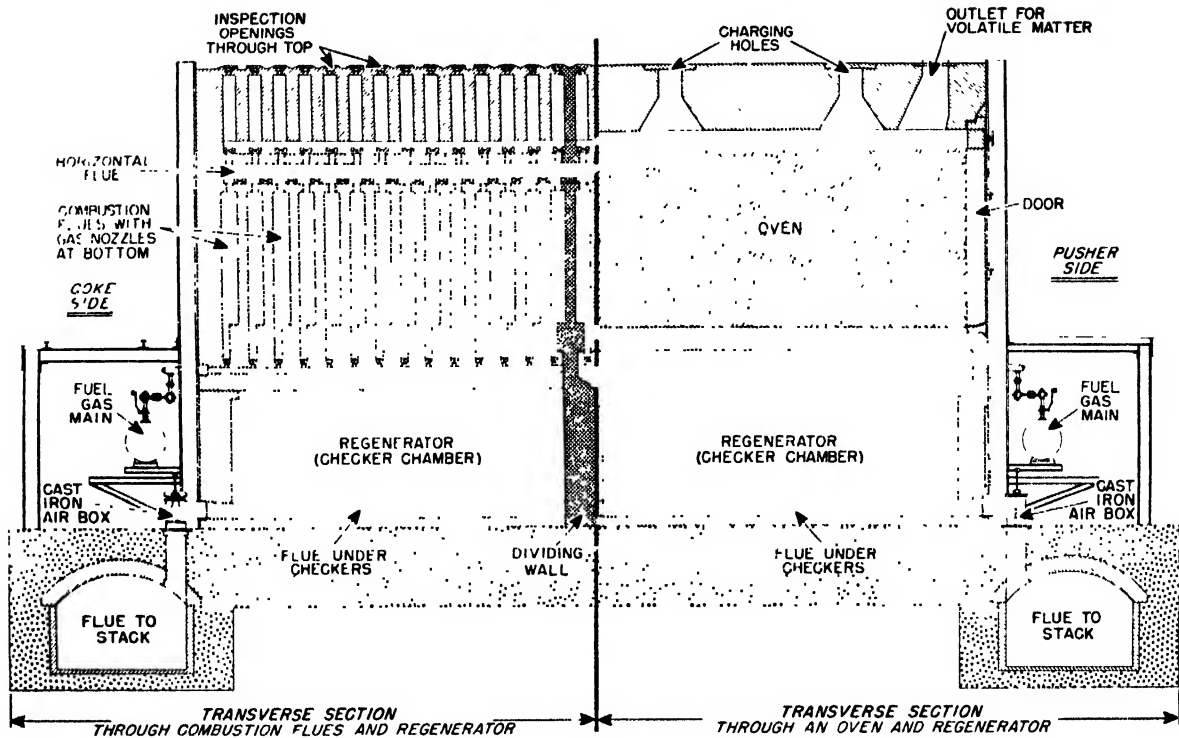


FIG. 4-5. (Above). Transverse section of Koppers regenerative single-divided coke-oven battery. Section at left is through combustion chambers, that at right is through oven chamber. See also Figures 4-3, 4-4, and 4-6 (Courtesy, Koppers Company, Inc.).

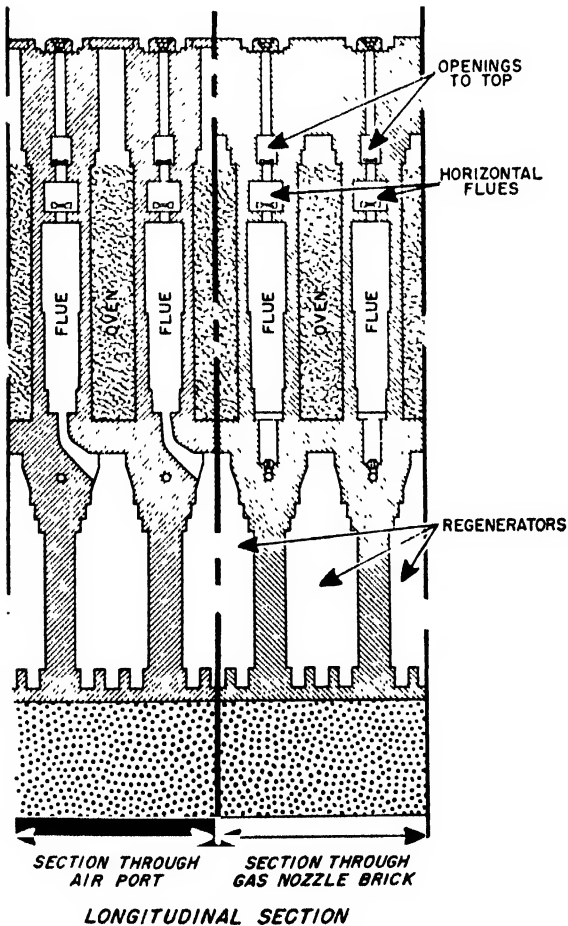


FIG. 4-6. (Left). Longitudinal section through part of a battery of Koppers regenerative single-divided by-product coke ovens. See also Figures 4-3, 4-4 and 4-5. (Courtesy, Koppers Company, Inc.).

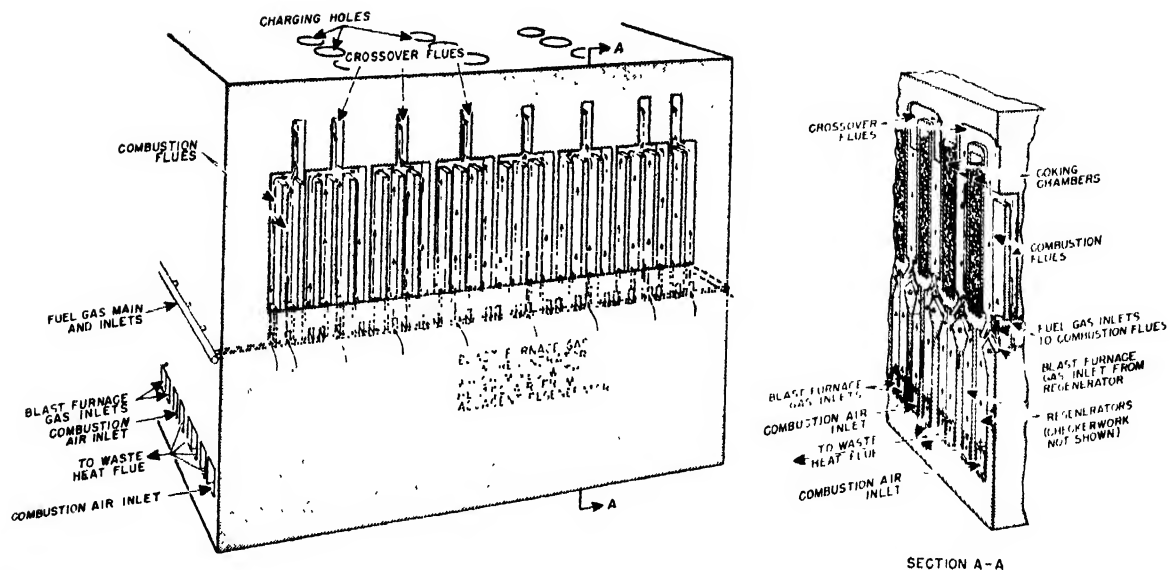


FIG. 4-7. Schematic sections showing paths of air, fuel gases and products of combustion in a Koppers-Becker combination coke oven. This illustration also shows arrangement for oven heating with blast-furnace gas.

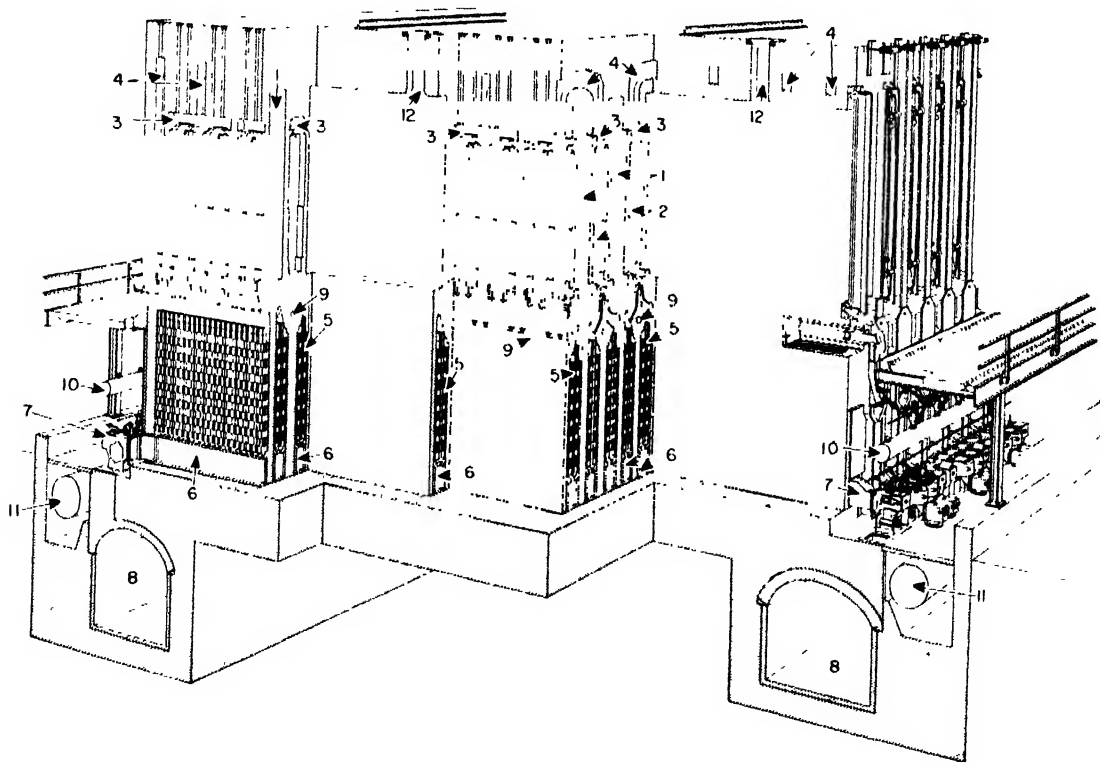


FIG. 4-8. General perspective cut-away section of Koppers-Becker combination ovens with gun-flue heating facilities (not the underjet type). 1. Oven chamber. 2. Vertical combustion flues. 3. Horizontal flues. 4. Cross-over flues. 5. Regenerators. 6. Oven sole flues. 7. Gas and air connections to waste-gas flue. 8. Waste-gas flue. 9. Gas ducts for coke-oven gas. 10. Oven gas main. 11. Blast-furnace gas main. 12. Charging holes. (Courtesy, Koppers Company, Inc.).

THE MAKING, SHAPING AND TREATING OF STEEL

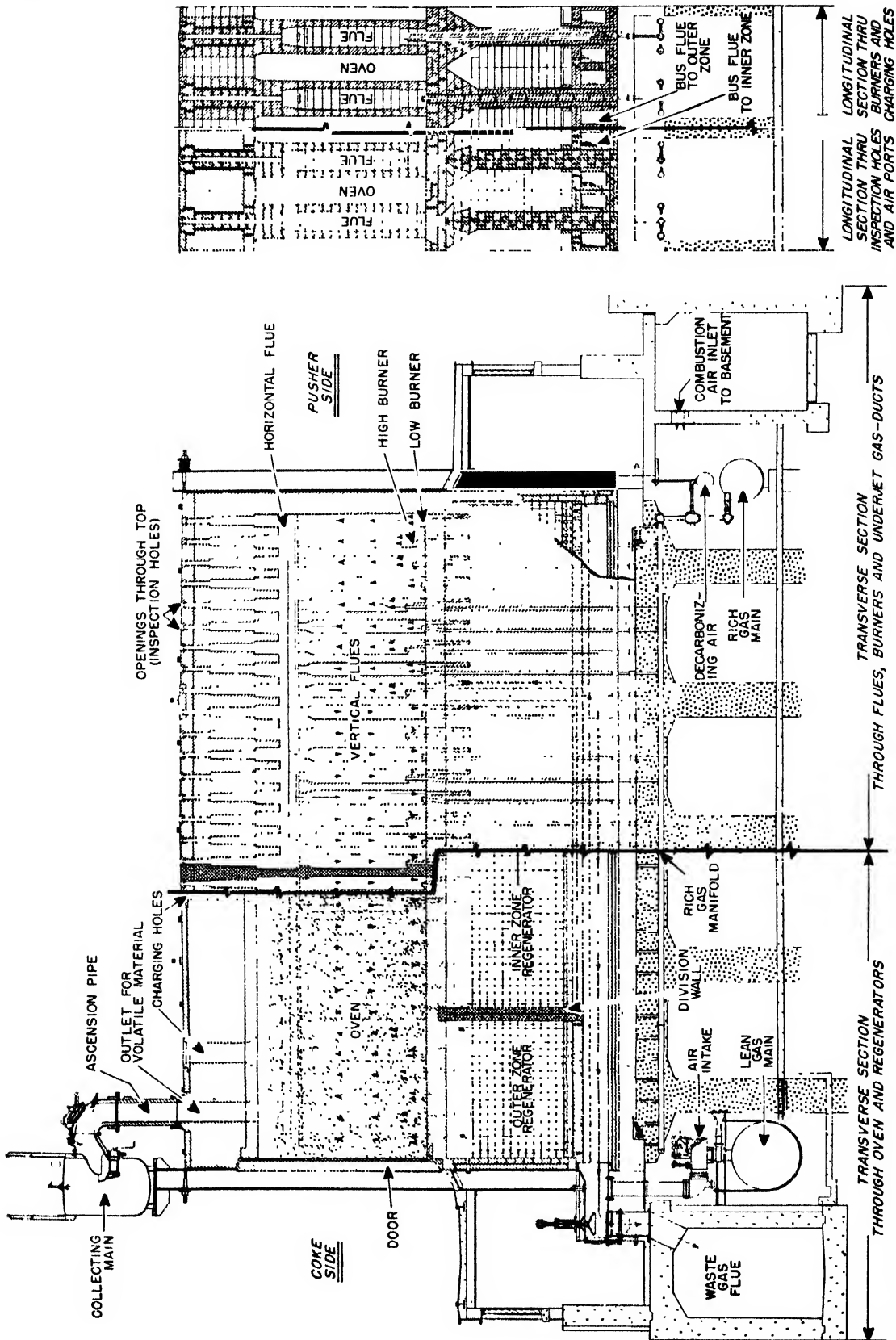


FIG. 4-9. Sections through a battery of Wilputte underjet, combination, by-product ovens, designated as "double-divided" ovens, having two outer zones and one double inner zone in the heating system. The rich gas and lean gas referred to on the drawing are coke-oven gas and blast-furnace gas, respectively. See also Figures 4-3 and 4-14 (Courtesy, Wilputte Coke Oven Division, Allied Chemical and Dye Corp.).

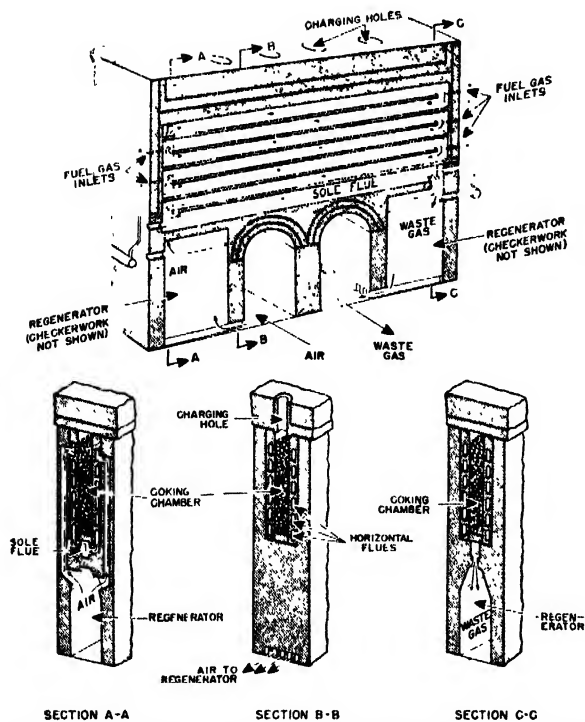


FIG. 4-10. Schematic sections showing paths of air, fuel gas and products of combustion in a Semet-Solvay coke oven. Upon reversal, the direction of the arrows in the upper drawing that indicate direction of flow of air and waste gas are reversed.

device. Two large underground flues, one on each side, extending along both sides in front of, and parallel to, the battery and connected to the checker chambers by

cast-iron air boxes, provide for escape of the products of combustion. These flues lead to a stack about 200 feet high at one end of each battery to furnish the draft necessary to draw the gases through the heating system. An idea of the magnitude of the structure may be gained from the fact that a single battery of sixty-four ovens contains the equivalent of about 2,500,000 nine-inch brick.

The Koppers-Becker Oven—The Koppers-Becker oven employs a different flue arrangement whereby the gas is burned on an entire wall simultaneously (both pusher and coke side). The products of combustion from groups of two or more vertical flues of the "on" walls in which the fuel gas is burning enter short bus-flues and are thence conducted over the top of the oven through cross-over flues to a companion series of bus-flues whereby the entire "off" wall is simultaneously conducting waste gas to the regenerators. On reversal, the opposite conditions obtain. Since the flues in each wall (coke side to pusher side) are connected only to the flues in its companion wall, there are no crossover flues over every other oven and the battery thus is limited to an uneven number of ovens (see Figures 4-7 and 4-8).

The Wilputte Oven—The Wilputte oven is known as a double-divided oven, having two outer zones in the heating system and one double inner zone. In this oven, the gas is alternately burned upwards in the two outer zones with the products of combustion being carried down through the double inner zone and, on reversal, burned upwards in the double inner zone with the products of combustion being carried down through the two outer zones (Figures 4-3, 4-9 and 4-14).

The Semet-Solvay Oven—The Semet-Solvay installations have oven batteries employing horizontal heating flues (the other three types use vertical heating flues), wherein fuel gas is introduced at one end and waste gas drawn off at the other end, with regular reversal of flow (Figures 4-3 and 4-10).

SECTION 4

CONSTRUCTION AND OPERATION OF BY-PRODUCT OVENS

For purposes of discussion, the different principal parts of a by-product oven will be considered in the following order; i.e., Coking Chamber, Heating System, Oven Doors, Gas-Collecting System, and Accessory Equipment.

Coking Chamber—The dimensions of the coking chamber are in each case a compromise of many correlated variables that will best suit the expected operations and produce the highest grade of product within practical limits. Past experience with coals of similar properties is the best guide, as there is no well defined academic method of arriving at definite oven dimensions for specific coals.

In general, the average dimensions of the present day oven are from 10 to 14 feet in height, 30 to 43 feet in length, and 15 to 19 inches in average width. The ovens are narrower on the pusher side and have a taper of from 2 to 4 inches, according to the expanding or contracting properties of the coals to be coked. The ends of the oven are closed with brick-lined removable doors. The side walls, or liners, are built of first-quality silica-brick shapes set in silica mortar that forms a ceramic bond at the higher temperatures. The oven floor may be of first-quality clay blocks, though silica bricks have been used for oven floors for many years.

Inasmuch as most of the heat for the coking process is conducted through the oven liners, the coking really starts at the side walls and progresses through to the center of the coal charge. There is no fusing together of the charge at the center as can be seen in Figure 4-11. This feature limits the length of any piece of coke to half the width of the oven minus any shrinkage. The structure of the coke mass at the end of the coking period is somewhat similar to two parallel slabs of irregularly interlaced pieces of coke that may be pushed from the oven by the pressure applied by the pushing ram with very little lateral pressure on the oven side walls. However, the walls must have sufficient structural strength to resist a high lateral pressure in case the interlaced structure of the coke mass is broken up for any reason during pushing. All modern ovens, from a structural point of view, also are designed to prevent, as far as possible, leakage of gases in either direction through the brickwork between the oven retort and the heating flues.

For a brief period after the establishment of this by-product coke oven in the United States, there was a trend toward using narrow ovens, but the modern trend for most plants is towards use of wider ovens. There is also a trend toward more taper, pusher to coke side, in the oven chambers. This increased taper permits the use



FIG. 4-11. View of coke being pushed from oven, showing central line of cleavage and block-like structure of the coke.

of a wider range of coals of varying characteristics, allowing for greater flexibility in making up blends for charging ovens whenever changing blends becomes necessary to utilize coals other than those normally used. In this way irregular blends may be processed with less danger of damage to the ovens or excessive operating delays by causing stickers.

Stickers are ovens that cannot be pushed in the normal manner due to excessive expansion during the processing, or coke of insufficient structural strength to hold the interlaced mass during pushing, thereby developing a lateral component of the pushing force that greatly increases the side-wall friction.

The coal is charged into the oven through charging holes provided in the roof of the oven. The oven retort and the heating system are designed for a coal charge of definite volume, having a level top surface a definite distance below the oven roof, usually one foot. The number of charging holes and the physical characteristics of the coal have a definite bearing on the time required for charging the oven with coals. A minimum charging time is particularly desired for many reasons. To prevent escape of gases from the oven during charging, it has now become standard practice at most plants to charge the oven on the main. As charging on the main is accomplished by steam-jet aspiration, which puts the oven retort under suction during the charging and leveling period, it is impossible to prevent the introduction of some air into the gas recovery equipment.

Excessive leveling tends to pack the coal along the top of the coal charge, particularly under the charging holes thus increasing the bulk density and heat requirements at this area. This has particular significance when using expanding coals. Excessive leveling may also cause local erosion of the oven wall.

Heating System—The present-day heating systems of the more prominent ovens in this country fall into two general classes; the **gun-flue type**, shown in Figures 4-12a, b and c, and the **underjet type**, shown in Figures 4-13 and 4-14. In the gun-flue type the gas is introduced through a horizontal gas-duct extending the length of each wall a little below the oven floor-line. Short connecting ducts lead vertically upward to a replaceable nozzle-brick at the bottom of each of the vertical flues. In the underjet type, the fuel gas is introduced into each flue from the gas distributing piping in the basement of the battery through a circular gas duct built integrally into the regenerator division and flue supporting walls. Each of these separate burner pipes is equipped with an orifice and metering pin to permit control of gas to each flue.

There are various designs of both general types, all attempting to heat the coal as charged at a controlled rate and temperature, uniformly from end to end of oven, and from bottom to top of charge (with the exception of the top few inches which may be held slightly lower for the better control of coal-chemical recovery), and at the lowest rate of Btu's per pound of coal carbonized. In the Koppers oven of 1917, as previously described, the coke-oven gas, stripped of various coal chemicals, is burned in all flues on the pusher side at once, the products of combustion passing into the horizontal flue and then down through the coke-side flues and regenerators to the stack. On reversal, the air is preheated in the coke-side regenerators and burns the gas in the coke-side vertical flues. The products of combustion are conducted through the horizontal flue to the pusher-side flues where they are carried down through the pusher-side regenerators and to the stack. In the Koppers-Becker ovens the arrangement of the regenerators beneath the ovens, as shown in Figure 4-15, is such that the ovens under the cross-over ducts have alternately one wall with gas on and one with gas off, while the ovens not under cross-over ducts have alternately both walls with gas on, and then both with gas off. This regenerator arrangement is in the interest of having a minimum number of walls operating under high differential pressure. Considerable care is taken in design to have the lowest possible differential pressure between the flues and ovens and between adjacent regenerator walls in order to prevent cross flow of gases at these locations.

Another point in flue design of particular importance is the relative position of the gas inlet in the vertical flue in relation to the air port, and the manner in which the gas and preheated air are caused to be mixed. Excessive turbulence will result in too sharp combustion, and a short intense flame will cause local over-heating at the base of the vertical flue.

For many years, oven operators and designers have felt that control of the rate of flame propagation and flame length in the vertical flues was desirable. Many suggestions were considered, the principal one of which was waste-gas dilution. The difficulties and expense associated with the use of waste gas that had been permitted to cool below the dew point were prohibitive. A system of waste gas recirculation by jet aspiration has been used in the Koppers-Becker design (see Figure 4-16). By this device a fixed amount of hot, waste gas is mixed with the incoming fuel gas. By this recirculation used in the Koppers-Becker design (see Figure 4-16), a fixed amount of hot waste-gas is mixed with the incoming fuel gas. By this recirculation the flame length may be controlled without purification of the waste gas and without having to heat the diluent as would be required with external mixing. The ratio of recirculated

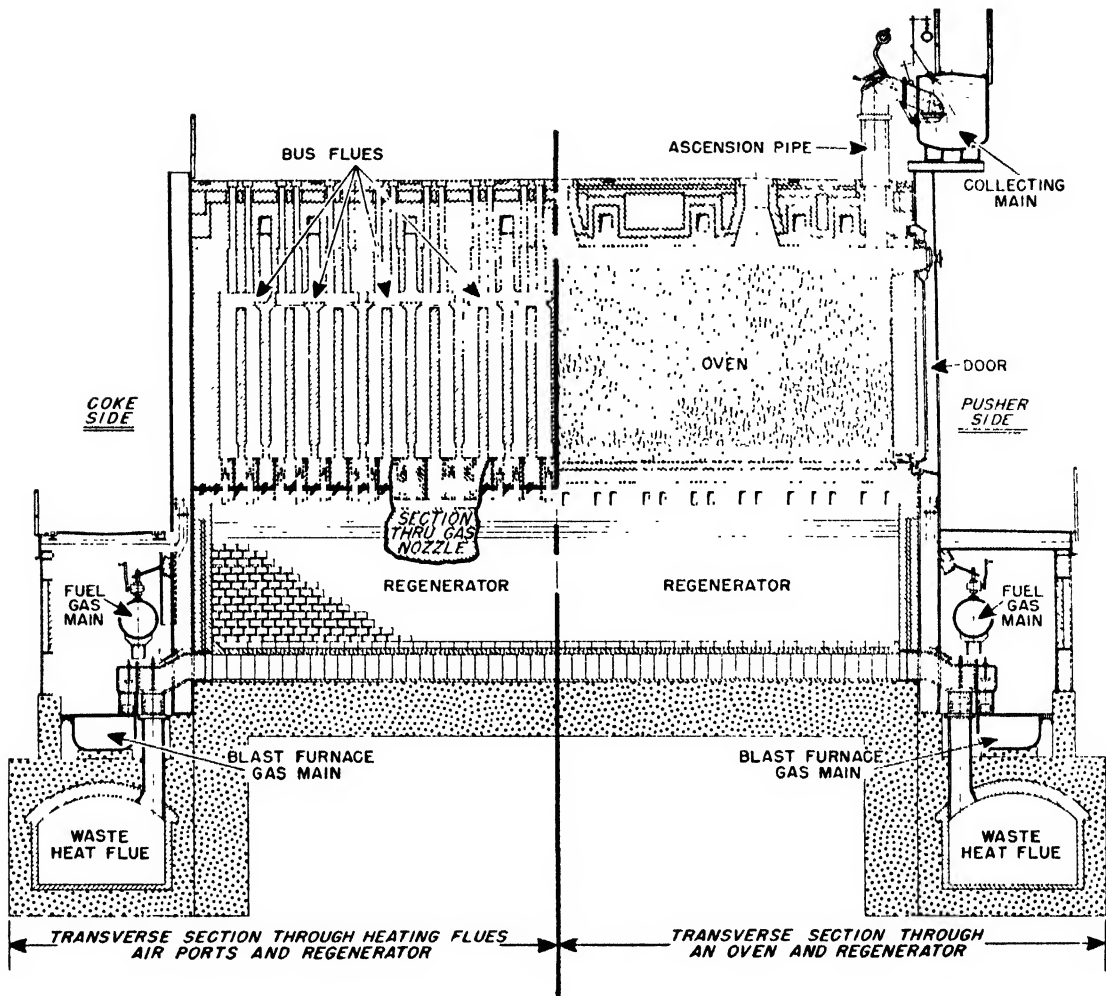


FIG. 4-12a. Transverse sections through a Koppers-Becker combination gun-flue type by-product coke-oven battery. (Courtesy, Koppers Company, Inc.).

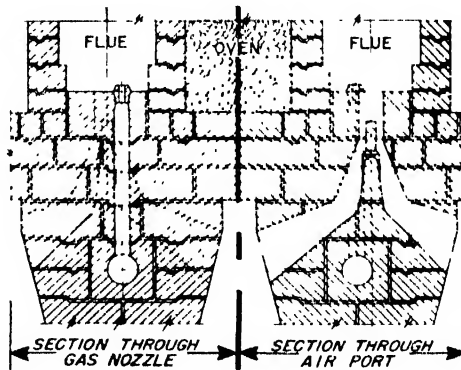
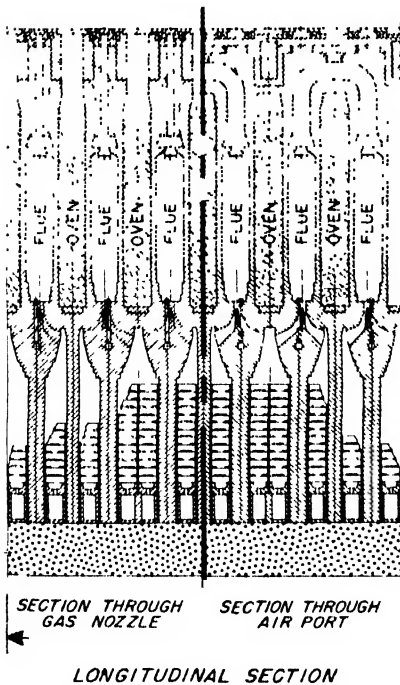


FIG. 4-12b. (Above) Enlarged sections through gas nozzles and air ports of a Koppers-Becker combination gun-flue type of by-product coke oven, showing detail of part of Figure 4-12c. (Courtesy, Koppers Company, Inc.).

FIG. 4-12c. (Left) Longitudinal section (left) through gas nozzles and (right) through air ports of a portion of a Koppers-Becker combination gun-flue type by-product coke-oven battery.

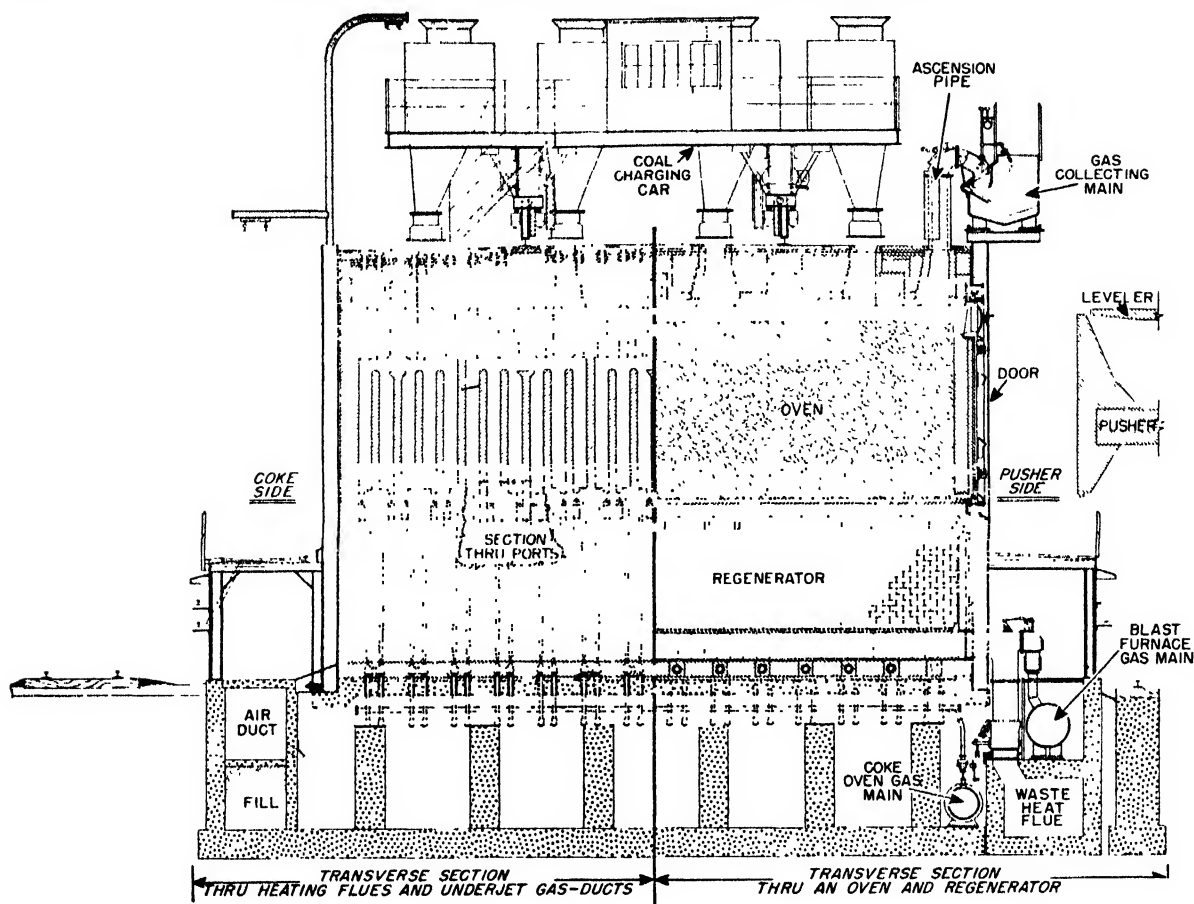


FIG. 4-13. Transverse sections through a Koppers-Becker underjet-fired low-differential combination by-product coke oven. See Figure 4-15 for longitudinal section of part of a battery of ovens of this type. (Courtesy, Koppers Company, Inc.).

waste-gas to oven gas is controlled by the orifice size and the fuel-gas pressure, and normally approximates a one-to-one mixture.

Waste-gas recirculation also prevents the accumulation of carbon in the underjet gas-ducts, as the contained carbon dioxide and water vapor both tend to inhibit carbon deposition. In ovens not recirculating waste gas, air must be introduced into the gas ducts on the "gas off" periods for decarbonization.

A feature of the Wilputte oven, shown in Figures 4-3, 4-9 and 4-14, especially those of over 10 feet in height, is the high-low burner construction. Low burners in alternating flues with adjacent high burners prevent overheating at the bottom of the flues and thus tend to give a better vertical distribution of heat.

In the gun-flue-type oven, the changes of the nozzle brick which regulate the gas flow to the various vertical flues is done from the top of the battery. Immediately over each vertical flue, a duct is provided from the horizontal flue through the battery top and ending at a removable flue-inspection cap flush with the oven top. When necessary, the nozzles are removed and replaced through these ducts by long rods especially designed for this purpose. In the underjet-type oven, the same flue extensions are provided for flue inspection, but the gas flow is regulated by changing the orifice or metering pin in the accessible external fuel-gas piping in the oven basement.

Where blast-furnace gas or other lean gases are used for oven heating, the regenerator system must be designed so that both the air and gas are preheated. In the Koppers-Becker oven, the regenerator chambers are so arranged that the fuel gases and waste gases are not regenerated in adjacent chambers but are separated by an air-regenerating chamber. With this design, the effect of leakage across the high-differential pressure wall is minimized, as the incoming air and gas are under the same pressure and traveling in the same direction, and the high-differential-pressure wall is between the incoming air and the outgoing waste-gas.

When a lean gas, such as blast-furnace gas or producer gas, is used for oven heating, supplementary heating with a higher calorific value gas may be needed in order to maintain coke production at as high a rate per oven operating hour as when firing with straight high-Btu gas. It is not practical to do this mixing before regeneration of the gas as the rich component of the mixed gas will be partially cracked passing through the regenerators and will cause an objectional deposition of carbon. To overcome this cracking tendency, the enriching gas is externally diluted to the desired degree with lean gas and introduced through the customary fuel system.

With the introduction of the underjet oven and the accurate proportioning of fuel gas necessary to each vertical flue, it is important that the reversing cocks,

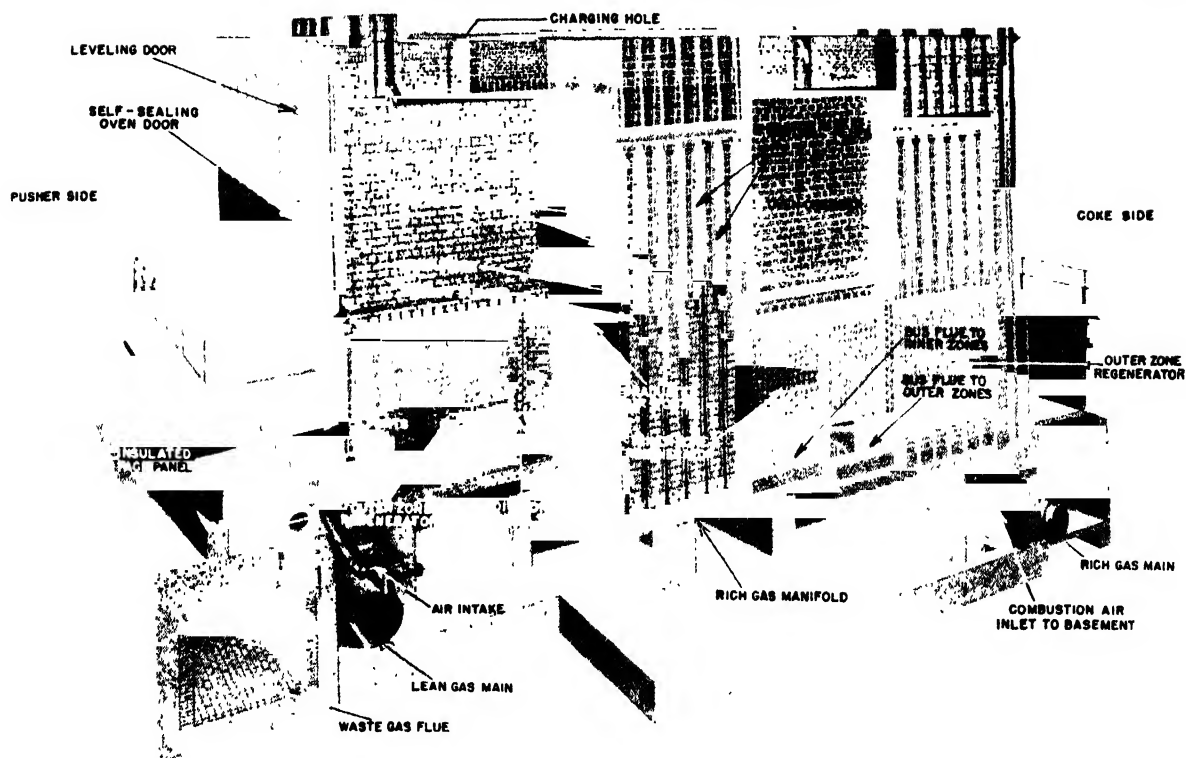


FIG 4-14. General perspective "cut-away" drawing of a Wilputte combination underjet-fired by-product coke oven. The "rich" fuel gas referred to is coke-oven gas; the "lean" fuel gas is blast-furnace gas. Compare with Figures 4-3 and 4-9. (Courtesy, Wilputte Coke Oven Division, Allied Chemical and Dye Corp.).

lines, and orifices or metering pins be clean at all times. This is accomplished by cleaning the fuel gas with electrical precipitators located at each battery of ovens.

After the electrical precipitators, and ahead of the battery fuel-gas mains, the gas is heated to assure its maintenance above the dew-point. On some of the older designs of the gun-flue type, the fuel-gas mains located in the alleys on both the pusher and coke side were insulated to limit the condensation in the headers and to keep the gas at a uniform temperature throughout the length of the header. This temperature control is important as temperature change affects the density and specific heat of the gas and influences the uniformity of battery heating.

In the gun-flue-type oven, the air required for combustion is taken into the sole flue at the base of the regenerator chambers through an air box equipped to regulate the amount of air taken from the alleys on both sides of the battery. In this arrangement the temperature, the velocity and direction of the wind have a marked effect on the heating and must be compensated for by the heater. In the underjet type, the air for combustion is taken from the enclosed basement and is independent of atmospheric wind velocity, direction, and temperature.

In the Wilputte design of underjet ovens, the basements are sealed and kept at a constant air pressure (slightly above atmospheric) by sensitive controls, with only the air required for combustion entering the basement through a wind tunnel extending along the entire length of the battery. A fan of sufficient size delivers air to the wind tunnel. Spaced along the length of the basement are suitable openings equipped with regulating louvers to distribute the air uniformly throughout the

basement. In this design, only the air required for combustion is available for basement cooling.

In Koppers-Becker underjet-type ovens of a recent design for large batteries, the air for combustion is introduced in much the same manner as that just described, with the exception that from two to three times the air required for combustion is forced into the basement, the excess finding its way out through suitable openings around the buckstays on the pusher side (the wind tunnel being on the coke side of the basement). In this arrangement, from two to three times the air required for combustion is circulated through the basement for cooling. In still later designs of this type of oven, only the amount of air required for combustion is introduced into the basement.

The effects of faulty heating may be serious, not only to the quality and quantity of the coke and coal chemicals produced, but also to the ultimate life of the ovens. The most serious damage to the ovens is local overheating beyond the critical temperature of the exposed brick, when fluxing or slagging occurs. This usually occurs in zones that are not readily accessible for repair. The advantages of an even and controlled heat throughout the oven cannot be overemphasized and remains a constant challenge to the engineer.

When a new battery, or an old battery that has been allowed to go cold, is to be put into operation, great care must be taken in bringing the battery up to operating temperature. Since the major portion of the battery is of high-grade silica-brick construction, and silica brick has a high coefficient of thermal expansion at lower than operating temperature, it can readily be seen that the heating up must be slow enough to insure maximum temperature equalization throughout the entire battery

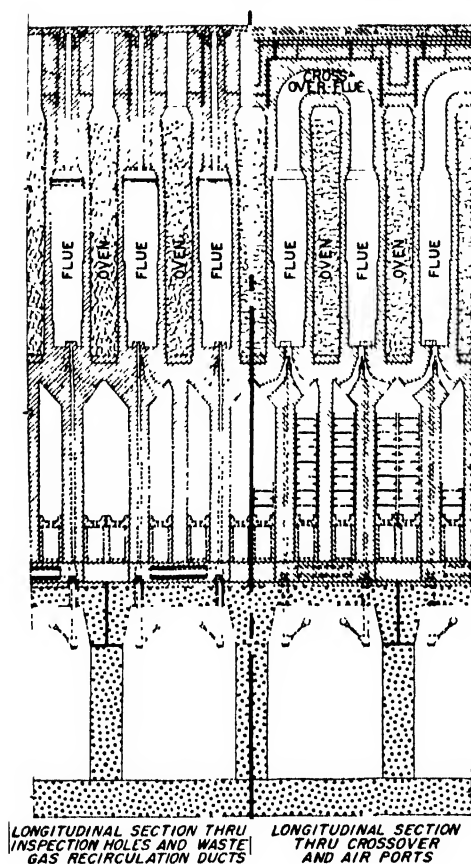


FIG. 4-15. Longitudinal sections through part of a battery of Koppers-Becker combination underjet low-differential by-product coke ovens of the type illustrated in Figure 4-13. Figure 4-16 shows further details. (Courtesy, Koppers Company, Inc.).

structure. In actual practice, the heating up from cold to carbonizing temperature takes from five to seven weeks. This same practice obtains in reverse when it is desired to allow a battery of ovens to go out of operation and cold. The cooling of the battery is a project to be undertaken only after full appraisal of all alternatives.

The usual method of heating up a battery is first by installing a temporary brick door at each end of each oven and burning gas in a burner inserted in the oven through an opening provided in the temporary door. The products of combustion are allowed to enter the heating system at the horizontal or bus-flue elevation through suitable openings provided for this purpose, which are later plugged and sealed. The hot gases during the drying-out period are vented down through the vertical flues and regenerators to the stack flue and stack. When the flues become hot enough to ignite fuel gas on reversal, the gas is introduced through the normal channels. Where gas is not available for heating up, coal or coke may be used by substituting a brick bulkhead for the oven door, leaving openings for firing and ash removal. The same type of false hearth is used to protect the oven liner-brick. After heating up, the false hearth and bulkhead are removed, and the oven door is installed.

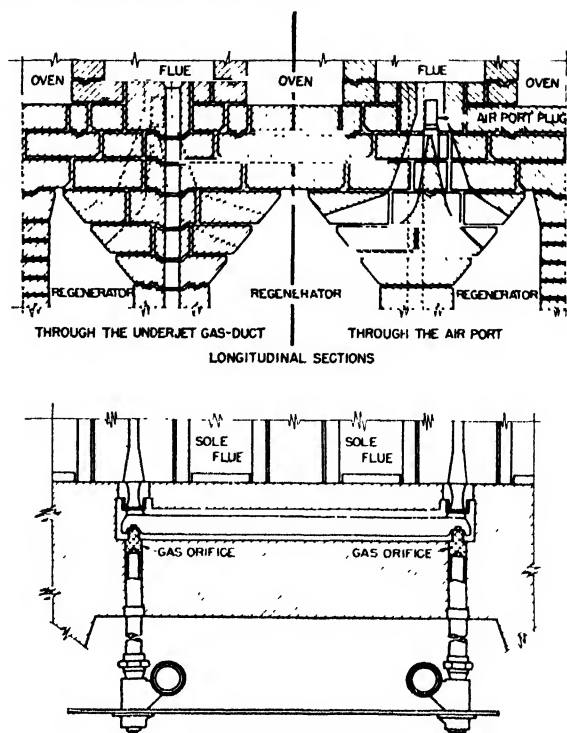


FIG. 4-16. (Top) Enlarged detail of an underjet gas-duct and air-port, and (Bottom) section through waste-gas recirculation duct of the Koppers-Becker by-product coke ovens shown in Figure 4-15. (Courtesy, Koppers Company, Inc.).

Many controls of the heating system are fixed in that they are built into the oven structure. The means of variable control are comparatively simple and to a great extent similar in all the modern ovens of the same type. In general, they consist of gas-pressure control in the headers; the size of nozzle, orifice, or metering pin used in the connections to the individual flues; stack-draft controls for the main stack-flue and for the individual ovens; combustion-air controls; various temperature controls and pyrometers.

Oven Doors - As has been mentioned, the ends of the oven are equipped with removable refractory-lined doors. After a coal charge is fully coked and the oven dampered off the main, suitable equipment on both the pusher and coke sides remove the doors and hold them during the pushing operation. After pushing, the doors are replaced and sealed preparatory to recharging the oven.

Until recent years, the method of sealing the doors was to trowel and smooth ground "mud" into a V-shaped opening between the door and the door jamb. In recent years there have been developed self-sealing doors that do not require luting. In principle, the self-sealing door has finally developed into a spring-loaded door that depends on a metal-to-metal contact between the door and the continuous machine-surfaced cast-iron jamb. The Wilputte design of self-sealing door is shown in Figure 4-17. The sealing edge of the door is carried by a flexible frame, and the door assembly is so designed that a powerful spring between each locking bar and the door forces the sealing edge against the metal door jamb with considerable pressure, thus preventing the escape of volatile products from the oven. The pusher-side door

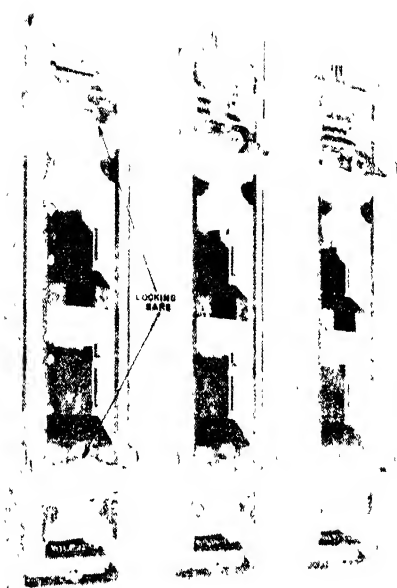


FIG. 4-17. Wilputte design of self-sealing coke-oven doors, showing spring-loaded bars that maintain metal-to-metal contact between the door and jamb. (Courtesy, Wilputte Coke Oven Division, Allied Chemical and Dye Corp.)

is also equipped with a somewhat similar small self-sealing door to permit leveling of the coal as charged and removal of any excess coal.

Oven-door expense is a large factor in over-all oven repair and maintenance costs. This expense can be controlled by careful design of door-handling equipment and strict adherence to good operating practice. The lining of the door is usually sectionalized and made up of clay-brick shapes. In some plants the doors have a monolithic lining of lumnite cement made with an aggregate of various grades of crushed brick and ganister which give good service. The thickness of the lining, and the position of the inside face of the lining relative to the end vertical flue of the oven is important as it influences the heating of the ends of the coal charge.

Gas-Collecting System—The oven may be equipped with one or two offtakes to carry off the volatile products liberated in the coking process. Where one offtake is provided, it is through the roof of the oven at either end of the oven, and where two are provided, there is one at each end of the oven. In either case the volatile products pass through the duct or ducts in the oven top and enter a refractory-lined standpipe which in turn is connected to a collecting main through a damper-valve. Between the damper-valve and the oven the standpipe is equipped with a cap valve, marked "elbow cover" in Figures 4-18 and 4-19, which, when open, vents the oven to the atmosphere. The use of a double or single collecting main is still a question of no little debate among coke-plant operators, each system having its advantages.

The damper valve is usually a water-sealed valve, so designed that the cooling spray furnishes the seal when the damper valve is in the closed position, the excess spray overflowing into the collecting main. With this arrangement, the cooling spray is always on. The water, called **flushing liquor**, used in damper box and collecting-main flushing, is the condensate from the volatile

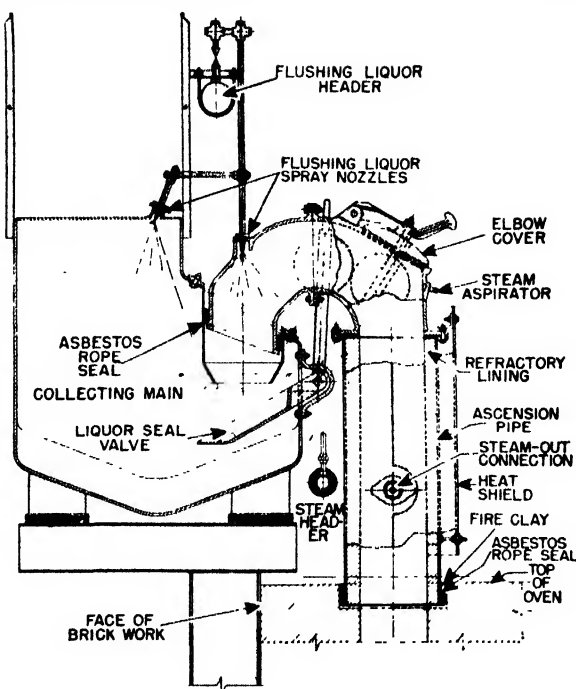


FIG. 4-18. Water-sealed damper valve of the design employed by Koppers Company, Inc., through whose courtesy this drawing is reproduced.

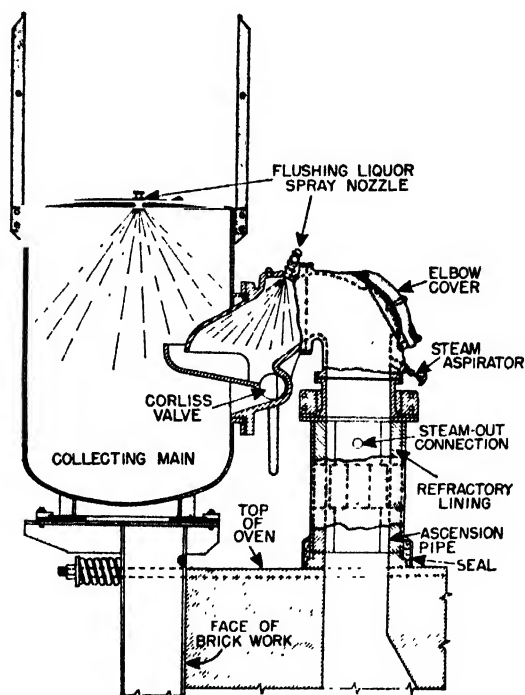


FIG. 4-19. Wilputte design of water-sealed damper valve, incorporating a Corliss-type valve in the bottom of the damper box. (Courtesy, Wilputte Coke Oven Division, Allied Chemical and Dye Corp.)

products driven off in the coking process. This water is later processed by the coal-chemical division for the recovery of ammonia. Figure 4—18 shows a design of water-sealed damper valve as built by Koppers Company, Incorporated. The Wilputte design, which may be seen in Figure 4—19, is somewhat different in that the valve is sealed by closing a Corliss valve in the bottom of the damper box which permits the spray to fill the box, thus sealing the damper. In addition to the wet type dampers, two dry type (not spray sealed) are in use at various plants; namely "dry butterfly" and the "mushroom" valve. All ovens of a battery are thus connected to a single or multiple pressure-equalized collecting main on either or both sides of the battery.

The function of either the single or double collector main is not only to collect the gas from the ovens but also to maintain at all times an accurately-controlled pressure in the oven during the coking process. Pressure in the oven during coking has a pronounced effect on the coke and coal chemicals. The pressure in the collecting main is usually kept at a point that will give about 1 mm. water-gage pressure at the bottom of the oven at the end of the coking period. It should not vary over plus or minus 1 mm. of water-gage pressure. The pressure is controlled by a regulator, usually of the Smoot or Askania type, located in the connection between the collecting main and the suction main which carries the gas to the coal-chemical recovery units. The collecting main also serves as a receiver for the products condensed from the gas by the flushing liquor.

Recent installations of Koppers-Becker underjet batteries have been designed for control of the temperature in the gas space above the coal charge. This has been accomplished by recirculating cooled raw coke-oven gas across the top of each oven above the coal charge by an artificially induced differential pressure between the pusher-side and coke-side gas-collecting mains. This differential pressure is provided by variable speed fans installed in the cross-over main connecting the two gas-collecting mains. On some predetermined cycle the dif-

ferential pressure is reversed to keep the pusher-side and coke-side temperatures in the oven tops in proper balance.

The use of a patented principle, in which gas from newly charged ovens (in which a relatively high pressure exists) is drawn off and passed through the space above the charge in ovens nearing the end of their coking periods (in which pressures are lower than average), is incorporated in ovens of recent Wilputte design. It is applied to ovens with double collecting mains, one main being used alternately with the other as a compensating main. In effect, one main is employed as a suction main on one side of the battery, with the other acting as a connection between the ends of all of the ovens on the other side of the battery, to assist in equalizing pressures in the spaces above the charges. These functions are reversed periodically. In addition to aiding in control of pressure in the ovens throughout a battery, the system also affords control of temperature throughout the tops of the oven chambers to prevent overheating of the gas space. The alternate use of each main as a suction main makes it possible to cause gas flow in either direction through the ovens near the end of their coking cycle so as to prevent overcooling in any particular area. In general, this system provides the same benefits as that described in the preceding paragraph, but by a different method.

The gas, on entering the damper box, or, in the case of dry dampers, the collecting main, is shock cooled with a liquor spray which causes the first precipitation of tar from the gas. The amount of liquor sprayed is important, as it controls the temperature to which the gas is cooled. Too little cooling allows only the heavy tars to precipitate, which may cause pitch deposits in the damper box and collecting main. Too much cooling may throw down too much light tar, and may result in naphthalene stoppage in the primary coolers or difficulty in tar dehydration. Some plants are equipped to permit segregation of the tar condensed in the collecting and suction main as opposed to that condensed in the primary coolers.

SECTION 5

ACCESSORY OVEN EQUIPMENT

Coal-Storage Bins and Charging Larries—To provide coal for the ovens, every modern plant has an overhead coal bin at the ovens of sufficient surge capacity to permit flexibility in coal preparation without interference to the scheduled uniform operation of the ovens. The number and size of bins required is determined for each individual plant. The location is determined from a study of each plant's layout of operating units.

All modern by-product coke ovens are designed to take a definite volume of coal per charge and are charged from a larry car operating between the overhead coal-storage bins and the ovens on a track supported by the battery top. Figure 4—20 is a view of a battery top, and shows a larry car.

The desired amount of coal for an oven charge is drawn from the storage bin, and is measured, usually by using either the track scales at the loading station, or by volumetric choke boxes on the larry car. Where track scales are used, the coal-bin gates over each hopper of the larry are usually hand operated. Where the separate hoppers of the larry, one for each charging hole, are equipped with volumetric choke boxes, the gates are usually linked together and power operated as all gates may be left open until the hoppers are full and the flow of coal is stopped by the choke boxes.

The larry car in principle is designed in connection with the number of charging holes per oven so that a predetermined quantity of coal is charged into the oven through each charging hole, the discharge from each hopper being independently operated, either manually or by power.

Improvements in larry cars, particularly the method of discharging coal, have been directed toward making possible better charging practices. The aim has been to reduce the charging time; to reduce the number of passes of the leveling bar necessary for leveling; to make a smokeless charge; to prevent hanging up of the coal in the larry hoppers; and to make a uniform charge as regards bulk density.

The gravity-discharge larry and the mechanically-unloaded larry are the two main types in use at the present time. The gravity-discharge larry is equipped with conical-shaped hoppers, shear gates and drop-sleeve mechanisms. Vibrators attached to the hoppers and stainless-steel liner plates may or may not be provided, depending upon the physical condition and flow characteristics of the coal. With this type the coal charge flows by gravity into the ovens. There are two designs of mechanically unloaded larry cars in use at present; namely, the "turntable" and "screw discharge" types.

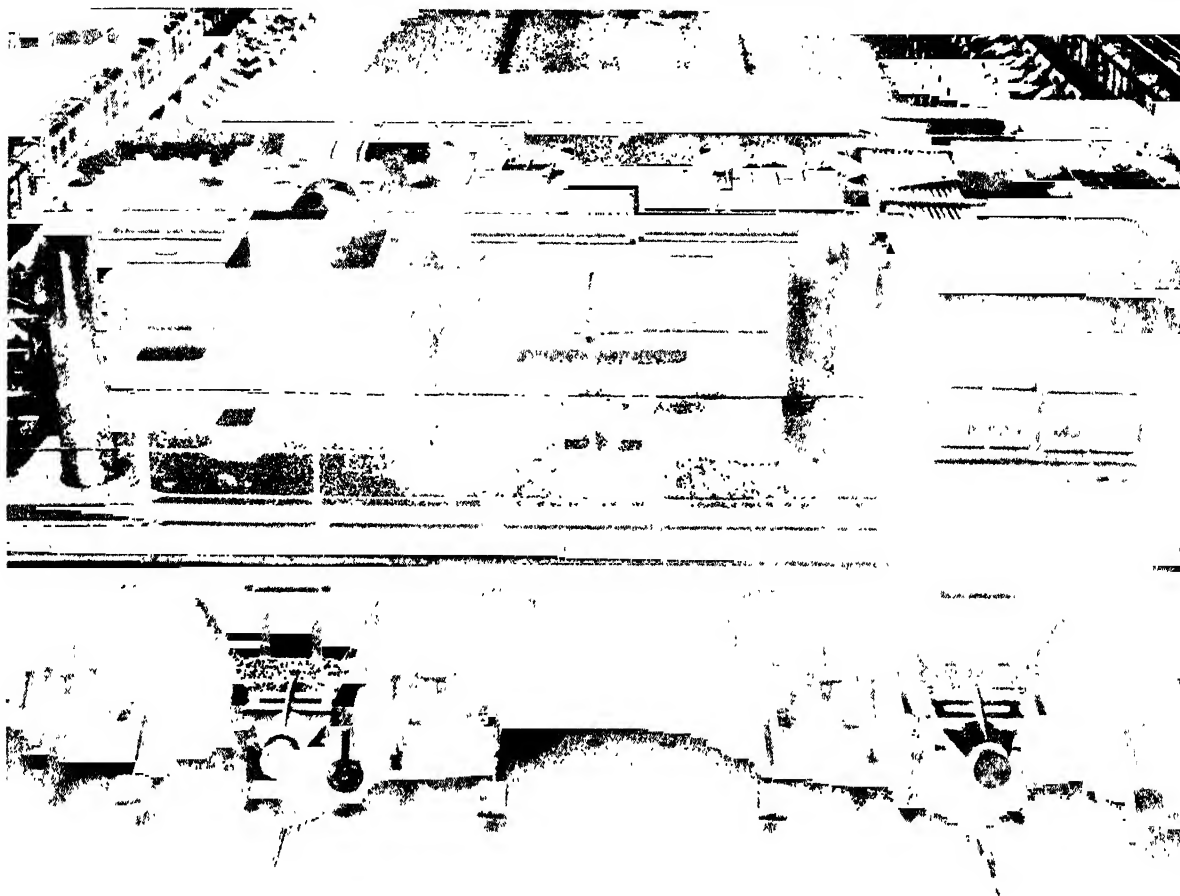


Fig. 4 20. Larry car equipped with screw conveyor for discharge of coal into ovens. The battery shown is fitted with double collector mains.

The turntable larry is equipped with a revolving table serving as the bottom of each hopper. The revolving table forces the coal through an opening in the side of the hopper leading to the shear gate and drop sleeve and thence to the oven. The screw-discharge larry is equipped with rectangular-shaped hoppers with the lower section tapering to a small opening directly over the screw trough. The trough contains the screw conveyor which forces the coal horizontally to the vertical drop sleeve and shear gate section. The hoppers are lined with stainless steel if adverse flow characteristics of the coal are encountered.

The screw-discharge larry gives a slightly better performance on wet or fine coal but considerable difficulty is encountered with any foreign matter in the coal, whereas the gravity-discharge larry and turn-table larry have less trouble with foreign matter in the coal. The gravity-discharge larry gives slightly more trouble with fine or wet coal than either of the other two types. The screw-discharge larry and the turn-table larry have much greater initial cost and greater total weight which is a source of potential damage to oven brickwork. The gravity-discharge larry is the least expensive initially and is the lightest of the three types. The screw-discharge larry is the most expensive to maintain, and the gravity-discharge, with vibrator and stainless steel hoppers, the least expensive to maintain.

Pusher-Side Equipment—The pusher-side equipment shown in Figure 4—21 is generally similar on all

types of ovens. The pusher may be a combination of three machines, a **pusher**, a **leveler**, and a **door extractor** so designed that it operates on a track parallel to, but independent of, the battery. In many plants, the door-handling equipment is a separate, self-propelled machine, operating on its own tracks.

The function of the door-extracting element of this machine is to remove and hold the pusher-side door during the pushing operation. It is either electrically and/or hydraulically operated from the elevated cab which contains all the machine controls. With self-sealing doors, an important feature of the door-extractor design is that its speed be relatively slow and easily controllable and the alignment be accurate to avoid damage to the sealing edges of the door. Contained in the head of the extractor is a mechanism for latching the door and for compressing the loading-springs.

The function of the pushing element (Figure 4—22, Diagram E) is to push the coke cake from the oven. This is done by an electrically-powered rack-and-pinion-operated ram, equipped with a suitable head that, when spotted immediately in front of the oven to be pushed, may be moved forward until all the coke has been pushed from the oven and through the coke guide into the quenching car.

The ram is equipped with a rider-shoe located about 5 feet behind the pushing head to support the ram during its passage through the oven. This rider-shoe is easily replaced as it is subject to considerable abrasion

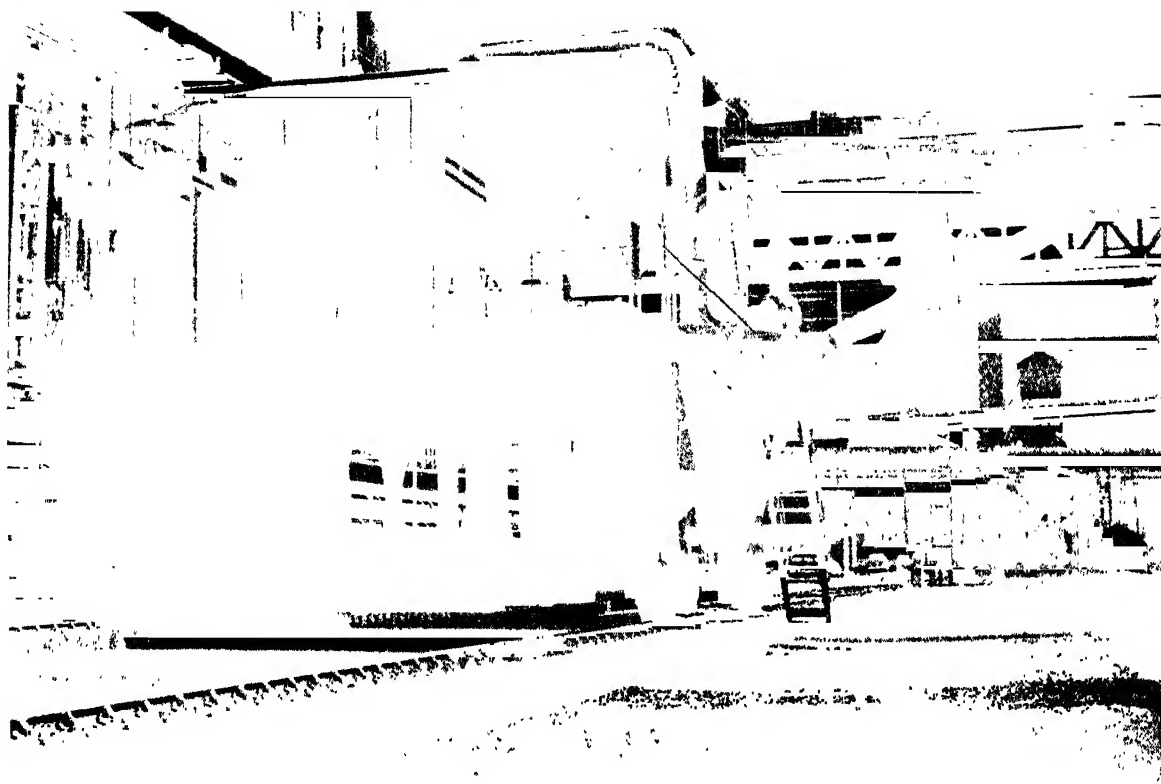


FIG. 4-21. Pusher-side equipment of a battery of by-product coke ovens, showing the machine which is equipped to level the coal charge in the oven and also push the finished coke out of the oven, as well as remove and replace oven doors before and after the pushing operation.

sliding over the brick floor of the oven, especially as there is always considerable coke breeze on the oven floor during the pushing and return. The ram is either a built-up box girder, "H" beam, or open-lattice electrically-welded structure. The open-lattice construction seems to be currently in favor as it is more easily repaired and resists the tendency to warp, due to more even cooling, regardless of wind direction. With the box-girder-type ram, it is necessary to have wind-and-rain guards for the ram in its retracted position. The pushing speed of the ram is about 60 feet per minute, and the maximum pressure exerted is controlled by overload relays to prevent damage to the oven brickwork.

The function of the leveling element (Figure 4-22, Diagrams B and C) is to level the coal charge in the oven, leaving a free-gas space below the roof of the charged oven. This is done by an electrically-operated leveling-bar carried by the pusher-machine structure in such a position that it may be introduced through a suitable opening in the top of the pusher-side door. The leveling-bar is a fabricated section consisting of two side-plates held apart by vertical plates spaced at from 2 to 4 feet which also serve as scrapers. When this bar is moved in and out of the oven, the scraper plates level the humps of coal into the valleys and, on removal from the oven, drag out all excess coal into a chute discharging into a receiving bin carried by the pusher machine. This excess coal is periodically dropped into a ground-level hopper for return to the coal bunker. At some plants, the excess coal is returned by a conveyor to the next oven charged.

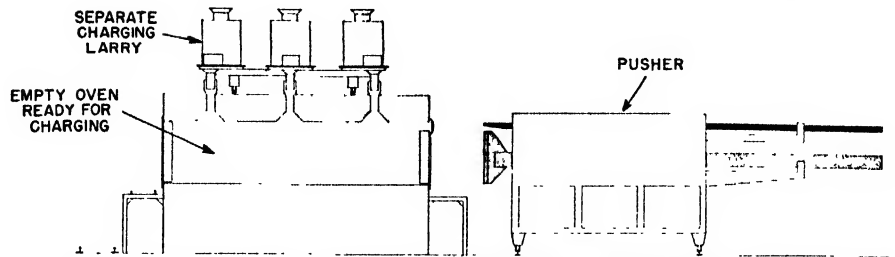
The various platforms, control rooms, and operating cab of the pusher machine should be designed to facilitate comfortable operation and accessibility for oiling, adjusting, and repair. Inasmuch as the three principal operations of the pusher machine always occur separately, one operator is all that is required. However, men known as door and jamb cleaners accompany the pusher and, in most cases, they also open and close the leveling door from the oven pusher-side bench and clean up any coal and coke spillage.

Where self-sealing doors are not used, it is necessary to provide equipment for the preparation of luting mud and for conveying the luting mud to the doors to be luted. The mud is usually prepared in the conventional-type wet pug mill from returned luting mud, coke breeze cleaned from the oven benches, and certain amounts of fresh clay. The mud buggies, or carriers for the pusher side, are usually electrically-operated and suspended from an overhead track, and are equipped with bins and platforms at two or three levels, according to the oven-door height, so that the luterman can easily trowel the mud required to seal the oven door.

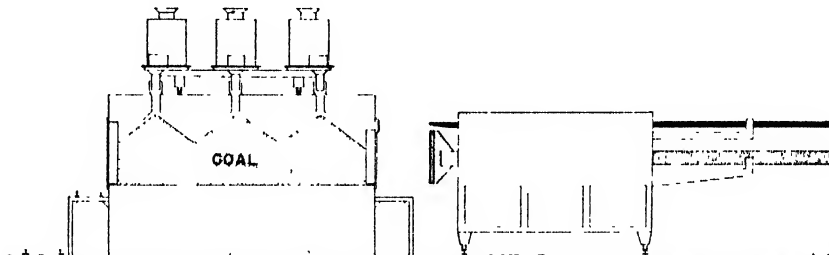
With self-sealing doors a motorized buggy is sometimes provided to facilitate manual cleaning of doors and door jambs. Recently-developed buggies are equipped with elevators and heat shields.

The pusher side of the battery is equipped with a bench serving as a walkway along the entire length of the battery and as a working platform for men serving the pusher-side doors and jambs. Until recent years, the level of this bench was a few inches below the level of the oven floor and could not be protected by hand rails

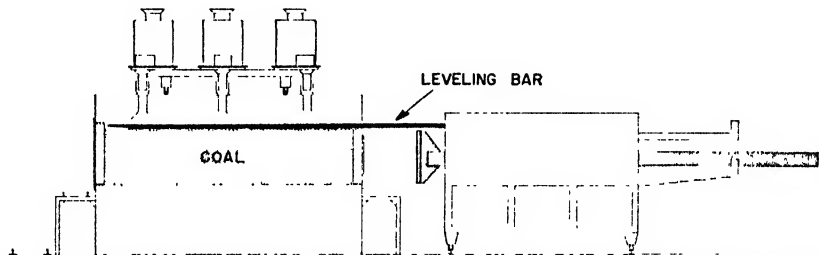
CHARGING, LEVELING AND PUSHING OPERATIONS IN ONE COKING CYCLE OF A BY-PRODUCT COKE OVEN



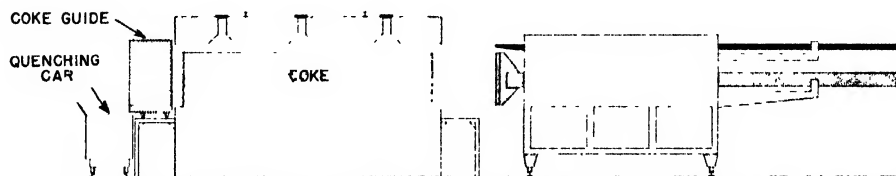
- A. THE CHARGING LARRY, WITH HOPPERS CONTAINING MEASURED AMOUNTS OF COAL, IS IN POSITION OVER CHARGING HOLES FROM WHICH COVERS HAVE BEEN REMOVED. THE PUSHER HAS BEEN MOVED INTO POSITION.



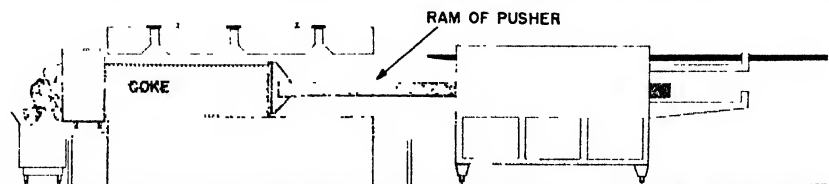
- B. THE COAL FROM THE LARRY HOPPERS HAS DROPPED INTO THE OVEN CHAMBER, FORMING PEAKED PILES.



- C. THE LEVELING DOOR AT THE TOP OF THE OVEN DOOR ON THE PUSHER SIDE HAS BEEN OPENED, AND THE LEVELING BAR ON THE PUSHER HAS BEEN MOVED BACK AND FORTH ACROSS THE PEAKED COAL PILES TO LEVEL THEM. THE BAR NEXT IS WITHDRAWN FROM THE OVEN. THE LEVELING DOOR AND CHARGING HOLES ARE CLOSED, AND THE COKING OPERATION BEGINS.



- D. COKING OF THE COAL ORIGINALLY CHARGED INTO THE OVEN HAS BEEN COMPLETED (IN ABOUT 18 HOURS) AND THE OVEN IS READY TO BE "PUSHED." THE OVEN DOORS ARE REMOVED FROM EACH END, AND THE PUSHER, COKE GUIDE AND QUENCHING CAR ARE MOVED INTO POSITION.



- E. THE RAM OF THE PUSHER ADVANCES TO PUSH THE INCANDESCENT COKE OUT OF THE OVEN, THROUGH THE COKE GUIDE AND INTO THE QUENCHING CAR.

FIG. 4-22. Schematic representation of the sequence of operations involved in charging, leveling and pushing in one coking cycle of a by-product coke oven.

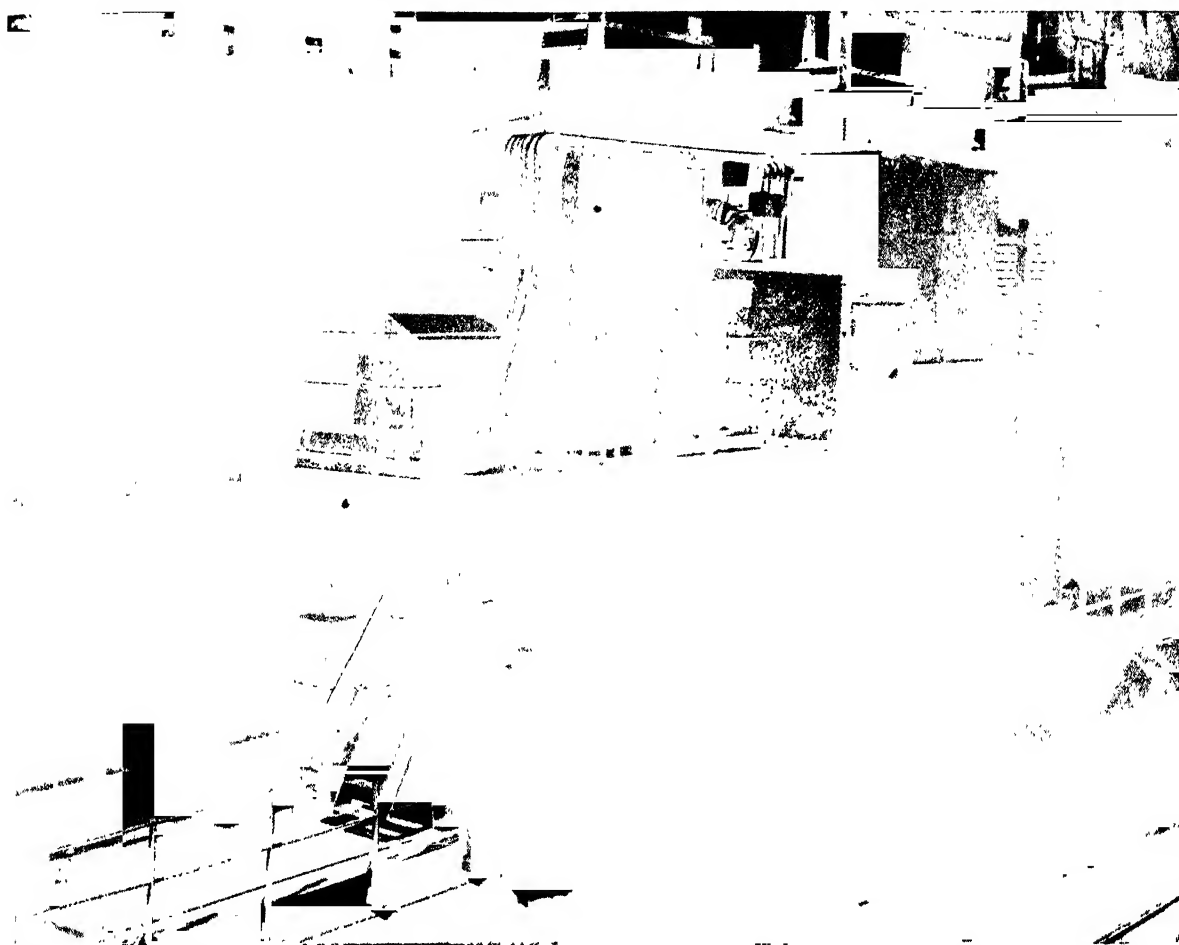


FIG. 4—23. Door-extracting machine and coke guide operating on tracks on the coke-side bench of a coke-oven battery.

in any practical manner on account of clearance for the pusher ram. It is now general practice to lower the bench about $2\frac{1}{2}$ to 3 feet below the oven-floor level so that the pusher ram may pass over the top of a railing. The benches are similarly lowered on the coke side.

The main operating units referred to above, and their functions, are illustrated schematically in Figure 4—22.

Coke-Side Equipment—The coke side of the battery is equipped with a door-extracting machine, coke guide, and, when using luted doors, a luting buggy. The coke-side equipment operates on a track integral with the coke-side bench, as shown in Figures 4—22 to 4—24, inclusive.

The function of the door machine is to remove and hold the coke-side door during the pushing of an oven and to place an attached coke guide in the proper position to conduct the coke across the bench into a quenching car operating on a ground level track which is parallel with the battery.

The design and operation of the door extractor is generally similar to the door extractor of the pusher machine. Recent developments place the extracted door behind a heat shield and have details of design that facilitate easy door cleaning.

The coke guide is attached to the door-extractor machine by a disconnecting coupler. It is conducted along the coke-side bench. At modern plants, the coke guide is equipped with a movable lattice framework that is

power-operated from the door machine. When spotted at the oven to be pushed, this movable framework is moved into the space between the buckstays and against the door jamb, thus preventing coke spillage at this point during pushing. As this part of the guide is subject to the greatest wear, it is designed for ease of replacement and is constructed of metal that will resist heat and wear.

On plants using luted doors, the coke side is provided with motorized luting buggies, generally similar in design and operation to the pusher-side luting buggies.

Quenching Station—There are two methods of quenching the hot coke that is pushed from ovens, namely, wet quenching and dry quenching, the latter being used more extensively in European countries than in the United States. All but one or two plants in the United States wet quench, principally for economic and operational reasons. In dry quenching, this sensible heat is used for the production of steam for general plant use; this will not be discussed here, except to state that it is accomplished by dumping the hot coke into a closed system where the recirculation of inert gas conducts the sensible heat from the coke to a low-pressure boiler until an equilibrium is reached within practical limits, and the coke is below the ignition point in air, at which time the coke is discharged for screening and loading.

Wet quenching in most modern plants is accomplished by receiving the charge of hot coke from the ovens in

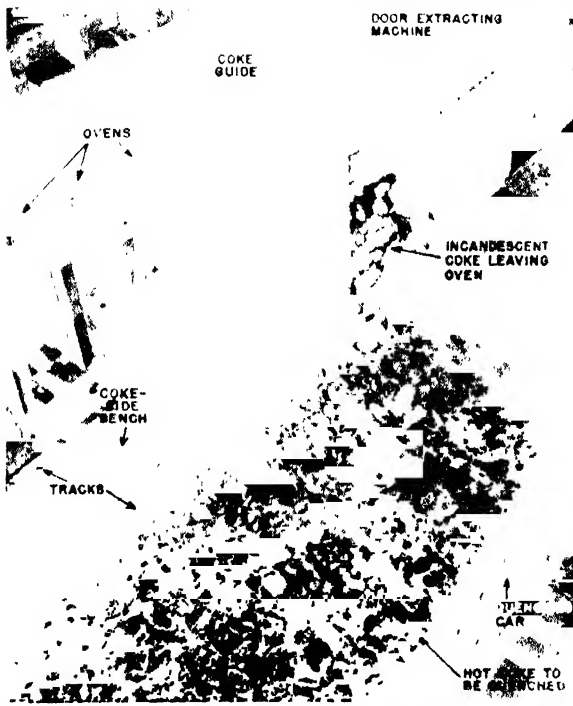


FIG. 4—24. Coke side of a battery of by-product coke ovens during the process of pushing the coke out of one of the ovens. The quenching car is self-propelled, and carries the coke to a quenching station where it is sprayed with water before being dumped on the coke wharf.

the quenching car, which is conducted to a quenching station by an electric locomotive, where it is quenched with water. The car is then taken to a coke wharf where the coke is discharged. The handling of the coke from the wharf will be discussed later.

The quenching car is designed so that by moving the car during the pushing operation the coke is caught in a relatively uniform bed about two feet thick on the sloping bottom of the car (Figure 4—24). The power-operated gates, either electric or air, at the low side of the sloping bottom are so designed and arranged that the quenching water not evaporated may readily drain from the car. When the charge of hot coke has arrived at the quenching station, it is spotted under a system of stationary sprays located in the quenching tower. The operator starts the quench by remote control. The purpose of quenching is obviously to rapidly cool the coke to stop any further combustion. However, it must be recognized that coke of low-moisture content is desired. This is accomplished by so arranging the sprays and the time of quench that sufficient heat will remain in the center of the individual coke lumps to dry excess surface water. The usual practice is to aim at 2½ per cent average moisture in the metallurgical coke after screening.

Most modern plants, of necessity, quench with contaminated water as there seems to be no other practical method of disposal of these waters. It is therefore necessary to design the quenching station in such a way that all such waters can be recirculated until evaporated. This is done by pumps delivering water from a contaminated water sump to the spray lines and returning the unevaporated water to the sump. Any makeup above the plant production of contaminated water is fresh water.

Instrumentation and Control—Most instrumentation around the ovens is confined to the heating facilities. As has been mentioned, the variable controls of individual ovens of a battery are so adjusted that master controls may be installed for each battery of ovens. In

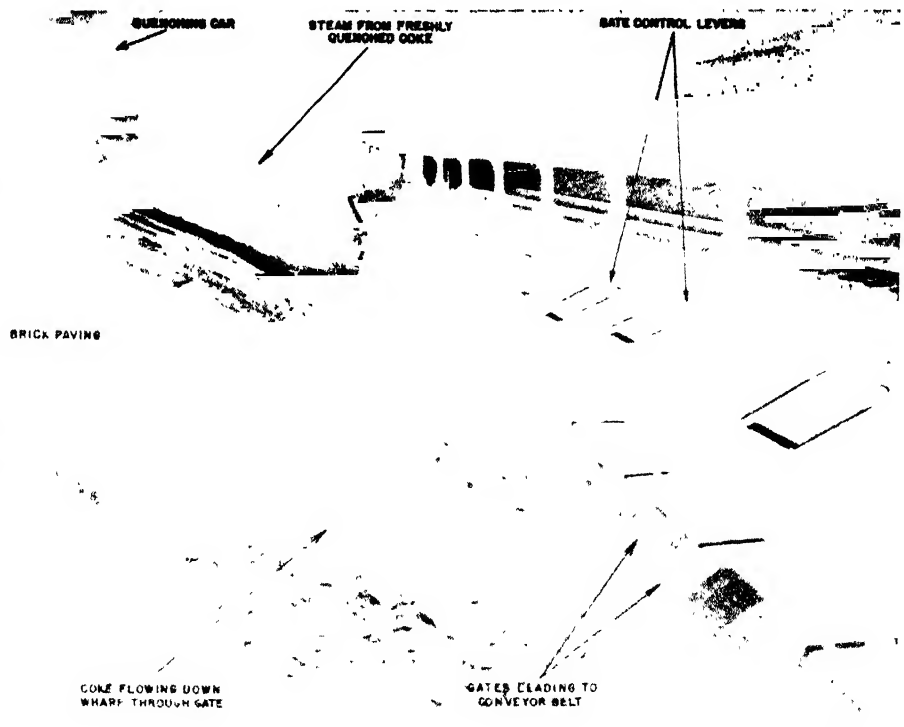


FIG. 4—25. General view of a coke wharf, showing freshly quenched coke being discharged from the gates of the quenching car onto the brick-paved, sloping wharf. When the wharf gates are opened, coke slides down the wharf as at the lower left onto a conveyor belt that carries it to the screening station.

this manner, single machines, meters, gages, and other devices control the various functions of the battery heating system after the individual-oven controls have been adjusted and connected. Thus, the gas required for underfiring is metered in the battery header only. The choice of a flowmeter for this service will be influenced in large part by its accuracy over a wide range of flow. The meter should be recording, preferably one adapted to a uniform graduated chart for reasons of legibility and computation facility. It is also necessary to control accurately the pressure in the fuel-gas main with an instrument capable of close control at relatively low pressures. It may be desired to have a recording of these pressures. A recording thermometer is also installed in the fuel-gas header so that the gas quantities may be computed to standard conditions. The battery is also

equipped with recording draft gages, waste-heat recording thermometers, and various indicating gages. Portable pressure gages, thermometers, and pyrometers are used in the setting of individual-oven controls. Another instrument used by the heaters is a portable pyrometer with which periodic temperature readings are taken of the flues and emerging coke. Probably the most widely accepted pyrometer is of the incandescent-filament type. Recording pressure and temperature gages are also installed in the gas-collecting mains.

Many plants record the regularity of oven operation by some related activity, such as the time of pushing, by the peak-load recording of the pushing-power circuit, or by a recording of the time of quenching of each charge. All utilities have the usual types of instrumentation.

SECTION 6

COKE SCREENING AND HANDLING

Coke Wharf—The coke wharf receives the quenched coke from the quenching car where it is spread out in a thin bed for quick drying and visual inspection for unquenched coke. For obvious reasons such spots must be hand quenched before it is conveyed on rubber belts to the screens. Figure 4—25 shows a view of a modern coke wharf. It is substantially a long, narrow inclined platform with the shorter dimension sloping away from the quenching-car track towards a belt which runs along the lower side of the structure. A properly designed wharf should be of such size that it will serve as a surge storage for quenched coke ahead of the screens so that short delays incidental to screening and loading operations will not interrupt the desired regularity in oven-pushing sequence. Most modern wharfs are paved with hard-burned clay brick, although some plants use cast-iron plates and even refractory concrete for the purpose.

The coke is retained on the wharf by a series of hand-operated gates which, when opened, permit the coke to slide down the wharf and onto the belt conveyor to be delivered to the screening station.

Conveyor System for Coke—The transfer of coke from the wharf to the screening station is accomplished now almost universally by a system of rubber-belt conveyors and chutes.

Screening and Crushing—The purpose of coke screening and crushing is to provide a controlled size of coke for blast-furnace use from which fines and, in some cases, pieces over a set maximum size, are removed. The latter are crushed and screened before use. The very small sizes, commonly called coke breeze are usually used in the coke plants as a boiler fuel or screened for domestic trade.

Storage and Shipping—In the loading of furnace coke and the subsequent transportation to the blast furnaces, care must be taken to prevent additional breakage. For this reason, the coke is loaded into railroad cars and, where practical, is consigned directly to the blast furnaces, as additional handling into and out of stock results in inevitable coke degradation. Where the blast furnaces are located close to the coke plant, belt conveyors are often used to conduct the coke from the screening station to the blast-furnace bins.

SECTION 7

LIMITATIONS AND FUTURE OF BY-PRODUCT COKING

Perhaps the most important over-all consideration that should be kept continually in mind in designing by-product coke ovens and the various auxiliary equipment is that the great majority of design features incorporated in the battery construction cannot be altered during the 20 to 30 years of battery life. It is, therefore, imperative that design as well as construction be of the highest quality. While theory must naturally play an important part in this design, it is well not to overlook the fact that, since the coking characteristics of the coals to be used have such a vital effect on oven design, empirical data developed from actual experience with similar coals is invaluable.

While it is true that the great majority of design work required for by-product coke-oven construction is necessarily carried on by the builders, the engineering work

required of the operating company and plant organizations is of sizable amount and of tremendous importance. In the first place, the builders of coke ovens have always been extremely receptive to suggested improvements, and many have been incorporated in modern oven construction. Secondly, there is considerable latitude in the selection of type and size of ovens, in auxiliary equipment, and particularly in facility layout, all of which require intelligent action on the part of the operating company. And finally, the incentive to produce products of higher quality at reduced costs, coupled with the experience gained through actual operation, repair, and maintenance of the by-product coke-oven facilities, places the organizations of the operating companies in an increasingly important position in furthering the advancement of coke-plant design.

Chapter 5

RECOVERY COAL OF CHEMICALS

SECTION 1 INTRODUCTORY

Chemical Nature of Coal—Coal is the remains of vegetable matter which has been partially decomposed in the presence of moisture and the absence of air and subjected to variations in temperature and pressure by geologic action (see Chapter 3). It is a complex mixture of organic compounds, the principal elements of which are carbon and hydrogen with smaller amounts of oxygen, nitrogen, and sulphur. The ash consists primarily of inorganic compounds which became imbedded in the coal matrix during the coalification process.

The molecules making up coal are basically composed of groups of hexagonal carbon rings with attached hydrogen atoms. In addition, some five-membered carbon rings and carbon chains are attached to the six-membered carbon rings. The chemical structure may be pictured as an aggregate of "mosaics" consisting of cyclic groupings of carbon and hydrogen with six-membered rings and side chain linkages; multi-molecular structures in which component rings are joined together. These compounds like most of those in animal and vegetable life are unstable when subjected to a high degree of heat or thermal treatment.

Since coal may be considered as a hydrocarbon, it reacts in the same manner as other organic compounds of this type. When heated to high temperatures, in the absence of air, the molecule breaks down to yield gases, organic compounds of lower molecular weight and a relatively non-volatile carbonaceous residue (coke).

Products of Coal Carbonization—Although the oxides of iron may be reduced to metallic iron by many agents, carbon (directly or indirectly) is the reducing agent found to be best suited for the economical production of iron. Carbon of suitable reactivity and physical strength was at one time produced from wood by distillation, yielding wood charcoal; but for operation of a modern large blast-furnace the carbon required for the smelting of iron is obtained from the destructive distillation of selected coking coals at 1650° F to 2000° F. Such distillation of coal (carbonization) in the absence of air results in the evolution of gases and vapors, leaving a coke residue in the oven.

The reactions occurring during the carbonization of coal for the production of metallurgical coke are complex. The process can be considered as taking place in three steps: (a) Primary breakdown of coal at temperatures below 1296° F (700° C), yields decomposition products some of which are water, oxides of carbon, hydrogen sulphide, hydroaromatic compounds, paraffins, olefins, phenolic, and nitrogen-containing compounds. (b) Secondary thermal reactions among these liberated primary products as they pass through hot coke, along hot oven walls and through highly-heated free space in the oven involve both synthesis and degradation. A large evolution of hydrogen and the formation of aro-

matic hydrocarbons and methane occur in the stage above 1296° F (700° C). Decomposition of the complex nitrogen-containing compounds produces ammonia, hydrogen cyanide, pyridine bases and nitrogen. (c) Progressive removal of hydrogen from the residue in the oven produces hard coke.

During carbonization, about twenty to thirty per cent by weight of the initial charge of coal is evolved as mixed gases and vapors which pass from the ovens into the collecting mains and are processed through the coal-chemical recovery section of the coke plant to produce coal chemicals. When the production of coke is accomplished in modern by-product coke ovens with equipment for recovering the coal chemicals, one net ton of coking coal in typical American practice yields about the following proportions of the several coal chemicals and coke, depending upon the type of coal carbonized, carbonization temperature and method of coal-chemical recovery:

Blast-Furnace Coke	1,200-1,400 lb.
Coke Breeze	100-200 lb.
Coke-Oven Gas	9,500-11,500 cu. ft.
Tar	8-12 gal.
Ammonium Sulphate	20-28 lb.
Ammonia Liquor	15-35 gal.
Light Oil	2.5-4 gal.

The coke-oven gas contains the fixed gases so classified because they are gases at 760 mm. pressure and 60° F (15.5° C). They are hydrogen, H₂; methane, CH₄; ethane, C₂H₆; carbon monoxide, CO; carbon dioxide, CO₂; illuminants which are essentially unsaturated hydrocarbons, such as ethylene, C₂H₄; propylene, C₃H₆; butylene, C₄H₈; and acetylene, C₂H₂. Other compounds present are hydrogen sulphide, H₂S; ammonia, NH₃; oxygen, O₂; and nitrogen, N₂.

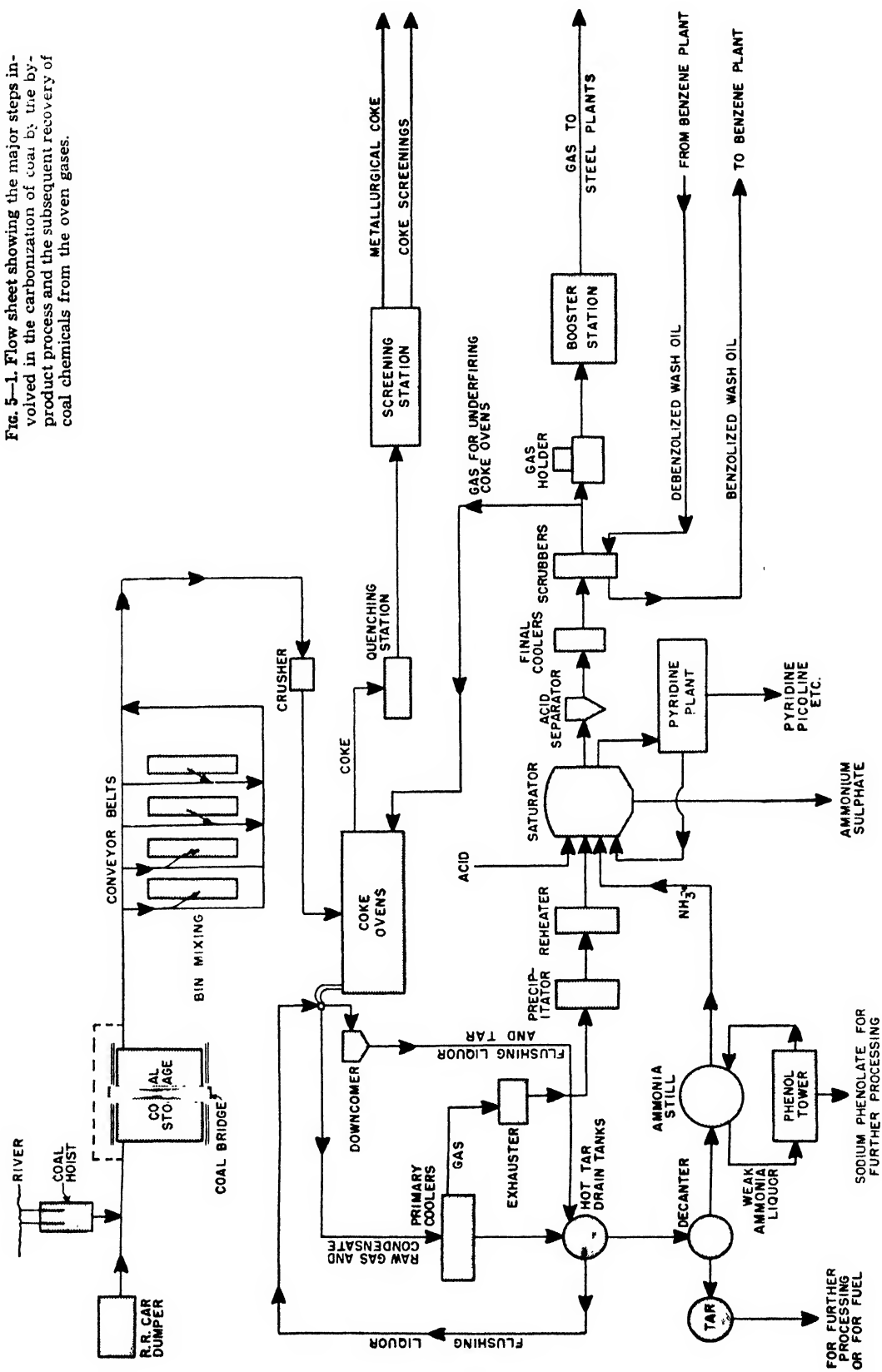
Other substances in the raw gases and vapors leaving the ovens, which are liquids at ordinary temperatures, are:

(a) **Ammonia Liquor** (primarily the water condensing from the gas), which is an aqueous solution of ammonium salts of which there are two kinds—free and fixed. The free salts are those which are decomposed on boiling to liberate ammonia. The fixed salts are those which require boiling with an alkali such as lime to liberate the ammonia.

(b) **Tar**, which is the organic matter separating by condensation from the gas in the collector mains. It is a black, viscous liquid, a little heavier than water. From it the following general classes of compounds may be recovered: pyridine, tar acids, naphthalene, creosote oil and pitch.

(c) **Light Oil**, which is a clear yellow-brown oil with a specific gravity of about 0.880. It contains varying

FIG. 5-1. Flow sheet showing the major steps involved in the carbonization of coal by the by-product process and the subsequent recovery of coal chemicals from the oven gases.



amounts of coal-gas products with boiling points from about 40° C to 200° C and from which benzene, toluene, xylene and solvent naphthas are recovered.

In the recovery of coal chemicals (Figure 5—1), the first step is the recovery of the basic crude materials

(coke-oven gas, ammonia liquor, tar and light oil) as a primary operation in accordance with commercial practice. Secondary operations consist of the processing of these primary products to separate them into their components.

SECTION 2

COLLECTION OF VOLATILE PRODUCTS FROM OVENS

Collecting Main and Suction Main—In the collection system for the recovery of the volatile products from coal, the first operation reduces the temperature of these products generally referred to as foul gas. This takes place in a system of gas mains through which the foul gas passes.

The foul gas passes out of the oven chamber through a refractory-lined ascension pipe and into a gooseneck which connects into the collecting main through a damper valve. The collecting main serves an entire battery of ovens, running parallel with the battery and extending above it on one side or on both sides depending on whether it is a single or double collecting-main operation.

The gas and vapors ordinarily leave the oven at temperatures in the range 1100–1300° F (600–700° C), and are shock-cooled by spraying with flushing liquor in the goosenecks and further cooled by spraying again with flushing liquor at different points along the collecting main. The temperature of the gas and vapors at the point of exit from the collecting main is 295 to 420° F (145 to 215° C). The cooling is effected by the evaporation of a portion of the water from the flushing liquor which removes some of the sensible heat from the gas and condenses some of the vapors with the resultant condensation of the heavy tar.

The gas and remaining vapors pass from the collecting main through one or more cross-over mains into the suction main. A pressure-regulating valve, automatically controlled, is located in each cross-over main. After the gas and vapors have passed this control valve their temperature drops to 175 to 212° F (80 to 100° C), as a result of atmospheric cooling and further evaporation of the flushing liquor.

The flushing liquor, used for cooling in the spray system, is liquor which has been condensed in the mains, collected and recirculated, amounting to 800 to 1,000 gallons per ton of coal carbonized. The flushing liquor, which cools and condenses various vapors in the gas, provides a carrying medium for the condensable tars and other compounds formed in the operations. These liquid materials flow from the collecting main through a seal into a downcomer and are delivered through the return flushing liquor lines to a collecting unit customarily called a hot tar drain tank (Figures 5—1 and 5—2), described in Section 3.

The uniform flow of gas and vapors into the system is accomplished by the charging of coal into the ovens at regularly prescribed intervals and the withdrawal of the evolved gases at a constant rate. This constant rate of removal of the gas is controlled by the automatic pressure regulator in the cross-over main. This pressure regulator provides a slight pressure of about one millimeter of water at the base of the oven prior to pushing. This control of pressure is for the purpose of eliminating the infiltration of atmospheric air or gases from the heating system into the oven, which would have a deleterious effect on the quality and quantity of the coke and coal chemicals.

As a consequence of this practice the pressure on the collecting main is about eight to twelve millimeters of water with a variable suction of about two to three hundred millimeters of water in the cross-over main after the regulating valve.

These pressure differentials are maintained by the use of either low speed positive turbo or centrifugal type exhausters designed to remove the gases and vapors at a controlled rate.

SECTION 3

RECOVERY OF CRUDE COAL TAR

Hot Tar Drain Tank—The hot tar drain tank (Figure 5—2) serves a two-fold purpose in the processing of the liquid condensates and recirculating liquor in the primary liquid system:

(a) To provide a settling basin in which the velocity of the tar and liquor is reduced to permit separation of the tar and liquor by the difference in specific gravity.

(b) To serve as the first settling point for carbonaceous and other finely divided material that is carried along with the tar and liquor from the collecting main.

There are several designs of hot tar drain tanks, but in general they consist of a rectangular tank divided into several compartments. In the first compartment the tar and liquor overflows a baffle into the main compartment. At this point an appreciable quantity of the carbonaceous sludges is collected as a residue. The tar and liquor pass into the main settling compartment where they separate into two layers. The tar settles to the bottom, due to its higher specific gravity, and flows

through a decanter pipe due to a hydrostatic head that is controlled by the level of the total liquid in the tank. The rate of tar removal is regulated by an adjustable weir located in the top of the decanter. The tar flows by gravity to a tar compartment of the hot drain tank, from which it is pumped periodically to tar-storage tanks.

Normally the tar recovered from the hot tar drain tank contains 2 to 5 per cent water. When the water content of the tar is in excess of 5 per cent further decantation or centrifugalization may be required to separate the water from the tar. If further separation between the tar and liquor is required and can be obtained by gravity separation, a tar separator tank or decanter is placed in the process lineup prior to the tar and liquor storage tanks. The mixture of tar and liquor is pumped from the tar compartment of the hot drain tank to approximately the middle section of the separating tank. Here the same process takes place as described for the separation of tar and liquor in the

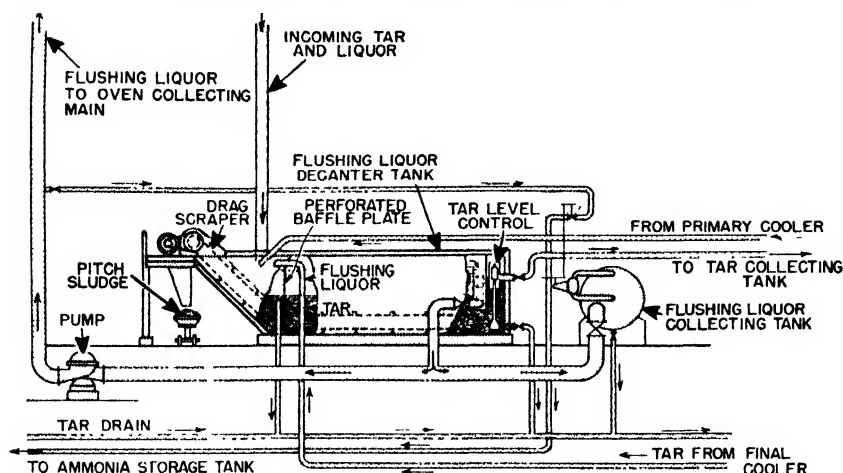


FIG. 5-2. Schematic representation of a flushing-liquor decanter tank or hot tar drain tank.

settling compartment of the hot drain tank. The tar flows from the bottom of the tank through a decanter pipe by hydrostatic head (controlled by the level in the tank) to a tar-storage tank, while the liquor overflows a fixed weir plate into a liquor-storage tank. The overflow liquor from the weir plate provides flushing liquor for the sprays in the goosenecks (damper sprays) and in the collecting mains on the ovens. All of the liquor in excess of the flushing requirements is pumped to storage tanks and further processed for the recovery of ammonia and phenol. It is necessary to take a hot drain tank out of service periodically for cleaning, as the accumulation of finely divided carbonaceous sludges reduces the capacity of these facilities for separation of tar and liquor.

The more recent developments in the design and construction of hot tar drain tanks have incorporated a rotating drag scraper for the continuous removal of the accumulation of solids from the settling compartment. This eliminates a protracted out-of-service period for cleaning this equipment.

Primary Cooler—The non-condensed gas and vapors leaving the collecting and suction mains at a temperature of 167-176° F (75-80° C) require further cooling to 95° F (35° C) to remove additional tar and a major portion of the water vapor and to reduce both the volume and temperature of the gas before its admission to the exhausters. This cooling may be conducted in either direct or indirect primary coolers.

The **direct primary cooler** (Figure 5-3) consists of a tall, cylindrical scrubbing tower fitted with hurdles or baffles usually constructed of wood. The top portion is equipped with a series of spray nozzles and the lower portion contains a chamber to collect the liquor and condensate which flows by gravity into the hot drain tank.

The gas enters the bottom of the tower and the cooling liquor is pumped into the top of the tower through the spray system to provide a downward flow of cooling liquor in counter-current flow to the gas stream. This direct contact between the gas and liquor provides for exchange of heat which is transferred from the hot gas to the cold liquor. This heat is removed from the liquor by indirect heat exchange, through tubular heat exchangers, with circulating water. As a result of this cooling, 20 to 25 per cent of the total tar recovered is condensed along with a considerable quantity of weak liquor containing ammonia. These condensates are

processed through the hot tar drain tanks either separately or in conjunction with the tar and liquor condensates from the collecting main.

The **indirect primary cooler** used for cooling the gas is a heat exchanger, in which water is used for cooling by heat exchange through steel tubes. The water flows through the tubes and the gases pass around the tubes.

Tar Extractor—The gas leaving the primary coolers still contains small amounts of tar that would cause difficulty in the operation of subsequent units in the recovery system. There are two methods available for removal of this entrained tar: (a) mechanical impingement and (b) electrostatic precipitation. In the case of operation by mechanical impingement, it is necessary to place the equipment after the exhausters, but the electrostatic precipitator may be placed before or after the exhausters. The preferred location is after com-

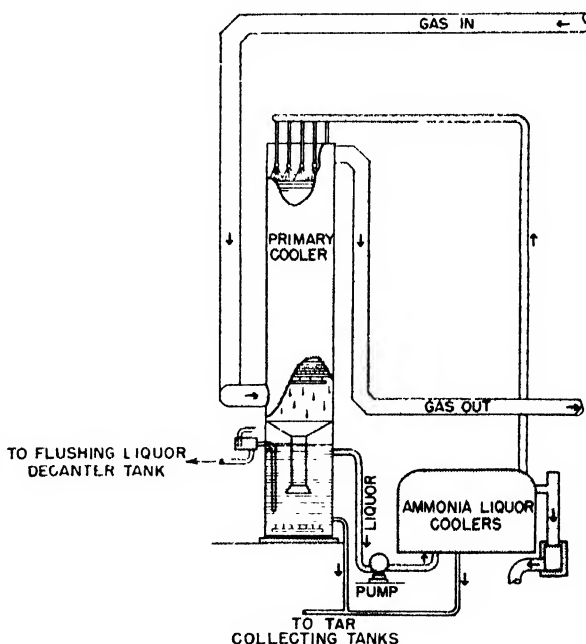


FIG. 5-3. Schematic diagram of a direct primary cooler. (Courtesy, Koppers Company, Inc.)

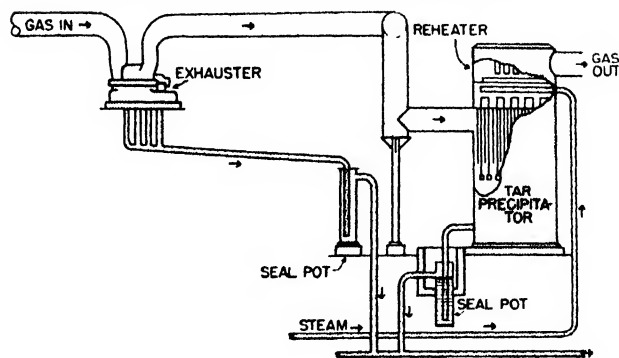


Fig. 5-4. Diagrammatic representation of an electrostatic precipitator for precipitation of tar from raw coke-oven gases. (Courtesy, Koppers Company, Inc.)

pression to avoid any infiltration of air into the unit. The electrostatic type represents the modern concept of removing tar from gas.

The mechanical-impingement type of tar extractor was developed by Pelouze and Audouin, and is commonly referred to as the P & A tar extractor. The principle of operation is based on the fact that when gas containing suspended solids is impinged (at high velocity) on a solid surface, deposition of the suspended particles takes place. This principle is made effective by forcing the gas through restricted spaces directly against a plate to develop a dispersion of the mass and effect a deposition of tar particles, which coalesce to form a liquid and flow to the bottom of the separator. The fixed gas then continues on through this unit for subsequent processing operations. The gas enters the tar extractor through a pipe connected near the bottom and passes through seven standpipes over each of which a series of three concentric perforated shells is suspended. Each assembly of three shells is called a bell. The ascending gas enters the inside of each bell and passes through the perforations, impinging on the solid

face of each succeeding shell to deposit the tar, and finally passes out of the top of the extractor.

The operation of this type of extractor requires the maintenance of a constant differential pressure of eight to ten inches of water, regulated by the liquor seal.

The electrostatic precipitator has been found to be more efficient and is gradually replacing the mechanical impingement type of tar extractor. In the electrostatic precipitator (Figure 5-4) removal of tar fog from gas is achieved by passing the gas through two electrodes having a high electrical potential. The discharge electrode is of small cross section, such as a wire or a series of points, in order to develop the high-intensity electrical field at its surface which is required for ionization of the gas. The collecting electrode has a large cross section and serves as a collector for the suspended particles which are ionized and transferred to this electrode. In this operation, the electrostatic precipitator, in addition to its function as a collecting unit for dispersoids, also serves to catalyze the formation of vapor-phase gums formed by oxides of nitrogen and unsaturated hydrocarbons.

SECTION 4

RECOVERY OF AMMONIA AS AMMONIUM SULPHATE

Ammonium sulphate is formed by the reaction between ammonia and sulphuric acid. The dry chemically pure salt is white in color and contains 25.78 per cent of ammonia. The commercial salt varies in color from white to grayish tan and contains from 25.0 to 25.7 per cent of ammonia.

The ammonia formed during coking exists in both the water and gas that form part of the volatile products. The recovery of this ammonia can be accomplished by three different methods: (a) the direct process in which the total vapor, after separation of condensed tar, is passed through a saturator containing a solution of sulphuric acid to absorb the ammonia; (b) the indirect process in which the ammonia is removed from the gas by scrubbing with water and then removed from the water by distillation and treatment with an alkali, after which the ammonia and steam are passed through the saturator and (c) the semi-direct process in which the ammonia in the liquor produced during carbonization is removed by distillation and alkali treatment and added to the gas stream, the gas containing all of the ammonia being then passed through an absorber con-

taining dilute sulphuric acid for the extraction of ammonia.

Semi-Direct Process—Of these three processes, the semi-direct is most extensively used at the present time and will, therefore, be discussed in detail. The ammonia present in the weak liquor is in two forms classified as "free" and "fixed". The free ammonia is that which is readily dissociated by heat, such as the ammonium carbonates, sulphide, cyanide, etc., while the fixed ammonia is that which requires the presence of a strong alkali to effect displacement of the ammonia from the compound in which it is present, such as ammonium chloride, thiocyanate, ferrocyanide, sulphate, etc. The operation to recover this ammonia is carried out in an ammonia still.

Ammonia Still—In the processing of the liquor, a constant-head tank supplies a uniform flow of liquor, by gravity, to the top of the "free leg" of the ammonia still (Figure 5-5) and this liquor passes down the column over a series of plates equipped with bubble caps and overflow pipes. This liquor is heated by an upward flow of steam which vaporizes the ammonia

sprayed again with a more concentrated solution of sulphuric acid. The gas leaving the pyridine scrubbers is passed through the acid separator where entrained liquids are removed, after which the gas enters the final cooler.

As the dilute sulphuric acid sprays the gas rising through the ammonia absorber, the ammonia in the gas combines with the acid to form ammonium sulphate. The resulting solution drains to a crystallizer from which it is recirculated to the absorber. A constant flow of sulphuric acid is added to the ammonia absorber to replace the acid neutralized by the ammonia in the coke-oven gas. After the solution becomes super-saturated, crystals of ammonium sulphate are precipitated in the crystallizer and accumulate as a slurry in the bottom. A portion of this slurry is removed from the crystallizer and is pumped to the slurry tank where the salt settles, the liquid overflows and returns to the ammonia absorber. The concentrated slurry is withdrawn from the bottom of the slurry tank and is fed in batches to the centrifugal dryers. These dryers currently are arranged to perform the following sequence of operations automatically: (a) rinse the dryer-basket screen with water; (b) feed the slurry into the basket; (c) neutralize the acid remaining in the salt with a dilute solution of aqueous ammonia; (d) rinse the salt with water to remove excess ammonia; (e) centrifuge the water from the salt in the basket; (f) remove the dried salt from the basket; and (g) discharge it onto a conveyor belt. The liquid portion of the slurry is recovered and returned to the ammonia absorbers. The partially-dried ammonium sulphate is conveyed to heated rotary-drum dryers for final drying to a content of approximately 0.1 per cent water.

Following the ammonia removal in the ammonia

absorber, the gas comes in contact with more concentrated sulphuric acid in the pyridine scrubber which removes the tar bases (pyridines, picolines, etc.). Tar bases and sulphuric acid form a weak bond, therefore, the bases are the last to be removed from the gas and the first to be released at low acid concentrations. The tar-base sulphuric acid solution is continuously recirculated and systematically portions are removed for recovery of the tar bases by methods described in Section 6. The pyridine-free liquor is returned to the ammonia absorbers. The gas entrains some of the acid solution in the pyridine scrubber, which is recovered by centrifugal force as it passes through the acid separator.

Another process for the recovery of ammonia is the Wilputte low-differential controlled-crystallization process for producing ammonium sulphate, the equipment for which is constructed of stainless steel throughout (Figure 5-7). In this process the gas is passed through a spray-type absorber over which is circulated a 6 per cent solution of sulphuric acid nearly saturated with ammonium sulphate. The acid entrainment arrestor is an integral part of the absorber. Leaving the absorber, the solution is delivered to the solution-circulating system of a crystallizer in which crystallization takes place by the combined cooling and concentration effects of vacuum evaporation. By the variation of circulating rate and the degree of concentration, the size range of the product can be controlled closely within narrow limits. As the crystals grow in size, they settle or gravitate to the bottom of the suspension tank from which they are delivered to a slurry feed tank and from there to a continuous centrifuge. The accumulation of deposits of hard salt within the equipment is minimized and "killing" for removal of such deposits is very infrequent. The product is dust-free, neutral, free flowing

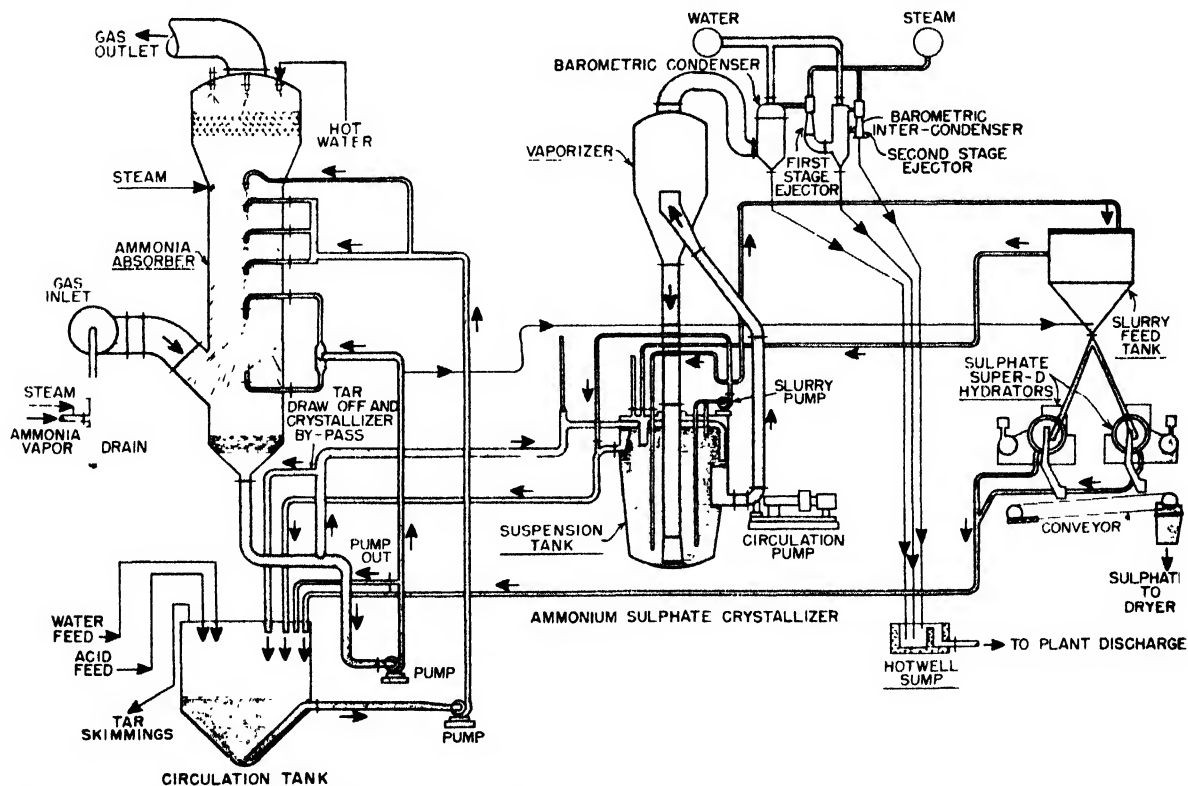
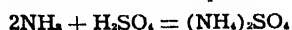


FIG. 5-7. Flow sheet of the Wilputte low-differential controlled-crystallization process for producing ammonium sulphate.

and non-caking and can be made with a size consist favorable to any type of application.

Saturator—At coke plants built prior to about 1930, ammonia-absorbing facilities consist largely of a device often referred to as saturators. These facilities are large dome-shaped, cast iron, lead-lined vessels, sometimes operated in pairs. Gas is admitted to the saturator through a distributor called a "cracker pipe" which, in the older designs, runs completely around the inside circumference of the unit and has a cross section in the shape of an inverted U. The bottom of the pipe is open, but the gas is discharged from it through numerous vertical slots, located on each side of the pipe near the bottom. This arrangement was adopted to provide a large surface for direct contact between the ammonia and dilute acid, which reacts according to the following to form ammonium sulphate:



The salt precipitates and settles to the bottom of the unit when the saturator bath becomes super-saturated. The salt is either siphoned through ejectors to elevated drain tables by means of compressed air, or removed from the base of the saturator by centrifugal pump. When the saturators are equipped with air ejectors, the salt is removed periodically from the elevated drain tables, placed in a pendulum-type centrifugal dryer,

and centrifuged for about 5 minutes, which process removes nearly all the mother liquor. The ammonium sulphate is washed with hot water, to free the crystals of absorbed mother liquor, and whizzed for about 10–12 minutes longer. It is then removed from the centrifugal basket by an unloader or plow and delivered to a belt conveyor, which carries it to a salt pit where any free acid is neutralized by spraying with concentrated ammonia liquid. The mother liquor derived from this operation, as well as the wash water, flows back into the saturators. The acid concentration of the bath is held approximately constant by adding periodically high-strength sulphuric acid.

Continued use of a saturator results in the accumulation of salt within the saturator, which interferes with the flow of gas. It is common practice to prevent such accumulations and to dissolve accumulated salt deposits by "killing" the bath periodically. "Killing" the bath is a term used to describe the method for rectifying this condition, whereby the height and acidity of a bath, prior to "killing", are allowed to decrease and the deficiency then made up by the addition of a definite quantity of acid, water, and mother liquor. The bath then has a free sulphuric acid concentration of 12–14 per cent. These added materials reduce the saturation of ammonium sulphate in the total liquor and all deposits of salt are redissolved.

SECTION 5

RECOVERY OF PHENOL

The water, i.e., weak ammonia liquor, recovered with the volatile products of coal carbonization contains 0.5 to 3.0 grams per liter of phenol (sometimes called carbolic acid) and its homologues. In order to recover this phenol, two processes are available—the vapor-recirculation process and the solvent extraction process. (Phenol is also recovered from coal tar, by processes described in Section 9 of this chapter.)

Vapor-Recirculation Process—This process utilizes the vapor pressure of phenol and operates in conjunction with the ammonia still. The weak liquor first is distilled in the free leg of the ammonia still in order to remove the maximum quantities of the acidic gases, H_2S , CO_2 , and HCN , but the minimum amount of phenol. In order to make this separation, improvement in the rectifying efficiency of the free leg by addition of a section packed with Raschig rings has been necessary.

The quantity of steam to the free leg is also reduced so that the vapors leave the top of the free leg at about 203°F (95°C), instead of at about 220°F (104°C) which otherwise would be their temperature. This

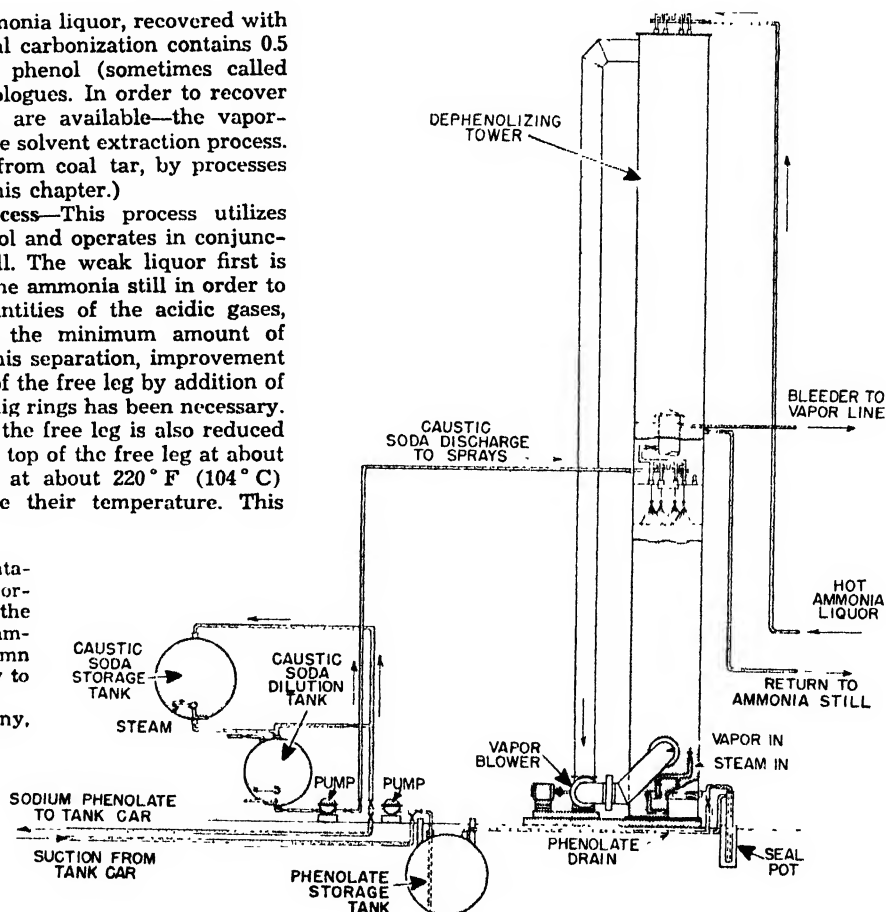


FIG. 5—8. Schematic representation of the steps in the vapor-recirculation process for the recovery of phenol from ammonia liquor. Side of column has been partially cut away to show spray arrangement. (Courtesy, Koppers Company, Inc.)

quantity of steam is sufficient to remove ammonia and the acidic gases satisfactorily without an excessive loss of phenol. This method of operation has resulted in diminishing the phenol loss to an average of 5 per cent compared with 20–30 per cent in the absence of this control.

The ammonia liquor leaving the base of the "free leg" is then transferred to the dephenolizing unit, where the phenols are removed. The dephenolized liquor is returned to the "lime leg".

In the operation of the dephenolizing unit (Figure 5–8), the liquor is pumped into the top of a dephenolizing tower consisting of two main sections. In the upper section it passes downward over wooden hurdles and meets a countercurrent flow of steam which vaporizes the phenols. The liquor from the base of the upper section returns to the ammonia still. The phenol vapors and steam are carried into the bottom of the tower and travel upward through steel turnings where they meet a countercurrent flow of caustic soda which extracts the phenols and forms sodium phenolate.

This operation is conducted at 212° F (100° C). At this temperature, the equilibrium of the phenol-sodium phenolate reaction is such that a suitable balance between the utilization of sodium hydroxide and the loss

of phenol results in the conversion of about 50 per cent of the available sodium hydroxide into sodium phenolate with a loss of about 5 per cent of the phenol.

Solvent Extraction Process—This method is based on the principle that the phenols are more soluble in benzene or light oil than in water and that the phenols can be extracted from benzene or light oil with caustic soda.

The process consists of treating weak liquor containing phenols with benzene or light oil in two or three stages by countercurrent flow. In the first stage of contact, the weak liquor from which the major part of the phenol has been extracted is treated with benzene or light oil from which the phenols have been removed. The solvent is then contacted with liquor that is richer in phenol. This operation may be carried to a third contact zone in order to increase the phenol content of the solvent.

The solvent is removed from the liquor system and treated with caustic soda in two or three steps in order to remove the phenol from the solvent and recover the phenol as sodium phenolate. The sodium phenolate is then boiled to remove the entrained solvent. It is then neutralized with sulphuric acid or carbon dioxide gas to liberate the crude phenol.

SECTION 6

RECOVERY AND REFINING OF PYRIDINE BASES

Among the products of carbonization of coal are the pyridine bases. Part of these bases dissolve in the tar, while the remainder is present in the gas going to the saturators. The pyridine bases include pyridine, picolines (methyl pyridines), lutidines (dimethyl pyridines), etc. These compounds being basic in nature dissolve in the saturator liquor with the formation of pyridine sulphate. Where pyridine recovery operations are practical, the acidity of the saturator liquors must be carefully controlled. Two processes for the recovery

of these bases from saturator liquors are described in the following. Recovery of pyridine bases from tar is described in Section 9 of this chapter, dealing with the refining of coal tar.

Saturator liquor is withdrawn continuously from the feed tank to the continuous dryers at a predetermined rate and conducted to a neutralizing still, where the neutralization of the saturator liquor and separation of the bases from the neutralized liquor are effected simultaneously. Ammonia vapors (containing carbon

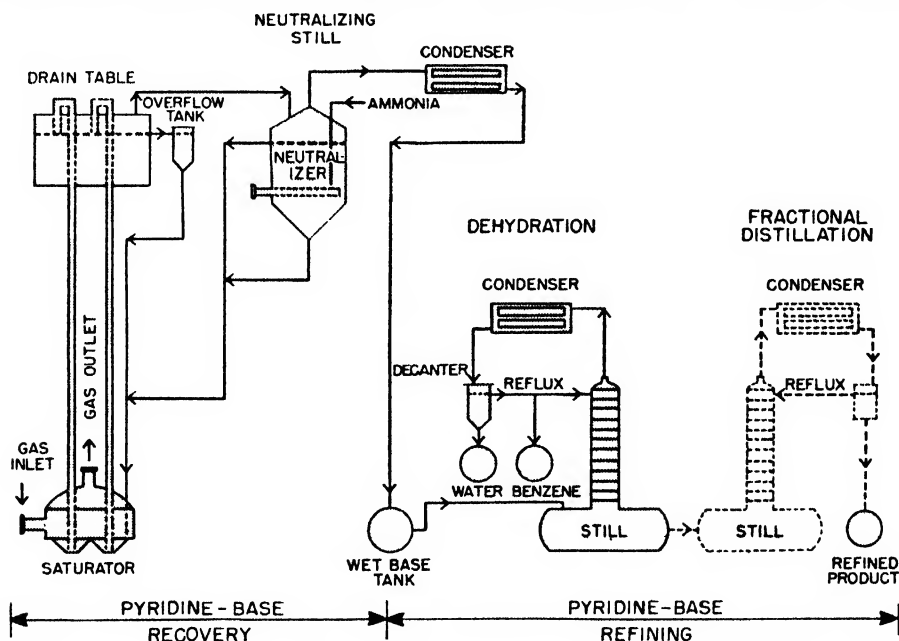


FIG. 5–9. Flow sheet of a batch-type plant for recovery and refining of pyridine bases. The equipment at the far right is shown dotted because the same still is used for both dehydration and fractional distillation, except that, in the latter case, the decanter is not employed.

dioxide) are taken directly from the fixed leg of the ammonia stills, and conducted into the neutralizing still at a constant rate.

The neutralizing still is fitted with a dephlegmator which operates with a top vapor temperature of from 185 to 195° F (85 to 90° C). Pyridine bases, ammonia, carbon dioxide, and water, distill overhead and are conducted to the separating unit, while the ammonium sulphate liquor is withdrawn continuously from the bottom of the still and returned to the saturator.

The vapors from the dephlegmator pass downward through a condenser and the condensate therefrom is conducted below the level of liquid in the separator. The ammonia, carbon dioxide, and water unite to form a solution of ammonium carbonate with a specific gravity of 1.07 to 1.08. The pyridine bases separate from this ammonium carbonate solution as a top liquid layer containing about 15 per cent water and are decanted continuously into a receiving tank. The ammonium carbonate solution is maintained at a constant level by withdrawing through a constant level device and returned to the neutralizing still. This is done in order to maintain an equilibrium of ammonia and carbon dioxide in the system which results in a solution of the desired specific gravity in the separator.

In another method of continuous operation, the pyridine bases are completely condensed at the time of neutralization and separated from the saturator liquor by gravity.

Batch-Type Recovery Process for Pyridine Bases—Pyridine bases may be recovered by the intermittent removal of saturator liquor from the saturator and batch treatment of this liquor for the recovery of the pyridine bases (Figure 5—9).

The recovery equipment consists of a lead-lined tank which serves as a receiving tank and still. Its capacity

is designed to hold the saturator liquors that will be removed from all saturators in 24 hours. It is equipped with a steam coil and an open pipe for the ammonia feed. A condenser is provided to condense the water and pyridine bases distilled over and a receiving tank for the wet product.

In the daily operation for the recovery of the bases 250–300 gallons of saturator liquor are removed from each saturator twice per 24 hours. The liquor is removed from the drain table and delivered to the mother liquor wells. This liquor, containing from 25 to 40 grams per liter of pyridine bases and 80 to 100 grams per liter of free acid, is transferred to the neutralizing still where it is treated with ammonia until the bases are liberated; this operation is called "springing". The liquor in this still is then heated and the bases are distilled off and condensed as a water solution containing 40 per cent of bases.

Refining of Pyridine Bases—The crude pyridine bases are first dehydrated by azeotropic distillation, using benzol as the entrainer.

The wet pyridine bases are charged to a still with a quantity of benzene. The quantity of benzene should be sufficient to form the azeotrope with water during distillation. The distillation is controlled by maintaining the vapor temperature at 157.1° F (69.5° C) until the water has been removed. The benzene and water are separated in a decanter tank where the water is sent to the waste disposal system and the benzene returned to the still.

After the water has been completely removed, the still is adjusted to provide a balance between the forward flow and reflux return to separate the benzene from the pyridine and then continued for the recovery of refined pyridine, picolines, and higher boiling point bases.

SECTION 7

RECOVERY OF COKE-OVEN LIGHT OIL

Light Oil—The gas leaving the saturators contains light oil. This oil is a clear yellow-brown oil with a specific gravity of about 0.880. It is a mixture of all those condensable products of coal gas with boiling points up to 390° F (200° C), containing well over a hundred constituents (Table 5—I). Most of these are present in such low concentrations that their recovery is seldom practicable. Many of the constituents, such as, olefin and diolefin hydrocarbons, some straight chain and cyclic paraffins and small amounts of sulphur, nitrogen and oxygen compounds, are present in small quantities, while those few present in larger quantities constitute the products which are economically recoverable. The principal usable constituents are benzene (60–85 per cent), toluene (6–17 per cent), xylene (1–7 per cent), and solvent naphtha (0.5–3 per cent). Light oil constitutes approximately one per cent of the coal carbonized.

Light-Oil Recovery Process—The removal of light oil from coal gas is generally the last step in the coal chemical recovery process.

There are three general methods used for the recovery of light oil:

(a) Refrigeration and compression involving temperatures below -70° C and pressures of 10 atmospheres.

(b) Adsorption by solid adsorbents involving the removal of light oil from the gas by passing it through a bed of activated carbon and recovering the light oil from the carbon by heating with direct or indirect steam.

(c) Absorption by solvents involving washing the coal gas with a petroleum wash oil or other absorbents followed by steam distillation of the enriched wash oil to recover the light oil.

The latter practice (Figure 5—10) is the one almost universally followed in the United States, due to the availability and low cost of petroleum wash oil, and is the method that will be discussed here. The efficiency of recovery varies widely with the seasons since one of the major considerations is the temperature of the coal gas and wash oil entering the absorbing process. Another consideration is the ratio of wash oil to gas. The absorption equipment should be of reasonable design as to size and contact time. The oil-and-gas ratio varies depending on the equipment design and light-oil content of the gas prior to light-oil removal.

Typical operating conditions are as follows: the temperature of gas entering the absorption process is 59–86° F (15–30° C), the temperature of wash oil entering the process is 60–90° F (17–32° C) and the wash oil circulated per ton of coal carbonized is 150–200 gallons.

The boiling point of the wash oil should be well above 390° F (200° C) so as to permit an effective separation of light oil from wash oil in debenzolization. The oil should not thicken and should have a low viscosity to permit its distribution in the scrubbing towers. It should not deteriorate readily but maintain its initial properties as long as possible to keep makeup oil at a

Table 5—1. Fractions of Coke-Oven Light Oil, and Boiling Points of Some of Their Constituents

CONSTITUENT GROUP	FORERUNNINGS (2% of Light Oil)	CRUDE BENZENE (60% of Light Oil)	CRUDE TOLUENE (18% of Light Oil)
AROMATICS	Traces of Benzene	Benzene	Toluene
PARAFFINS	n-Pentane	n-Hexene 2-Methylhexane n-Heptane	n-Heptane n-Octane
CYCLOPARAFFINS NAPHTHENES	Cyclopentane	Cyclohexane	Methylcyclohexane
UNSATURATEDS (Olefins-Diolefins and Aromatic Hydrocarbons with Unsaturated Side Chains)	Butene-1 Pentene-1 Amylenes n-Hexylene Cyclopentadiene-1, 3 Butadiene-1, 3 Carbon Disulphide Hydrogen Sulphide Hydrogen Cyanide Carbonyl Sulphide Methyl Mercaptan Ethyl Mercaptan Dimethyl Sulphide	Hexene-2 Hexadiene-1, 3 n-Heptylene Cyclohexene Unidentified Compounds	Cycloheptane Unidentified Compounds
SULPHUR COMPOUNDS		Thiophene Diethyl Sulphide	Methylthiophene
NITROGEN & OXYGEN COMPOUNDS			Pyridine
CONSTITUENT GROUP	CRUDE NO. 1 SOLVENT (8% of Light Oil)	CRUDE NO. 2 SOLVENT (6% of Light Oil)	CRUDE RESIDUE (6% of Light Oil)
AROMATICS	o-Xylene m-Xylene p-Xylene Ethyl Benzene	n-Propyl Benzene Ethyl Toluene Mesitylene Pseudocumene Hemimellitene Cymenes Durenes n-Decane	Wash Oil Naphthalene Solvents Pitch Residue
PARAFFINS	n-Octane n-Nonane		
CYCLOPARAFFINS NAPHTHENES	Cyclooctane	Cyclononane	
UNSATURATEDS (Olefins-Diolefins and Aromatic Hydrocarbons with Unsaturated Side Chains)	Octylene Styrene Unidentified Compounds	Coumarone Dicyclopentadiene Indene	
SULPHUR COMPOUNDS	Thioxenes	Trimethylthiophene Thiophenol Tetramethylthiophene	
NITROGEN & OXYGEN COMPOUNDS	Picoline	Cresols Dimethyl Pyridines Phenol	

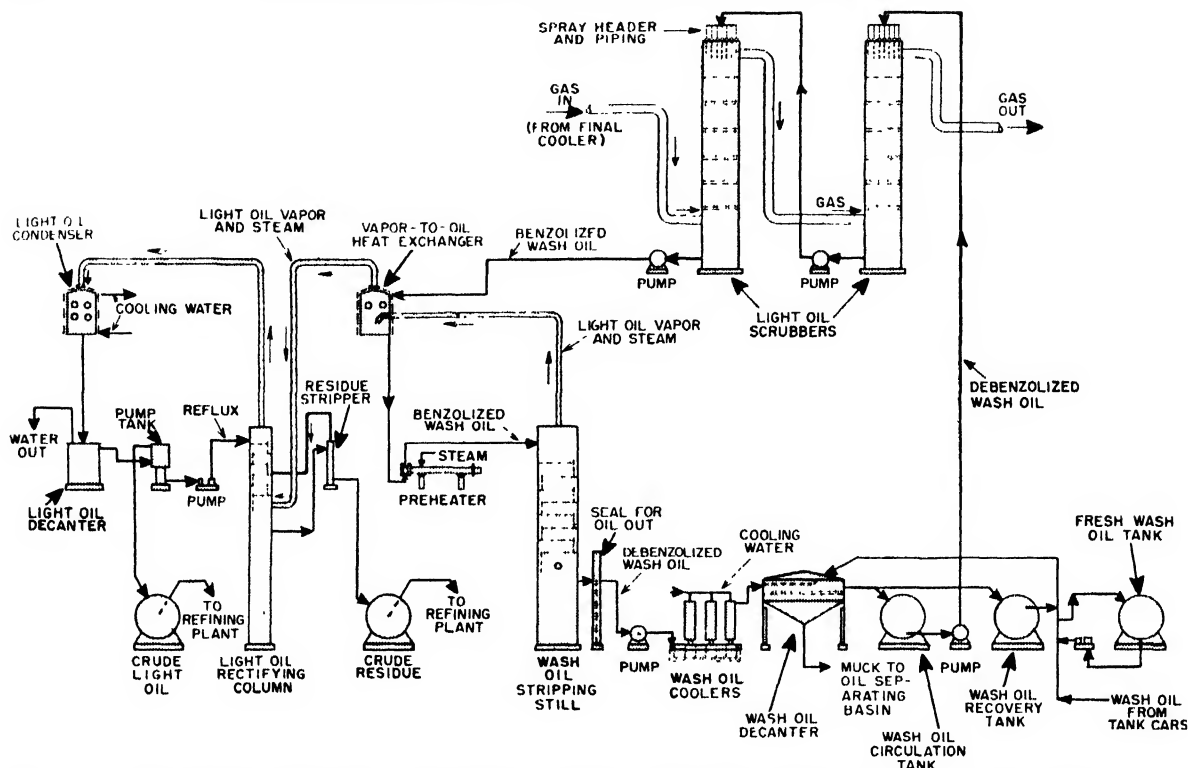


Fig. 5-10. Flow sheet of a light-oil recovery system. (Courtesy, Wilputte Coke Oven Division, Allied Chemical and Dye Corp.)

minimum. It must be especially stable with respect to the repeated heating which takes place in the recycling of the oil in the process. Its absorptive capacity should be very high and it should not react with or contaminate the coal gas. The specific gravity should be low enough to permit effective separation of wash oil and water in the processing and keep emulsification of the two to a minimum. The specific heat should be low because the oil is subjected to repeated heating and cooling as it is recycled in the process.

The petroleum wash oil normally used for this absorption process has a boiling range of 518-622° F (270-350° C). Other specifications which are general for a petroleum oil include a specific gravity of about 0.830, a viscosity of 45 seconds Saybolt at 100° F (38° C), a pour point of 35° F (2° C), an emulsification of 95 per cent separation in 50 seconds, a flash point of 300° F (150° C), fire point of 335° F (168° C), and a low residue under 0.10 per cent when heated for a period of five days at approximately 300° F (150° C).

Final Cooler—The first step in the recovery of light oil by absorption in a liquid medium is that of cooling the gas leaving the saturators at a temperature of 112-140° F (50-60° C) by direct contact with water in a tower scrubber called a final cooler. The facilities are so named since the gas is here given its final cooling in the coal-chemical processing. This is necessary to remove naphthalene from the gas and also cool the gas prior to its admission to the wash-oil scrubbers.

The tower consists of a tall cylindrical shell of steel approximately 10 to 15 feet in diameter and 50 to 75 feet in height filled with a suitable packing material, either metallic or wooden. The gas enters near the bottom of the tower and passes up through the packing material and out near the top. The cooling water enters the top of the tower through a spray system and

the water passes down through the tower, coming in direct contact with the gas in a countercurrent manner. The water leaves the tower at the bottom through a sealed outlet pipe to prevent escape of any gas. The heat from the gas is transferred to the water, which in turn is cooled in an induced-draft water-cooling tower with air or in an atmospheric water-spray cooling operation. Cooling of the water depends upon air circulation and the vaporization of a part of the water in circulation, the latent heat of vaporization of the water being responsible for additional cooling. Operating practice is to cool the water, and in turn cool the gas, to as low a temperature as practicable, depending upon atmospheric temperature, since most effective absorption of light oil is obtained at low temperatures. Cooling is not carried below 60° F (15° C), since below that temperature petroleum absorbing oil becomes too viscous to flow freely.

This direct-cooling operation causes the condensation of a major portion of the naphthalene and any entrained tar and vapor-phase gums. The naphthalene is recovered in a sump operation and is either added to the tar or refined directly to provide a salable product.

In some of the more modern facilities, the lower part of the final-cooler tower is redesigned to permit the outlet water to come in direct contact with tar in order to dissolve the naphthalene as it is being removed from the gas.

Wash-Oil Scrubber—The second step in the recovery of light oil is its absorption in the liquid petroleum wash oil. The gas comes in direct contact with the wash oil in one or more tall scrubbing towers containing packing. The gas passes from the first tower to the last in series and the wash oil travels from the last tower to the first in reverse series. The flow of gas and wash oil is countercurrent in each tower. The steel towers are

approximately 15 to 22 feet in diameter and 100 feet in height.

The wash oil is introduced through a number of sprays in the top of the tower and comes into direct contact with the gas, which flows from the bottom to the top. An oil-storage tank is provided in the base of the tower to receive the oil and maintain a surge capacity for pumping the oil away. The oil passes from the gas compartment to oil storage through a sealed pipe. It is pumped from the base of one tower to the spray system in the top of the next tower in the series. From the last tower the oil is pumped to the stripping stills for separation of light oil from wash oil. Wash oil prior to light-oil absorption is called debenzolized and, after absorption, benzolized. The benzolized wash oil contains 2 to 3 per cent light oil. The debenzolized wash oil is cooled in indirect cast-iron or steel-pipe cooling coils with water to a temperature several degrees higher than that of the gas entering the scrubbers, which is 60 to 75° F (15 to 25° C). This is to prevent condensation of water from the gas, which would form an emulsion with the oil, causing clogging of the free space in the packing of the tower. The rate requirement for the circulation of oil through the scrubbers is a function of the vapor-pressure distribution between the light oil dissolved in the absorbent oil and that remaining in the gas at the temperature of operation. From 90 to 95 per cent of the light-oil content of the gas is recovered in this operation. The wash oil, after being cooled, passes through a large decanting tank which acts as a settling compartment for the emulsified and resinous materials present in the wash oil. This material accumulates in the bottom of the tank, and the wash oil decants off at a higher level to a small receiving tank, from which it is pumped to the top of the first scrubbing tower in the series. Generally, two decanting and two receiving tanks are provided to permit cleaning the residue from the tanks periodically.

Recent designs of wash-oil scrubbers are not fitted with hurdles and packing used in the previous type. Contact between the gas and absorption oil is accomplished by the use of single conical sprays placed at three different elevations within the tower, as shown in Figure 5—11. Absorption oil pumped through the top pressure spray is collected by steel-plate umbrellas, and passed through second and third spray nozzles of the gravity-flow type. Baffles direct the flow of gas toward the spray in every pass and horizontal angles restore correct distribution above each umbrella collector. Restrictions to gas flow by accumulations of residues commonly found in packed scrubbers are minimized or eliminated in scrubbers of this design.

Debenzolization of Wash Oil—In the debenzolization step, the light oil (2 to 3 per cent) in the benzolized wash oil is separated by steam distillation. The carry-over of absorbing oil into the light oil is kept to about 5 per cent and the debenzolized absorbing oil contains 0.2 per cent light oil.

In the straight steam-distillation process at atmospheric pressure, the benzolized wash oil is preheated to approximately 212° F (100° C) with a vapor-to-oil and an oil-to-oil heat exchanger. Heating is continued to 295° F (145° C) with an indirect preheater of the shell-and-tube type with the oil flowing through the tubes using steam as the heating medium on the shell side. The preheated oil is introduced near the top into a multi-plate bubble-cap fractionating column leaving several plates above the feed to keep entrainment of wash oil to a minimum. (Figure 5—13 shows the plate and bubble-cap arrangement in a still of similar design.) The benzolized wash oil flows down the column counter-

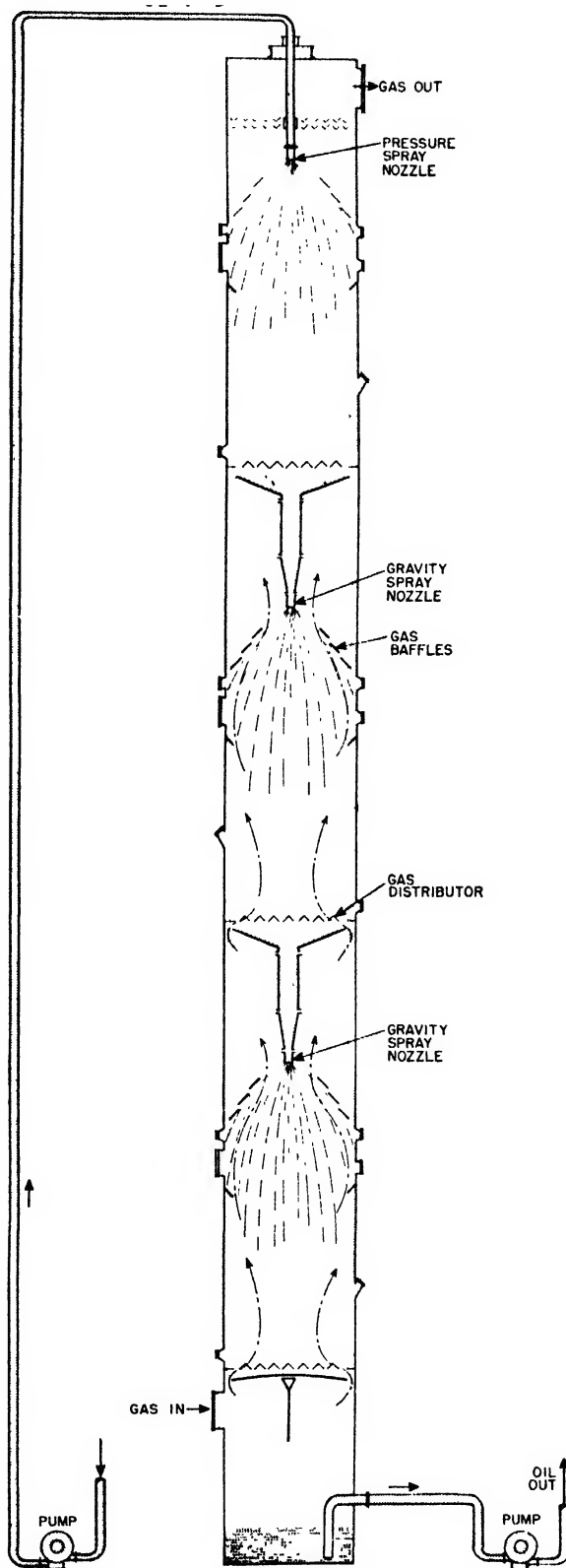


Fig. 5—11. Schematic representation of a spray-type wash-oil scrubber.

current to upward flow of live steam, which is introduced in the base of the still column. The debenzolized wash oil leaving the base of the column through a sealed outlet at a temperature of 290 to 300° F (145 to 150° C) passes through the oil-to-oil heat exchanger in which it is cooled to 212° F (100° C) giving up its heat to the incoming benzolized wash oil. Water separates out at this point and is drained off. The wash oil passes through a pumping tank and is pumped at 212° F (100° C) to cooling coils for cooling prior to being used again as an absorbent for the light oil in the scrubber towers.

The mixture of steam and light oil vapors leaving the top of the column flows through the tubular vapor-to-oil heat exchanger which recovers heat and also acts as a partial condenser. Sufficient heat is imparted to the incoming benzolized oil to raise its temperature 45° F (25° C) and, at the same time, the vapors are cooled to cause a portion of the steam and high-boiling constituents of light oil to condense (the condensate of which carries along some of the wash oil which was carried over the top of the column as entrainment). The mixture of oil and water is separated in a gravity separator tank, the water flowing to the sump system and the oil returned to the debenzolized oil streams. The mixture of steam and light-oil vapors leaving

the vapor-to-oil heat exchanger passes to a water-cooled condenser, which is of a multi-pass design, with the vapor and water flowing countercurrent to each other. The condensate flows to a gravity separator effecting a separation of the light oil and water, the light oil flowing to storage and the water to the sump system.

In some designs, an additional fractionating column is added to the debenzolization process for rectifying the light-oil vapors from the vapor-to-oil heat exchanger. In this case, the condensate of the vapor-to-oil heat exchanger, after separation of the water, is also introduced into the rectifying column. The mixture of steam and light-oil vapors enters the multi-plate bubble-cap rectifying column near the middle section. The light oil is separated into two fractions: the distillate containing forerunnings, benzene, toluene, xylene, and low-boiling solvent; while the residual fraction contains an admixture of high-boiling solvents, naphthalene and wash oil. A portion of the distillate is returned to the top of the column as reflux, the control point being the vapor temperature at the top of the column.

In the more modern debenzolization processes, the benzolized wash oil is processed at a temperature of 195 to 250° F (90 to 120° C) and this eliminates the need for an oil-to-oil heat exchanger in the process lineup.

SECTION 8

REFINING OF COKE-OVEN LIGHT OIL

Light oil is separated into a number of marketable products by refining according to trade specifications. The aromatic hydrocarbons in crude light oil, which are present in large quantities, are the principal source of the commercial products. These hydrocarbons are ben-

zene, toluene, xylene, coumarene and indene. The remaining substances, which are present in small quantities, are generally considered to be impurities. These consist of saturated straight-chain hydrocarbons, paraffins, saturated closed-chain or cycloparaffin and naphtha-

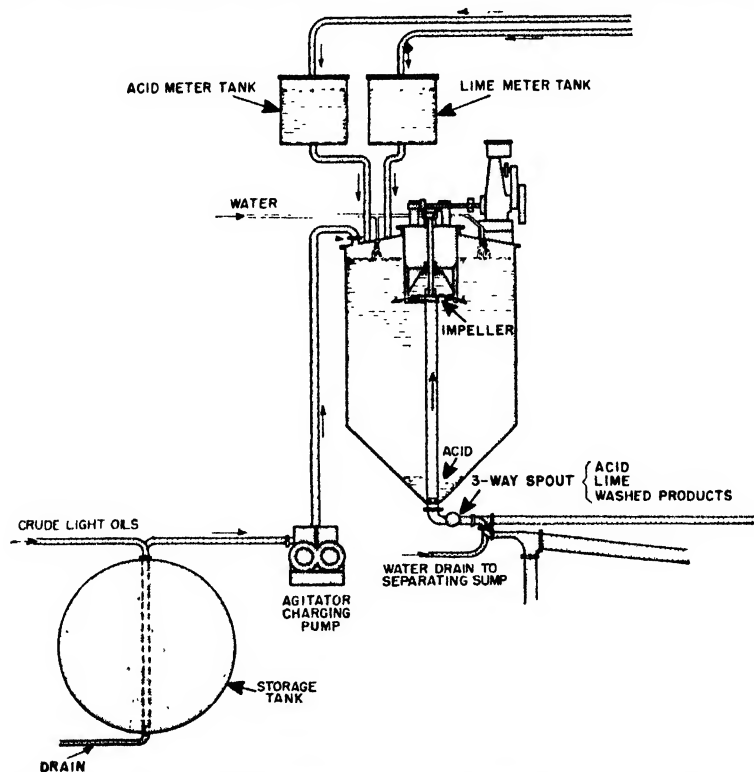


FIG. 5-12. Schematic representation of an acid washer. (Courtesy, Koppers Company, Inc.)

lene hydrocarbons, unsaturated hydrocarbon olefins and diolefins, sulphur compounds (carbon disulphide, thiophene and its homologues), nitrogen compounds (pyridine and its homologues), and oxygen compounds (phenol and its homologues).

Acid Washing—Washing light-oil fractions with sulphuric acid removes impurities by chemical reaction of the acid with the impurities (Figure 5—12). These particular impurities cannot be removed by fractional distillation. The chemical reactions involve oxidation, polymerization, sulphonation and other complex reactions with the unsaturated hydrocarbon olefins and diolefins, sulphur, nitrogen and oxygen compounds. The reaction products remain in solution in the acid and are removed with it by gravity separation from the washed oil. The acid-washed oils are, after neutralization, separated into individual constituents by fractional distillation.

Fractional Distillation—Fractional distillation separates the principal aromatic components of light oil by repeated vaporization of a mixture of these compounds (which have different boiling points) followed by condensation of the vapors in such a way that the desired degree of separation of the components is obtained.

The following proportional distribution of the main constituents in light oil, based on distillation range, indicates the relationship of quantities involved:

Fraction	Distribution	Approx. Boiling Points
Forerunnings	1.5- 2.5%	77-158° F (25-70° C)
Pure Benzene	60.0-85.0%	176.2° F (80.1° C)
Pure Toluene	6.0-17.0%	231.1° F (110.6° C)
Pure Xylenes	1.0- 7.0%	275-293° F (135-145° C)
Heavy Solvents	0.5- 3.0%	338-392° F (170-200° C)
Residues	5.0-11.5%	392° F (Above 200° C)

The particular operating sequence in regard to acid washing and distillation varies, depending on quality of

crude light oil, the equipment available, and the market demands. A complete examination is required in each plant to plan effectively the type of operation which will be most satisfactory.

The evolution of the refining practice for the processing of light oil started with its use as a motor fuel, which has been followed through the years with the gradual consumption of individual products for solvent and chemical purposes. During World War I the important use was the production of trinitrotoluene from toluene. In World War II, in addition to the use of toluene as mentioned, benzene became important for synthetic rubber and aviation fuel, xylene for protective coatings. Advances in importance of the aromatic products for commercial use in recent years has been accompanied by the improvement of refining techniques.

In the earlier distilling operations, only batch stills were used whereas modern refining techniques make use of continuous stills.

Batch Still Operation—A batch still consists of a kettle connected to a fractionating column by means of a vapor pipe to carry vapor to the column, and a liquid-sealed pipe to permit return flow of liquid to the kettle. This column contains thirty to forty plates which are fitted with bubble caps and return flow lines to permit condensate to return to each succeeding lower plate. The actual number of plates, bubble caps, etc., must be determined from design characteristics dictated by the separation to be performed on the material to be distilled. The kettle is equipped with steam coils with sufficient surface for the available steam pressure to supply the column with vapor at a predetermined rate. A vapor pipe receives the vapors from the top of the column and delivers them to a condenser which may also be equipped with an aftercooler. The condensate may then be collected in a receiving tank and a part pumped to the top of the column to provide reflux for

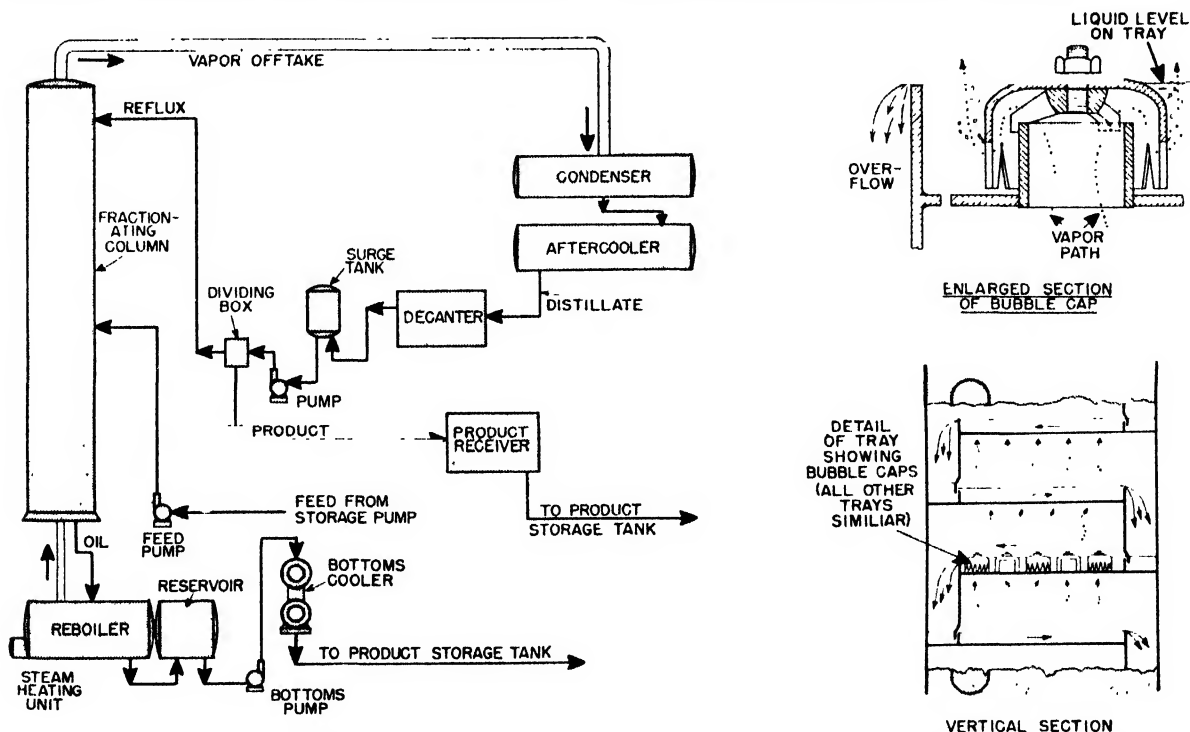


Fig. 5-13. (Left) Schematic diagram illustrating component parts and principle of operation of a modern continuous still. (Right) Schematic details of plates and bubble caps used in stills for fractional distillation. (Courtesy, Smet-Solvay Engineering Division, Allied Chemical and Dye Corporation).

control of the temperature, or this equivalent part may be returned by gravity flow. It is usually the better practice to effect the reflux by pump flow, to provide positive control of this operation.

In the operation of a batch still, the kettle is filled to its designed capacity which leaves space in the upper section to receive the vapors, and the steam is turned into the heating coils. As the material reaches the boiling point, it is necessary to control the rate of steam flow by the pressure or temperature in the kettle. The vapors ascend the column, are condensed in the condenser, and a predetermined amount of condensed liquid is returned to the column. It may all be returned for a while until the column is in product and temperature balance, at which time product may be removed and sent to storage. The vapors are forced into the fractionating column by the pressure in the kettle and pass through the bubble caps on each plate where they come in intimate contact with the downward flow of reflux. This contact provides a series of steps consisting of evaporation and condensation on each plate, and a temperature gradient is developed through the column such that the lowest temperature is at the top and the highest temperature is at the bottom. This temperature gradient in a batch still column is in a constant state of change as the lower-boiling-point materials are first distilled over and replaced with the heavier fractions, but each plate maintains a relative temperature differential between the plates above and below. During the progress of the distillation progressively higher-boiling-point materials are removed with consequent increase in the column and kettle temperatures. Intermediate fractions are always produced between the principal components, benzene and toluene. These products are stored separately and re-run in the still to maintain the maximum recovery of finished products.

Continuous Still Operation—A continuous still (Figure 5-13) consists of a kettle or reboiler of considerably smaller capacity than that of the batch process, because the feed material is being constantly pumped into the unit and the residue pumped out of the reboiler. It is equipped with a fractionating column, a condenser, and reflux control. There is usually more than one still unit in a group, because the hot residue from the first unit is used as feed to the second unit and this arrangement may be carried on with as many units as there are fractions to be recovered.

The feed to a continuous unit is pumped into the side of the fractionating column instead of into the kettle as in a batch still. In a batch still, a number of products are progressively removed from the top of the fractionating column, but in a continuous unit only one product is obtained from the top of a fractionating column, this product having the lowest boiling point of any component in the feed. Accordingly, there must be as many columns as products and intermediate off-grade fractions. The principles of fractional distillation apply to the operation of the column in regard to evaporation and condensation on each plate with a temperature gradient from top to bottom of the column. However, unlike the batch still, the conditions of temperature and pressure throughout the unit in the reboiler and column remain constant, and each plate operates at a constant temperature and pressure.

Control of Distillation Processes—The operations of batch and continuous distillation require control of various factors in the operating unit.

In batch-still operations, the principal control factors are the temperature of the vapor leaving the top of the column, rate of vaporization, and volume of reflux pumped into the top of the column. The vaporization is

controlled by the amount of steam introduced into the heating elements in the kettle, which can be done by hand operation of the steam valve or with a pressure controller actuated by the pressure in the kettle. The reflux provides a control of the temperature gradient in the column and makes it possible to remove in order the various components of the oil being fractionally distilled, the lowest-boiling-point fraction being the first removed. Other control factors are the level of oil in the kettle, rate of flow of product and reflux, temperature of condenser water in and out of the unit, and temperature of distillate.

Continuous distillation requires more control than batch in that all factors must be kept constant. A uniform feed of constant composition to the column is provided with a flow controller. Constant measurements of column temperatures, generally bottom of column, below feed, above feed and top of column, along with temperature of water in and out of the condenser, and temperature of distillate and bottom reboiler fractions, are taken with pyrometric recorders of the potentiometer type. Liquid-level controllers are used to regulate pumping of the bottoms of one still unit to the next still and also pumping of the distillate as it flows from the condensers. The oil vaporization is regulated by a pressure controller automatically operating the steam-header valve. Liquid levels in reboiler, surge and receiving tanks are indicated by instruments. Oil flows through pipe lines are indicated by rotameters and steam flows are recorded.

Type of Refining Process—Plant designs are classified as batch, semi-continuous and continuous, depending on the type of distillation.

Of all of the variables affecting the design of a plant, that of the size of the installation probably has the most bearing on whether the process will either be semi-continuous or continuous. It is not generally agreed just where the dividing point is. Small and medium-sized plants have the semi-continuous design and larger plants have continuous design.

One of the most elementary light-oil refining processes is to acid-wash a crude light oil containing no heavy solvent, and refine it into washed motor benzene, pure benzene, pure toluene, commercial xylenes, and pure still residue by either batch or continuous fractional distillation.

Refinement of light oil of this character would involve acid-washing the crude light oil, then provide settling and storage from which the washed light oil is charged either to batch stills for production of desired grades of pure products, or is fed continuously to the pure continuous stills. The first continuous column, a residue remover, removes the residue from the washed light oil. The second continuous column, a forerunnings stripper, takes off heads of the washed light oil as forerunnings or motor benzene. The third column takes off pure benzene. The fourth column takes off pure toluene with the commercial xylenes as a side cut or they can be pumped from the reboiler and accumulated for a batch distillation.

Residue removal prior to the separation of the oil into its principal components is an improvement in refining since it eliminates impurities in the pure products and the need for treatment of the still vapors or products with an aqueous solution of caustic soda. The continued heating of the residue, if it is a residual product of all the distillations, results in decomposition reactions of the polymerized materials producing sulphur dioxide, which, together with the hydrogen sulphide present, results in free sulphur.

Forerunnings, which are the lowest boiling point con-

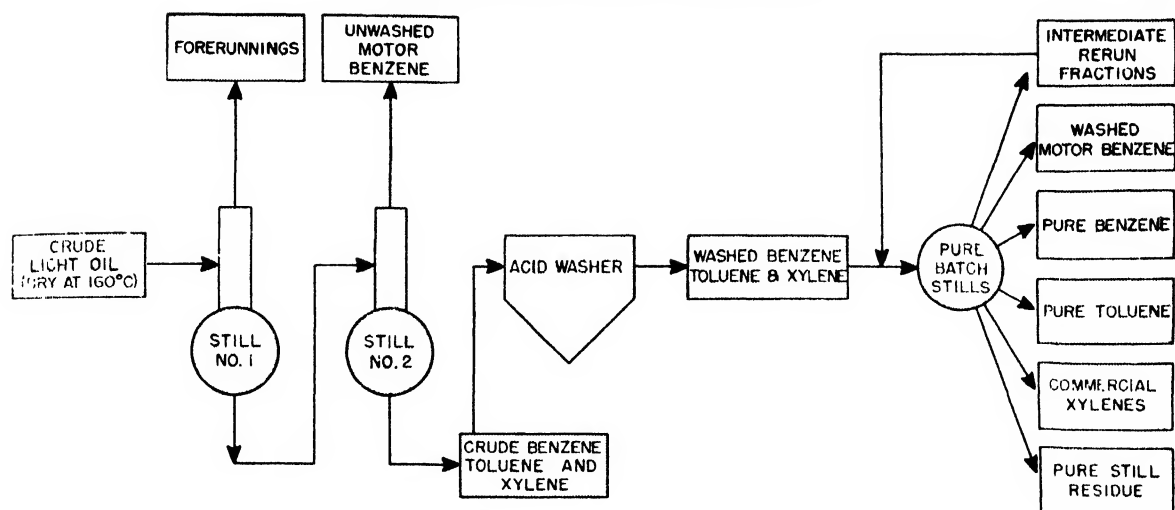


Fig 5-14. Flow sheet illustrating combined use of continuous and batch-type stills in the semi-continuous recovery of the products shown in the boxes at the right. The stills labeled No. 1 and No. 2 are continuous stills.

stituents of the light oil (such as hydrogen sulphide, methyl mercaptan, butadiene, butylenes, amylenes, cyclopentadiene, carbon disulphide, and other compounds) must be removed in any operation with a stripper column if they are present in the oil. This is necessary to produce good grades of motor benzene and pure benzene. If the sulphur impurities are sufficiently low in the light oil, all of the forerunnings can be added to the motor benzene fraction. Unfortunately, non-gum-forming unsaturated hydrocarbons which have good motor-fuel ratings are removed along with gum-forming unsaturates in acid washing.

The semi-continuous process is adapted to refining

light oils which require the removal of forerunnings and in which the production of unwashed motor benzene is desired. The general plan of design with a light oil dry at 320° F (160° C) would consist of two continuous stills followed by as many batch stills as desirable with an acid-washing process interposed between the two distillation units (Figure 5-14). The bulk of the distillation is carried out in the continuous crude stills. The first column is a forerunnings stripper and the second column takes off all or part of the benzene content of the light oil as unwashed motor benzene, the amount depending on the pure benzene requirements.

The motor-benzene fraction thus produced contains

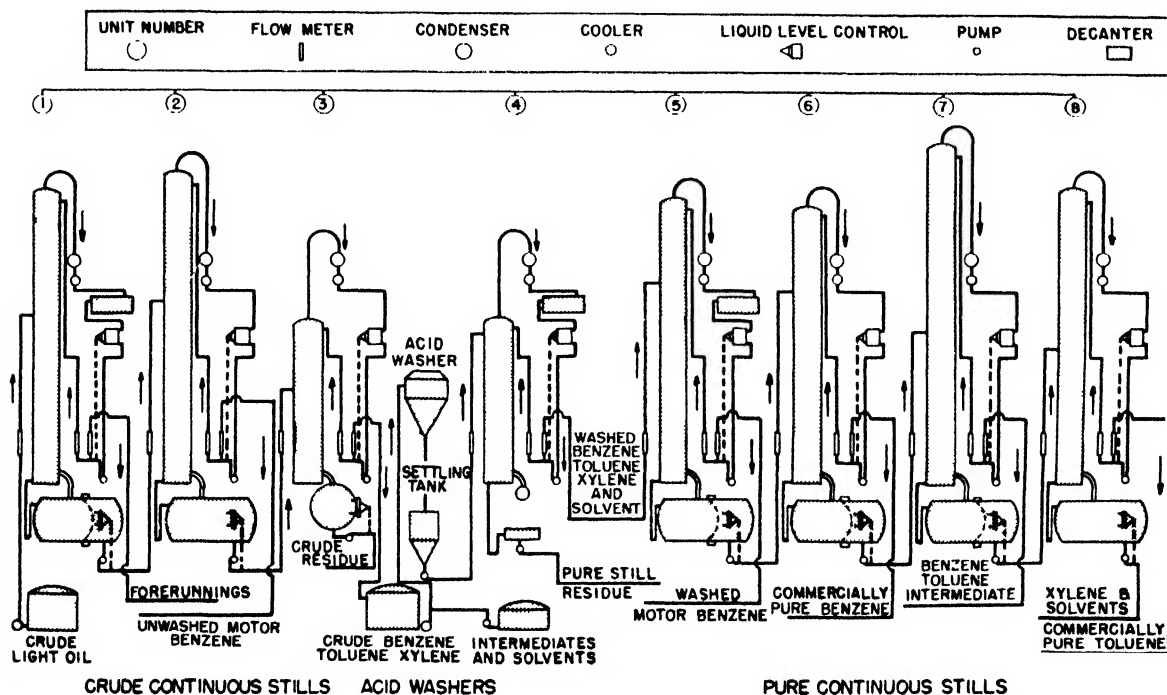


Fig. 5-15. Flow diagram for a continuous light-oil recovery process. (Courtesy, Semet-Solvay Engineering Division, Allied Chemical and Dye Corporation.)

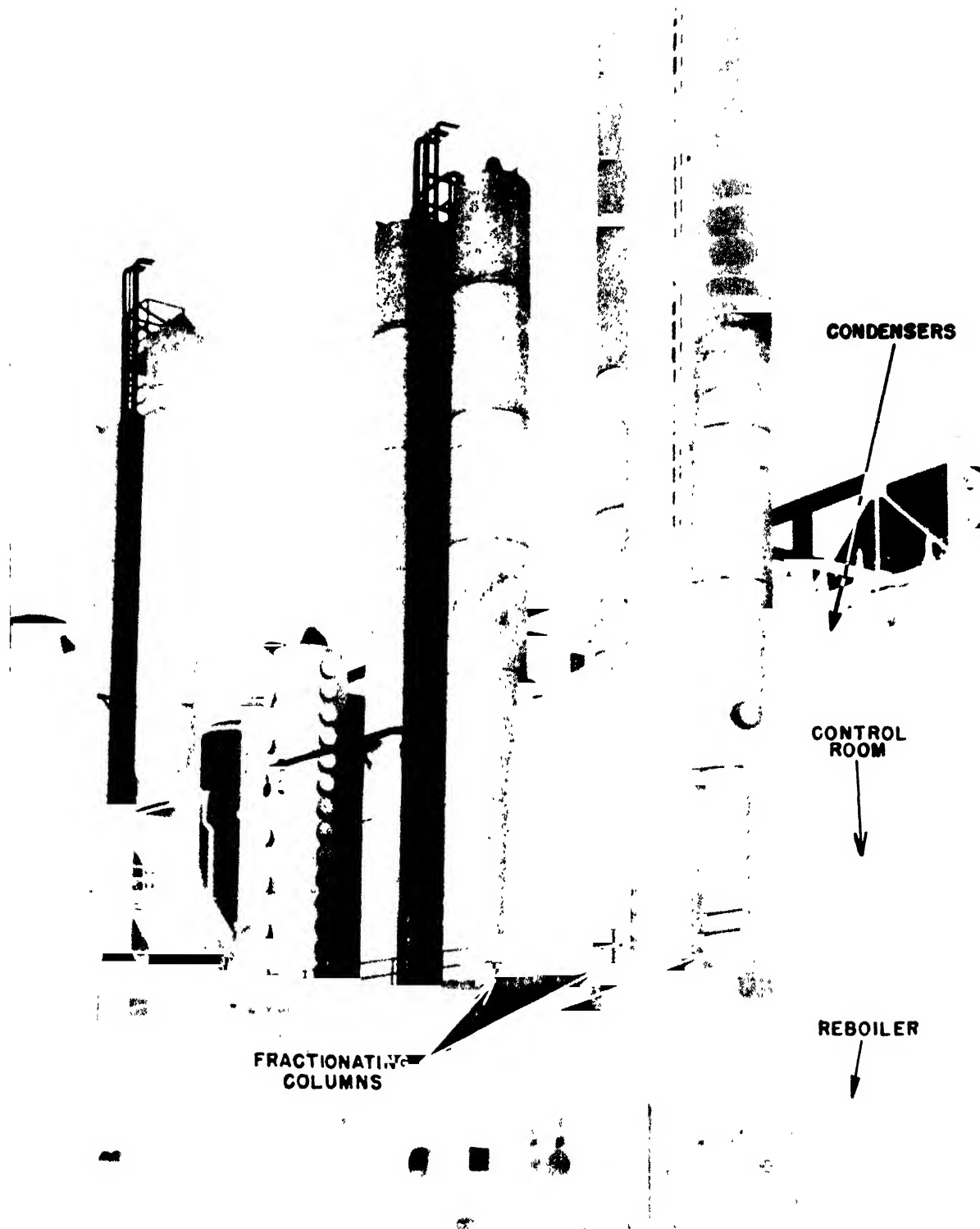


FIG. 5-16. Exterior view of a benzene plant showing distillation columns and other equipment for the continuous distillation of light oil.

impurities which would normally be removed by an acid-washing operation. If allowed to stand, certain of the unsaturated impurities polymerize to form gums which result in trouble in an engine burning such motor benzene. Formation of the gums in the motor benzene is prevented by addition of an inhibitor (an organic anti-oxidant), but none of the unsaturates or other impurities, which are desirable due to their fuel value, are removed by this treatment. After removal of this unwashed motor benzene, the fraction containing the remainder of the benzene, toluene and xylene is acid-washed and batch distilled for production of washed motor benzene, pure benzene, pure toluene and commercial xylene.

If light oil, containing heavy solvents, and having a dry point of 390°F (200°C) is processed, an additional column becomes necessary in the layout. This would be a continuous crude column and number three in the lineup. The overhead of this column would be benzene, toluene and xylene, which would be acid-washed. The bottoms of this column, crude residue consisting of used wash oil, solvents, and naphthalene, would be batch distilled to recover crude heavy solvent. The residue of this batch distillation, containing used wash oil and naphthalene, is processed in crystallizing pans for the recovery of crude naphthalene, while the used wash oil is returned to the wash-oil system.

Another technique used in the smaller installations is to have one batch still in addition to the continuous crude stills and operate part time continuous and finish up with batch operation. In this case, washed oil is fed

to the column and benzene is removed continuously, allowing the toluene and xylene to build up in the kettle. When the kettle becomes filled, the feed is discontinued and the unit is then operated as a batch still, and first the toluene distilled over, then the xylene.

In a continuous process (Figures 5-15 and 5-16), with a light oil dry at 390°F (200°C), the continuous crude unit would have three columns, the first for fore-runings, the second for unwashed motor benzene, and the third for producing the benzene, toluene and xylene overhead for subsequent recovery of pure products after acid washing. The residue, consisting of used wash oil, heavy solvents and naphthalene, is batch distilled.

The continuous pure unit would consist of four columns, the first for removal of pure still residues, the second for stripping the small volume of paraffins and other low-boiling constituents which may be in the washed oil subsequent to pure benzene production on the next column. This column can also be used to remove any part of the benzene content of the oil for washed motor benzene production. The third column produces pure benzene, and the fourth pure toluene as a top product, and xylene with a small volume of heavy solvent as a bottom product. The bottom product is distilled in a batch operation to produce commercial xylenes and a refined heavy solvent.

For the production of additional grades of product, or to remove any intermediates between products, it is necessary to add another column to the design layout since each column produces an individual fraction or product.

SECTION 9

REFINING OF COAL TAR

The volatile products resulting from the thermal decomposition of bituminous coal contain a yellow vapor as a finely divided suspensoid which, upon condensation, yields a black viscous material called coal tar. There are many types of coal tar produced, depending

on the temperature and conditions of carbonization. The coal tar which has the most important commercial significance is that produced during the high-temperature carbonization of bituminous coal. These coal tars consist essentially of aromatic hydrocarbons

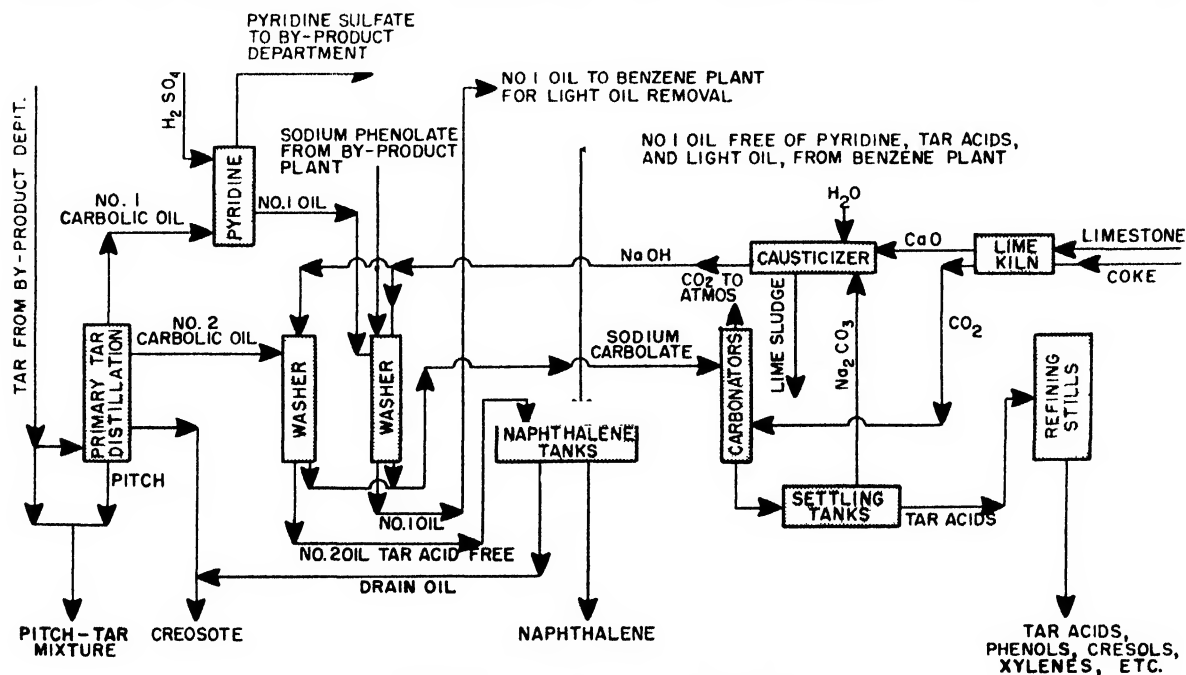


Fig. 5-17. Flow diagram of a tar plant.

which are characterized by ring-type chemical structure and contain such compounds as benzene, naphthalene, anthracene and their related homologues; oxygenated compounds usually referred to as tar acids which include phenol, cresols, and xylenes; nitrogen compounds in which the nitrogen is included in the ring structure such as pyridine, picolines, etc., and only very small amounts of the paraffinic compounds (Table 5-11).

This type of tar is used for the recovery of tar acids, naphthalene, creosote oils, pyridine bases and pitch (Figure 5-17). Their recovery requires the use of fractional distillation of the tar as the first operation, with subsequent use of fractional distillation procedures in some of the other operations, chemical extraction for the removal of acidic and basic compounds with subsequent purification and refining of the extracts, and crystallizing operations for the recovery of such compounds as naphthalene, anthracene, carbazole and phenanthrene.

These operations may be conducted as batch or continuous operations depending principally upon the quantities of material to be processed. The following description is restricted to one procedure that has been found satisfactory and does not necessarily represent operating practices throughout the industry.

Distillation of Tar—The raw tar, which is maintained at a temperature of about 140° F (60° C) in the storage system, is continuously pumped into the tar distillation system (Figure 5-18). In its course through the system, it is preheated to about 300° F (150° C) by indirect heat exchange where it absorbs some of the sensible heat of the creosote oil and pitch. It then enters the bottom bank of tubes in the tar heater where the temperature is increased to about 435° F (225° C) before entering the dehydration column. This column is equipped with three fractionating plates and its function is to permit the water and low-boiling tar oils to be liberated from the hot tar. The water and light oil leaving the top of the dehydration column are condensed and run to receiving tanks where the oil and water are separated by gravity. It is necessary to control the temperature of the column by returning a portion of the condensed oil distillate to the top so that excessive amounts of oil are not removed. This operation is necessary to maintain a differential in the specific gravity of the oil and water whereby the oil is maintained at a density of less than that of the water.

The residue from this column is free of water and leaves the base of the column at a temperature of 355-390° F (180-200° C). It is then returned to the high-temperature section of the tar heater where it is heated to about 750° F (400° C) for the second flash-distillation operation. This operation is conducted in a fractionating tower where the hot tar is transferred to a section near its base and vaporization of the volatile constituents takes place.

The residue is removed from the bottom of the column at a temperature of about 700° F (370° C) whereupon its temperature is reduced to about 390° F (200° C) through heat exchangers to heat incoming raw tar. It is then disposed of according to requirements. Pitch is used as a binder in making carbon electrodes, as roofing pitch, fiber pitch, in the formulation of pipe-line enamels, and as fuel. If it is to be used as fuel it may be mixed with undistilled tar in a special type of mixer in which the foaming due to mixing hot pitch and wet tar can be controlled, or it may be mixed with selected fractions of petroleum oil or such combinations as may be dictated by operating requirements. A liquid-fuel residue may also be produced by reducing the tem-

Table 5-II. Compounds Recovered from Coal Tar and Light Oil*

I. Neutral Compounds		
(From Light Oil)		
Name	Formula	Boiling Point, °C
Benzene	C_6H_6	80.1
Toluene	$C_6H_5CH_3$	110.6
Meta-Xylene	$C_6H_4(CH_3)_2$	139.3
Para-Xylene	$C_6H_4(CH_3)_2$	139.4
Ortho-Xylene	$C_6H_4(CH_3)_2$	144
Mesitylene	$C_6H_3(CH_3)_3$	164.6
Pseudocumene	$C_6H_3(CH_3)_3$	169.2
Coumarone	C_9H_8O	174
Indene	C_9H_8	182.4
(From Coal Tar)		
Naphthalene	$C_{10}H_8$	218
2-Methyl Naphthalene....	$C_{10}H_7CH_3$	241
1 Methyl Naphthalene ...	$C_{10}H_7CH_3$	245
Dimethyl Naphthalene....	$C_{10}H_6(CH_3)_2$	255-270
Acenaphthene	$C_{12}H_{10}$	281
Fluorene	$C_{13}H_{10}$	298
Phenanthrene	$C_{14}H_{10}$	340
Anthracene	$C_{14}H_{10}$	342
Fluoranthene	$C_{16}H_{10}$	382
Pyrene	$C_{16}H_{10}$	393
Chrysene	$C_{18}H_{12}$	448
II. Phenolic Compounds		
(From Coal Tar)		
Phenol	C_6H_5OH	181
Ortho-Cresol	$C_6H_4(CH_3)OH$	191
Para-Cresol	$C_6H_4(CH_3)OH$	201
Meta-Cresol	$C_6H_4(CH_3)OH$	202
2, 4-Xylenol.....	$C_6H_3(CH_3)_2OH$	211.5
3, 5-Xylenol.....	$C_6H_3(CH_3)_2OH$	219.5
Alpha-Naphthol	$C_{10}H_7OH$	280
Beta-Naphthol	$C_{10}H_7OH$	286
III. Nitrogen Compounds		
(From Coal Tar)		
Pyridine	C_5H_5N	115
2-Methylpyridine (Alpha-Picoline)	C_6H_7N	129
3-Methylpyridine (Beta-Picoline)	C_6H_7N	144
2, 6-Dimethylpyridine (Alpha Alpha' Lutidine). ..	C_7H_9N	144.4
4-Methylpyridine (Gamma-Picoline)	C_6H_7N	144.6
2, 4-Dimethylpyridine (Alpha Gamma Lutidine) ..	C_7H_9N	157.1
Quinoline	C_8H_7N	238
Isoquinoline	C_8H_7N	243
Quinaldine	$C_{10}H_9N$	247.6
2-Methyl Isoquinoline	$C_{10}H_9N$	252
4-Methyl Quinoline	$C_{10}H_9N$	264
Acridine	$C_{13}H_9N$	346
Carbazole	$C_{12}H_9N$	352

*This table does not list all of the compounds that have been identified. It does contain some compounds which are recovered commercially in very small amounts or not at all.

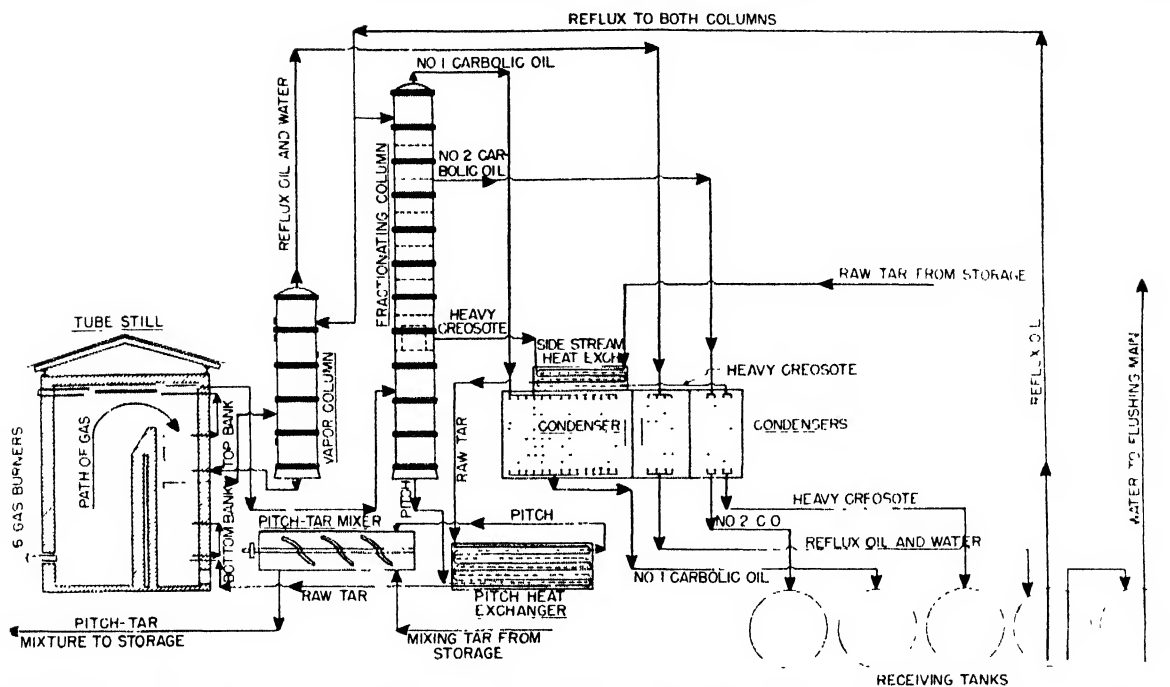


FIG. 5-18. Schematic representation of a primary tar distillation plant. (Courtesy, Foster-Wheeler Corporation.)

perature of distillation to permit the heavy creosote-oil fractions to remain with the pitch, which makes it unnecessary to blend any cut-back material with the residue. It has been demonstrated that this residue is a desirable liquid fuel for open-hearth operations as it has a heating value of 160,000 to 165,000 Btu per gallon.

The vapors passing upward within the column travel through tray sections equipped with bubble caps and downflow pipes to provide a fractional separation of the vapors into the several components with a prescribed distillation range. Reflux is provided by pumping a portion of the light distillate into the top of the column. This reflux provides control of the temperature gradient in the column and makes it possible to remove fractions of definite boiling points from selected trays at various levels in the column.

The operation of continuous distillation requires careful control at all points in the operating unit. The first requisite for successful operation of continuous distillation is a uniform feed of constant composition, which must be followed by careful control of the temperatures and pressures at all points in the system. The control points selected are the inlet and outlet temperatures and pressures of the heating section used for the dehydration operation, and the comparable points for the bank of tubes used to supply heat for distillation of the oils. The temperature at the top of each column must be controlled by regulation of the reflux oil that is pumped into these sections. The temperature of all condensers requires control to prevent the deposition of crystalline compounds such as naphthalene and anthracene.

The pitch removed from the base of the column may have a high solidification point, and its temperature must be controlled to provide easy flow in the pipe lines and heat exchangers.

The distillates recovered are separated into the following fractions: (a) a light-oil fraction (No. 1 carbolitic oil) with a distillation end-point of about 410° F (210° C) which contains, in addition to the normal light-

oil products, tar acids, pyridine bases and some naphthalene, (b) an intermediate-oil fraction (No. 2 carbolitic oil) with a distillation end-point of about 480° F (250° C) which contains tar acids and the major portion of the naphthalene, (c) an anthracene-oil fraction boiling above 480° F (250° C) in which the end-point is controlled by the temperature to which the tar is heated prior to entering the column.

The light-oil fraction is processed for the recovery of pyridine bases and crude tar acids, after which the light oil is removed by distillation and the residual naphthalene is processed for the recovery of salable naphthalene.

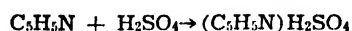
The intermediate-oil fraction is processed for the recovery of crude tar acids and then for naphthalene in the same process as noted above for the residual portion of the light-oil fraction.

The anthracene-oil fraction is blended with the drain oil from the naphthalene process for the production of various grades of creosote oil.

The residual pitch from the tar distillation is used for fuel in the open hearths and heating furnaces in the various steel-producing facilities. It is necessary to blend it with fuel oil or virgin tar to lower the viscosity prior to its burning as a fuel. Other uses previously mentioned are in the manufacture of carbon electrodes, roofing pitch, pipe-line enamels, and as saturating pitch.

A description of methods for the processing of the light-oil and intermediate-oil fractions will follow.

Pyridine Sulphate Recovery—Recovery of pyridine bases from tar oils is a comparatively simple process. Essentially, a solution of sulphuric acid is circulated through the oil, combining with the tar bases present to form pyridine sulphate according to the following reaction:



The pyridine sulphate is separated from the oil by gravity.

The first step in the process is dilution of the 60° Bé

sulphuric acid with water in a lead-lined tank equipped with an agitator. A solution containing about 17 per cent of commercial acid has been found to give best results.

About 8,500 gallons of fresh oil is pumped into the washer, along with 1,500 to 1,700 gallons of the acid solution which has been about two-thirds reacted by two previous washes. This effects a partial extraction of the pyridine, and completes the conversion of the sulphuric acid to pyridine sulphate. The crude pyridine sulphate solution is separated from the oil by decantation and is then ready to be rectified. The once-washed oil in the washer is washed again by acid solution which has been one-third satisfied by one previous wash. The oil is finally stripped of remaining pyridine bases by the third wash, using the fresh sulphuric-acid solution.

Rectification of the pyridine sulphate is the final step. It means simply that the sulphate solution is boiled in the rectifier by means of coils carrying live steam, the generated water vapor carrying away the unwanted oil, naphthalene and tar acids in the sulphate. These vapors are condensed and processed for the recovery of tar acids.

The rectified pyridine sulphate is then ready for springing with ammonia and distillation for recovery of refined pyridine, as described previously in Section 6.

The pyridine-free oil which has been separated from the pyridine sulphate by gravity in the washer still possesses a small acid content which must be removed to make it suitable for further processing. A weak solution of soda ash in water (about 12 to 14 per cent) is used for the purpose, giving the following reaction:

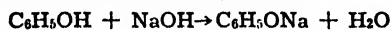


The reaction takes place as the oil is continually pumped through the neutralizer containing the soda-ash solution. The sodium-sulphate solution formed in the neutralizing reaction is a waste material and is disposed of in coke-quenching towers.

Pyridine bases also are recovered from saturator liquors, as described in Section 6 of this chapter.

Tar Acid Recovery—It is frequently difficult to make a choice between processes and equipment used in these processes for the recovery of tar acids from tar distillates. The processing steps involved, combinations of equipment used, and descriptions of the equipment follow.

Washing removes tar acids in the form of phenol ($\text{C}_6\text{H}_5\text{OH}$) and phenol homologues from the carbolite oils by simple contact with caustic soda to form sodium phenolate according to the equation:

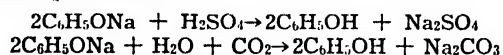


The washer is a suitably proportioned cone-bottom tank with a spray or other flow-distributing mechanism at the top. Steam-heated coils are provided in the bottom of the washer to maintain fluidity of naphthalene-bearing oils and a desirable reaction temperature of about 167° F (75° C). A mixture comprising 50 per cent carbolite oil and 50 per cent caustic-soda solution (8.5 per cent NaOH) is added to the washer. Caustic is circulated by pump from the bottom of the washer back into the top, filtering down through the carbolite oil by gravity and removing tar acids from the oil by contact. This process is preferred because more violent methods, such as mechanically agitating the oil and caustic together, tend to form emulsions which, on occasion, are almost impossible to separate. Oil and carbolite are readily separated by gravity when no emulsion exists.

Rectifying is a process required to remove undesirable pyridine bases (if not previously removed), oils and

naphthalene from the carbolite or sodium phenolate by boiling, usually under vacuum.

Springing (liberation of tar acids in sodium phenolate solution) can be accomplished by neutralization with sulphuric acid or carbon dioxide gas according to the following reactions:



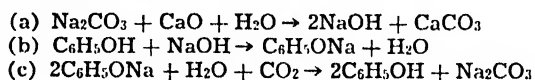
If sulphuric acid is used, no special equipment is needed except a tank equipped for agitation. Commercial acid is added to the carbolite during agitation to the extent needed to convert all the carbolite to tar acids and sodium sulphate. The tar acids and sulphate separate by gravity.

If carbon dioxide gas is used, more complicated equipment is necessary in the form of a series of 3 to 6 towers packed with grids or baffles. Carbolite pumped to the top of these towers trickles down through the grids, coming in contact with the CO_2 gas which is admitted to the bottom of the column and passes upward countercurrently to the flow of liquid.

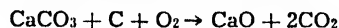
Method I (Figure 5-19) shows the minimum plant for tar acid recovery with three items of processing equipment—washer, rectifier and springer. The chief advantage of this plant is the very small investment required. The crude tar acids it produces may contain small quantities of sulphur compounds causing corrosion of equipment in the subsequent distillation of the tar acids.

Method II—The acid springer is replaced by carbon-dioxide springing towers (commonly called carbonating towers), and facilities for the production of lime and caustic soda added. A largely noncorrosive product is obtained at lower unit cost. Coke, limestone and small quantities of soda ash are required which produce the caustic soda for washing of oil and the carbon dioxide gas for springing of tar acids.

The particular virtue of this method lies in the fact that the sodium carbonate initially supplied to the system may be regenerated completely. The production of sodium hydroxide, the formation of sodium phenolate, the liberation of tar acids and the regeneration of the sodium carbonate are shown in the following series of formulas:



Lime (CaO) and carbon dioxide gas (CO_2) are produced in the vertical lime kiln from limestone (CaCO_3) and coke fuel, according to the equation:



Method III, as shown in Figure 5-19, is an all-continuous method in which the procedure and equipment are as illustrated.

Batch Rectifier—The boiling of the carbolite (sodium phenolate) produces a steam distillation which removes small quantities of naphthalene, oils and tar bases which gravity separation in the washers did not remove. The boiling is done under vacuum to permit use of low-pressure steam and to reduce hydrolysis of carbolites to tar acids which would be carried over with the vapors. This is a comparatively simple apparatus, the only design problem being provision of adequate heating and condensing capacities.

Batch Carbonating, or Springing System—The towers (about 25 ft. high) contain steel hurdles or baffles. Carbolite, circulated by pump from the reservoir at the bottom to the top of the tower, trickles down through

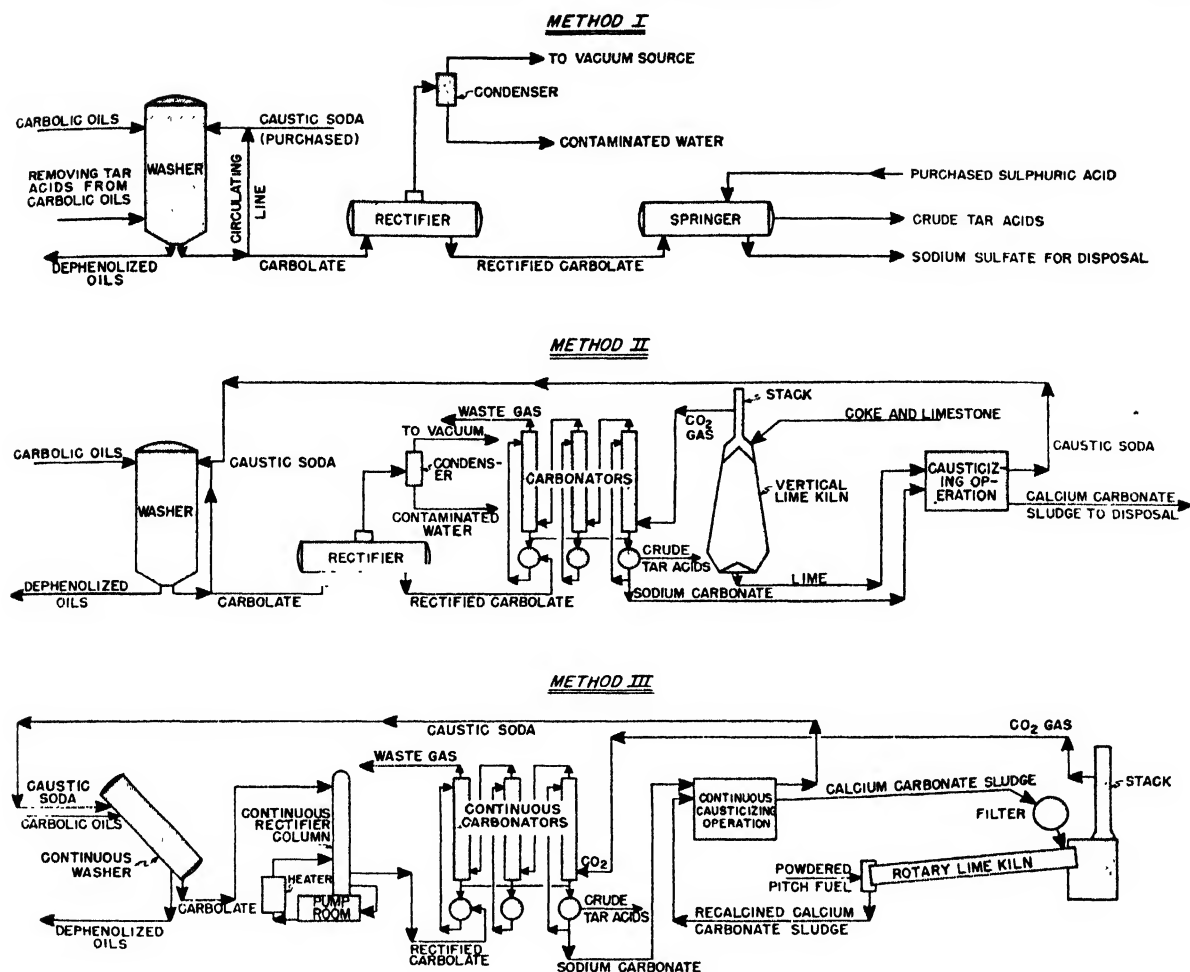


Fig. 5-19. Three methods for the recovery of crude tar acids.

the hurdles to be contacted by the CO_2 gas moving countercurrently to the liquid. The CO_2 gas is delivered to the carbonators from the lime kiln by rotary positive blowers. The following description applies to a system employing four towers.

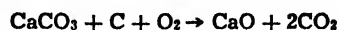
When the reaction has reached completion in the last carbonator (No. 4), the solution of free tar acids, carbonate and a low concentration of carbolate is pumped from the reservoir, and this reservoir recharged with the solution from the reservoir of No. 3 carbonator. The reaction in this solution has reached the stage of being three-fourths completed, having a higher concentration of carbolate and a lower concentration of free tar acids and carbonate than No. 4 carbonator. In turn No. 2 and No. 1 carbonator solutions are pumped to the carbonators ahead, No. 3 and No. 2 respectively, and No. 1 carbonator is recharged with fresh carbolate. In this way, the highest concentration of CO_2 gas is caused to react with the lowest concentration of carbolate in No. 4 carbonator, while the almost spent gas contacts the strongest concentration of carbolate in No. 1 carbonator. This system of circulation is designed to achieve maximum usage of the CO_2 gas available.

Here, the design problem is mainly one of capacity. If the contact time between liquid and gas is adequate, about 180 cu. ft. of 30 per cent carbon-dioxide gas will suffice to liberate each gallon of tar acid. Since salts have a tendency to build up on the hurdles in the towers, it

is necessary to provide means for removing and cleaning of hurdles. Ordinary materials of construction can be used throughout.

Lime Kiln—In order to reduce to a minimum the carry-over of undesirable salts into the subsequent chemical reactions, limestone of maximum purity is required for the lime kiln, as well as for most other chemical processes. The coke fuel must be carefully chosen as to size in order to be completely and efficiently consumed in the burning area of the kiln.

In operation, limestone and coke in proportions of 8 to 10 pounds of stone to one of coke are charged by skip hoist into the charging bell at the top of the kiln at regular intervals. Burned lime is removed either continuously or intermittently by rotation of the mound-shaped hearth at the bottom. Stack gases pass from the cast-iron acorn (situated at the center of the kiln near the top) to the atmosphere by way of the stack on one side of the acorn, or to the boosters for use in the system by way of the opposite side of the acorn. The reaction taking place in the kiln is chemically expressed by the equation:



or, in other words, limestone plus coke (carbon) plus air (oxygen) produces lime and carbon dioxide gas when heat results from burning of the coke. The stack gases produced are first passed through a cyclone dust

catcher of conventional design and a gas scrubber before taking part in the chemical process. The scrubber is simply a limestone-packed tower into which water is sprayed at the top. The kiln gases, entering at the bottom and leaving at the top, are cooled and stripped of their water-soluble constituents.

Chief design problems for local determination concern facilities for unloading, sorting, screening and proportioning the limestone and coke for charging into the kiln, along with screening, crushing, conveying, storing and distribution of the resulting lime.

Caustic soda (NaOH) is produced by bringing soda ash (Na_2CO_3) and lime (CaO) together in the proper proportions and at the right temperature in the presence of water according to the equation:



Batch Caustic System—A solution of soda ash in water of the desired quantity and strength is added to the batch causticizers (or causticizing tanks), and heated to about 90° C. The necessary quantity of lime is then delivered to each causticizer and agitated by slowly rotating blades. Heat of reaction raises the temperature of the charge to about 98° C. After reaction is complete, agitation is stopped and the batch allowed to settle, the calcium-carbonate sludge settling to the bottom. After settling, the clear caustic soda is skimmed off the top, following which the calcium-carbonate sludge is filtered to recover all possible liquid materials. The filter cake is disposed of to a dump to waste disposal.

This is a comparatively simple and flexible system involving no particular problems of design or materials of construction and can be installed at minimum cost. It presents an operating problem, however, since the heat of reaction when lime is added can readily cause the batch to overflow. Moreover, it is impossible, with this kind of equipment, to prevent carry-over of small quantities of calcium-carbonate sludge with the caustic soda. This entrained sludge distributes itself all through the tanks and processing equipment used in the recovery of crude tar acids, necessitating periodic cleaning of facilities.

Continuous Causticizing System—By this method a predetermined concentration of soda ash in water, together with the proper proportion of crushed lime, is fed continuously into a slaker. A tubular steam heater automatically maintains the soda-ash solution at a temperature of 207° F (97° C) at the entrance to the slaker. The slaker is a rotating cylinder which serves the dual purpose of completely slaking the lime and eliminating large solids such as unburned coke and raw limestone cores. The mixture passes, by gravity, into a classifier which removes small solids. The mixture is pumped from this point to three overhead, cascaded mixers, or causticizers, equipped with agitators which serve the purpose of allowing necessary time for the chemical reaction to be completely finished before entering the thickener. The completely reacted mixture of caustic soda and calcium-carbonate sludge flows into the top center of the thickener, where clear caustic soda is continuously drawn off the top outer rim of the thickener as washed calcium-carbonate sludge is pumped off the bottom. The sludge, after filtering, may be recalcined in a rotary kiln or sent to waste disposal.

This system, while more expensive to build than a batch plant, has the advantage of requiring much less space for a given capacity.

Rotary Kiln—Calcium-carbonate sludge from the thickener is dried in the vacuum rotary filter shown above the charging end of the kiln in Figure 5—19. The filter cake is dropped to a worm conveyor which charges

the cake into the rotating, brick-lined cylinder. Rotation of the slightly inclined kiln moves the sludge slowly toward the discharge end as the burning fuel (coke oven gas in this case), converts the sludge to lime with the same chemical reaction as that which takes place in a vertical kiln. A temperature of about 1650° F (900° C) in the flue is maintained.

The design problem for this type of kiln is exceedingly complex especially where high CO_2 content in flue gases is specified for use in the process.

Tar-Acid Refining—The refining of crude tar acids is a distillation problem similar, in most respects, to the problems discussed at length in benzene refining. Crude tar acids are fractionated to produce the comparatively pure components, usually as phenol, ortho cresol, meta-para cresol, and xylenes. Meta-para cresol is composed of two close boiling compounds, meta cresol and para cresol, differing in boiling point by about one degree Centigrade. Xylenes, composed of six isomers boiling between 410° F (210° C) and 437° F (225° C) and higher boiling tar acids, may be cut into such fractions as the market demands and the distilling facilities permit.

Only two essential items of equipment are involved (other than such obvious items as tanks, pumps and piping), a pot still or heater, and a fractionating column. The number of trays in the columns would be dictated by the degree of purity desired in the finished product. Known installations vary from 40 to 80 trays or plates per column. The present trend is towards more trays to obtain products of higher purity. Distillation can be made at atmospheric pressure or under vacuum.

Horizontal Pot Still—The horizontal pot still consists of a horizontal tank resting upon a brick furnace with about one-third of its surface exposed to the heat. Gas is used as fuel. The arched brickwork of the combustion oven is studded with small flues or perforations which admit the heat to the bottom of the still without exposing the still bottom to direct flame. A disadvantage of this type of pot still is the comparatively small heating surface. Vapors from the boiling tar acids pass through the small dome at the top of the still to the fractionating column for effecting separation in a typical batch distillation process. One of the problems of tar-acid distillation by this process is the necessity for frequent cleaning of salts and decomposition products from the inner surfaces of the pot.

Vertical Pot Still—This is a vertical tank resting upon a brick furnace with about two-thirds of the surface exposed to the heat. Residue removal from the pot is difficult, and the tail pipe (drain line) frequently is difficult to clean and maintain. Direct application of high heat not only causes a certain amount of decomposition of the tar acids with attendant loss of product but also causes burning and rapid deterioration and corrosion of the pot itself; the maintenance is, therefore, high.

Steam-Heated Vacuum Batch Still—The kettle, or pot, is simply a horizontal tank which serves as a reservoir for the crude tar acids and a chamber for the release of the vapors. The contents of the kettle are continuously circulated by pump through a heater and back into the top of the kettle where vapors are released and pass into the bottom of the column for fractionation. Since the whole system is maintained under high vacuum, steam at 150 pounds pressure is adequate as a source of heat. The heater consists of a group of steam tubes around which the flow of tar acids is directed by a series of baffles.

This method of distilling tar acids reduces both decomposition of the material handled and corrosion of

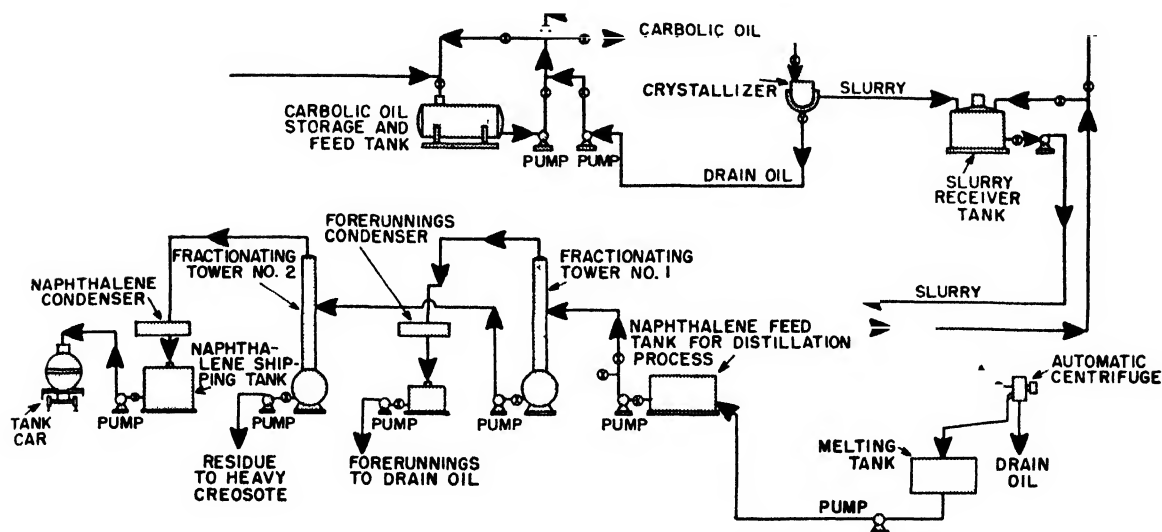


Fig. 5-20. Schematic representation of steps and equipment used in the recovery of naphthalene.

equipment. Standard materials of construction may be used throughout. Use of stainless steel in the vapor line, condenser and product line prolongs life and reduces contamination of products.

Naphthalene Recovery—The distillation of tar produces an overhead fraction consisting of carbolite oil, naphthalene and other neutral oils. Following the removal of tar acids by caustic washing, the remaining oils and naphthalene are pumped to equipment where the naphthalene is recovered by crystallization and/or distillation (Figure 5-20). The crystallization step is necessary sometimes since there are impurities present that cannot be removed by distillation alone.

The crude separation of the naphthalene where crystallization is necessary is accomplished in a series of continuous crystallizer units, each water-jacketed for cooling purposes and equipped with a slow-speed spiral stirrer to keep the crystals in suspension in the oil. The crystallizer feed tanks are maintained at 170° F.

The crude naphthalene crystals obtained from these crystallizer units are centrifuged, melted, and then pumped to the distillation section for refinement. To maintain a uniform feed for the centrifuges, the slurry from the crystallizer units at a temperature of about 70° F is first passed into an agitated slurry tank. From this slurry tank, the slurry is pumped to automatically controlled centrifuges which separate the naphthalene

crystals from the oils in the slurry. Part of this mixture of oils is recycled to the crystallizers while the remainder is pumped to storage. The centrifuged naphthalene crystals, on the other hand, are dropped into melting tanks to be melted by indirect steam and then pumped to a feed tank for the distillation section.

This distillation equipment is designed to refine the melted naphthalene by separating it into a light oil, refined naphthalene and a heavy oil. The first distillation column receives the melted naphthalene from the feed tank, which is maintained at 200° F. This column produces an overhead fraction of light neutral oil which is condensed in a water-cooled condenser and passed to a surge tank; reflux from this tank is returned to the first column at a controlled rate. The residue is pumped to the second fractionating column at a controlled rate. An overhead of naphthalene is produced in the second column and condensed in a water-cooled condenser. The condensed naphthalene is received in a shipping tank maintained at 250° F from which it can be pumped directly to tank cars or tank trucks for shipment. The residue from this second column is cooled to about 250° F and pumped to storage. The neutral oils after naphthalene recovery generally are blended with high-boiling anthracene oils also separated from tar to produce specification grades of creosote oils for use as wood preservatives.

SECTION 10

USES OF COKE, COKE-OVEN GAS AND COAL CHEMICALS

Metallurgical Coke—Coke is used for production of iron in blast furnaces and the coke dust as a fuel for steam generation in boiler houses.

Fuel Gas—After the recovery of coal chemicals, the gas provides fuel for heating the coke ovens, and the excess gas goes to the steel-producing units for heating open hearths and reheating furnaces. When practicable, other gas of lower Btu value may profitably replace the coke-oven gas for firing the ovens.

Ammonium Sulphate—The ammonium sulphate recovered from coke-oven gas is used for admixture with phosphate and potash constituents to provide balanced agricultural fertilizers for the various requirements, or

it may be used for direct application where nitrogen is the only requirement at the time of use.

Phenol—Phenol (C_6H_5OH), sometimes called carbolite acid, is recovered from both coal tar and ammonia liquor, and has the lowest boiling point of any of the tar acids. Its most important use is in the manufacture of resinous condensation products by reaction with formaldehyde, e.g., "Bakelite". As a chemical intermediate it is used in the preparation of synthetic tannins, dye intermediates, perfumes, plasticizers, picric acid, salicylic acid, and in the refining of lubricating oils.

Ortho Cresol—Ortho cresol ($CH_3C_6H_4OH$), another tar acid, is also used in the production of synthetic

resins to control the plasticity of the resin. It is nitrated to produce insecticides and weed killers. It is used in various organic syntheses and in the production of artificial flavors and perfumes.

Meta-Para Cresol—Meta-para cresol ($\text{CH}_3\text{C}_6\text{H}_4\text{OH}$) is a combination of two tar acids with boiling points too close together (394 and 396°F — 201 and 202°C) to permit separation at present with available plant scale distillation facilities. As in the case of the two previous tar acids, its chief use is in the production of synthetic resins and the plasticizer tricresyl phosphate. It is also used in organic synthesis and in the production of insecticides, dyestuffs, pharmaceuticals and photographic compounds.

Xylenes—Xylenes ($\text{C}_6\text{H}_4\text{OH}(\text{CH}_3)_2$) are composed of six isomers, with boiling points ranging from 410 to 437°F (210 to 225°C). Their uses are much like the other tar acids. Commercial xylenes are used as solvents for paints, enamels, varnishes, lacquers, azo dyes, aniline resins, acid proofing and rubber cement. The better grades, 3° and 5° xylenes are used for trinitro-xylene, an explosive, and as raw materials for manufacturing pharmaceuticals, mothproofing agents, perfumes, sulfonic acids and xylol bromide, a poison gas. Some of them find application as flotation oils used in the refining of ores.

Naphthalene—Naphthalene is recovered both from tar and light oil. It has the formula C_{10}H_8 and is one of the most versatile and important of the industrial organic chemicals, being an essential raw material for the preparation of a great many products. It is recognized by everyone in the form of moth balls. Its partial oxidation product, phthalic anhydride, is used quite extensively in the production of resins, lacquers and plastics of various kinds and for dyestuffs for fabrics, paper and paint. Derivatives of naphthalene are used in the manufacture of explosives, soaps, photographic chemicals, wetting agents, insecticides, disinfectants and pharmaceuticals, synthetic tannins, oils, fuels and in organic syntheses, all of which include only a part of its widespread present and potential application.

Creosote Oil—Creosote oil (from coal tar) constitutes a large part of the distillate from tar, and is a blend of different fractions to meet specifications established by the American Wood Preservers Association. Practically all of it goes into the pressure impregnation of wood, such as piling, telephone poles and railroad ties.

Pyridine Bases—The pyridine bases produced from the lighter tar oils include pyridine ($\text{C}_5\text{H}_5\text{N}$), the picolines ($\text{C}_5\text{H}_4\text{N}\cdot\text{CH}_3$), lutidines ($\text{C}_5\text{H}_3\text{N}\cdot(\text{CH}_3)_2$), and some quinoline ($\text{C}_8\text{H}_7\text{N}$).

Pyridine is an excellent solvent and is used as such in the rubber, paint and plastic industries. It is a basic material for chemical organic synthesis of industrial compounds, such as piperidine, etc. It is also used in the production of chemicals for waterproofing fabric and as

a carrying agent for impregnation of the fabric with the waterproofing agents.

The picolines are also basic materials for organic chemical syntheses. It is also used for production of nicotinamide and niacinamide, both vitamin substances used for augmenting the vitamin content of various foods and also as direct treatment for dietary deficiencies.

Pyridine bases are recovered from saturator liquor as well as tar.

Pitch-Tar Mixture—Pitch, mixed with virgin tar in proportions that will maintain fluidity, is an open-hearth fuel. Pitch is also employed as a binder in making carbon electrodes, roofing pitch, fiber pitch, and in pipe-line enamels.

Forerunnings—The forerunnings from light-oil refining contain cyclopentadiene, which can be processed to produce dicyclopentadiene. The polymerized products are useful as plastics and resins for use in paints and protective coatings.

Pure Benzene—Recovered by refining light oil, benzene is industrially the most important member of the aromatic family. It is used for the manufacture of a large proportion of all the coal-tar dyes, styrene monomer for synthetic rubber and polystyrene, aviation gasoline, dichlorobenzene and synthetic phenol. Plastics such as nylon, alkyd and polystyrene resins are end products derived from pure benzene. It is a chemical raw material for organic chemicals and is used as a general solvent.

Motor Benzene—While large quantities have been used in the past as an automotive fuel, its present use for this purpose has greatly diminished. It has a high anti-knock rating and has been used for this purpose on various occasions.

Pure Toluene—Toluene is used for the production of trinitrotoluene (T.N.T.), an explosive. Some use is found in the production of sulfonamide-formaldehyde resins for lacquers and adhesive formulation. It is a solvent for varnishes, waxes, resins, lacquer stains, enamels, dopes, cellulose esters and ethers. It is used by the oil-cloth industry, as a degreasing agent for bone and hair, as an extractant for animal and vegetable oils, and for printing ink and artificial leather. It is a raw material for chemicals and various dyes.

Crude Heavy Solvent—Crude heavy solvent is the source of coumarone-indene resins, which are thermoplastic resins used for rubber compounding, floor tile, printing inks, lacquers and chewing gum. It is used for ship-bottom paints, pipe coatings, shingle stains, wire enamel, brake linings, and bituminous and other dark-colored paints.

Refined Heavy Solvent—Refined heavy solvent is used in slow-drying solvents for manufacture of shoe polish, flat wall and interior paints, house paints, printing ink and enamels. It also is used in the linoleum industry.

Chapter 6

IRON ORES

SECTION 1

ORES AND THE IRON-BEARING MINERALS

Minerals and Ores—Any homogeneous inorganic substance having a fixed chemical composition or a definite range in composition and occurring naturally in the solid state is called a **mineral species**. A mineral, therefore, may be either an element or a compound of two or more elements. A few elements, like gold and platinum, occur for the most part native. Others, like silver, copper, mercury, sulphur, and carbon, may be found both native and combined. Most minerals, of which more than 1,500 species have been discovered and named, such as quartz, feldspar, hematite, hornblende, calcite, mica, etc., or their varieties, represent definite chemical compounds. A natural deposit of but a single mineral is seldom encountered in nature because the many forces at work usually produce conditions favorable to the creation or co-existence of more than one mineral in the same environment. While such a deposit is of course ideal, most natural deposits of which the ores are constituted are comprised of several minerals, the undesirable ones collectively being termed the **gangue** of the ore. In general, then, an **ore** is defined as a mineral or a mixture of minerals from which one or more elements may be extracted with profit.

The Iron-Bearing Minerals—There is a vast number of mineral species that contain iron. Only a few are of any importance commercially, because, in most cases, either the iron content is too low to justify the extraction of the metal or the mineral itself does not occur in sufficient abundance to make it available for use as an ore. Grouped according to their chemical composition, the iron-bearing minerals of chief importance are divided into four classes; namely, the iron oxides, iron carbonates, iron silicates, and iron sulphides. Of these, only the first and second classes may be considered as a factor in the manufacture of steel in the United States at present, although silicates have some promise of being used when cost factors change.

Table 6—I. Chief Iron-Bearing Minerals

Chemical Name	Mineralogical Name
1. Ferrosoferric Oxide	Magnetite
2. Anhydrous Ferric Oxide	Hematite
3. Hydrous Ferric Oxides	Limonite and others
4. Ferrous Carbonate	Siderite
5. Iron Silicates	Chloropal and others
6. Iron Sulphides	Pyrite and others

Magnetite Group—The only important mineral of this group is **magnetite**, chemical formula Fe_3O_4 , composed of iron, 72.4 per cent, and oxygen, 27.6 per cent. It varies in color from gray to black, has a specific gravity of about 5.0, and is magnetic. Advantage is taken of this last named property in locating ore bodies below the surface of the ground, and in mechanically purifying

ores of this group by magnetic concentration. Magnetite is often found closely associated with igneous rocks, in which case it is likely to contain appreciable amounts of chromium or titanium oxides which cannot be removed from it by magnetic concentration. In beneficiation operations, the ore is crushed to the fineness required, placed on a broad belt, and passed over a strong magnet, usually revolving. The revolving magnet holds the magnetic ore on the belt until after the non-magnetic gangue has dropped off into one chute, then the ore can be collected in another chute. Only about 5 per cent of the world's supply of iron is obtained from magnetite.

Hematite Group—The typical mineral of this group is hematite, which contains the equivalent of 70 per cent metallic iron, based on the chemical formula Fe_2O_3 . It furnishes the base of the world's most important ores. Being associated with rocks of various geological periods, these ores occur widely distributed, and in a variety of forms, which differ greatly in their iron content. Many of these varieties are known from their outstanding characteristics as, **red hematite**, **specular hematite**, **öolitic hematite**, **fossil ore**, and **Clinton hematite**. The last sometimes contains enough lime to be self-fluxing. The pure mineral has a true density or specific gravity of about 5.0 but the specific gravities of its ores vary from about 1.6 to 5.0 due to porosity.

Limonite or Brown Ore Group—Limonite is a field term generally referring to natural hydrous iron oxides whose real identity is unknown. Ores of the limonite group contain variable amounts of water and they do not all conform strictly to the preceding definition of mineral species. As a group they may be represented by the general formula $\text{Fe}_2\text{O}_3 \cdot n\text{H}_2\text{O}$. The most common member of the group, **goethite** is a definite mineral species with the formula $\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$. Terms applied to the other members of this group in order of their progressive increase in water content are **turgite**, **limonite**, **xanthosiderite**, and **limnrite**. On a theoretical basis, the iron content of this series will vary from 52.31 per cent to 66.31 per cent. Their true densities vary from 3.6 to 4.7, and the apparent densities of their ores from 1.5 to 4. These minerals are widely distributed throughout the world.

The Carbonate Group—The representative member of this group is the mineral known as **siderite**, or iron carbonate, FeCO_3 , which has a density of about 3.5 and contains 48.3 per cent iron. Owing to the fact that carbonic acid is dibasic, a part of the iron required to neutralize it may be replaced by other metals, thus giving rise to a series of minerals, such as iron-calcium carbonate, iron-magnesium carbonate, etc. Some of the names commonly applied to these ores are **spathic iron**

Table 6—II. Major Available Iron Ores *

Continent	Country or Area	Kind of Ore	Reported Reserves (Thousands of Tons) *	Approximate Range in Composition (Per Cent)					
				Fe **	SiO ₂	P	S	Al ₂ O ₃	CaO
Africa	Algeria & Tunisia	Mainly Hematite	140,000	50-55	3-4	0.01-0.15	0.01-0.1	1-5	up to 1
	British Africa	Hematite, Limonite, Magnetite & Spathic	175,000	37-70	1-15	0.1 -0.3	0.06-0.15	1-7	—
	French Morocco	Limonic	60,000	40-55	20-25	0.01-0.8	—	10	—
	French West & Equatorial Africa	Mainly Lateritic	1,400,000	45-55	3	0.06	0.10	—	—
	Liberia	Magnetite & Hematite	45,000	60-68	2-3	0.07-0.08	0.04-0.05	—	—
Asia & Adjacent Islands	South Africa	Magnetite, Hematite, Limonite	465,000	38-62	7-15	up to 0.05	—	2 max.	—
	Spanish Africa	Hematite & Magnetite	30,000	50-62	3-9	0.01-0.1	—	—	—
	Total Africa		2,315,000						
	China	Hematite, Magnetite & Brown Hematite	1,075,000	40-65	1-25	0.01-0.1	0.01-0.5	—	—
	India	Hematite	4,550,000	60-70	1-3	0.02-0.12	0.02-0.12	1-5	—
Australia & Oceania	Japan	Mainly Magnetite	35,000	30-44	—	high	high	—	—
	North Korea	Magnetite & Hematite	400,000	25-35	up to 45	0.1	0.01	—	up to 2
	Others	Brown & Hematite	150,000	various					
	Total Asia		6,210,000						
	Australia & Tasmania	Hematite	290,000	60-68	1-3	0.04-0.09	0-0.02	2-3	—
Europe (Including Asiatic USSR)	Austria	Spathic	115,000	32	6	0.02-	0.03-	3	11
	France	Minette Hematite, Spathic	4,200,000	25-40	6-24	0.4 -0.8	0-0.13	4-7	4-19
	Germany	Brown, Hematite, Spathic	3,075,000	15-35	12-29	0.4 -1.0	—	—	3-4
	Luxembourg	Spathic & Minette Hematite	235,000	25-28	—	0.5 -0.7	—	—	—
	Norway	Mainly Magnetite	610,000	40-70*	—	0.04-0.7	0.05-2	—	—
North & Central America	Soviet Orbit	Hematite, Magnetite, etc.	3,070,000	25-61	3-28	0.05-6.0	0.06-0.27	0-8	0-4
	Sweden	Mainly Magnetite	2,550,000	32-65	—	0.01-2.5	—	2-10	3-28
	United Kingdom	Spathic & Brown	2,160,000	18-35	7-20	0.03-0.72	0.05-0.42	1-8	1-7
	Yugoslavia	Spathic, Hematite & Brown	140,000	33-62	2-14	0.02-0.60	0.03-1.2	—	—
	Others	Various	75,000	—	—	—	—	—	—
South America	Total Europe		16,230,000						
	Canada	Mainly Hematite	3,335,000	35-58	3-16	0.03-2.0	0.03-2.4	2-5	2-3
	Mexico	Magnetite, Hematite, etc.	155,000	50-68	—	0.06-0.54	0.03-0.96	—	—
	United States	Hematite, Brown & Magnetite	2,555,000	35-63	3-14	0.02-0.5	low	1-12	0-1
	Others	Magnetite, Hematite, etc.	95,000	58-68					
South America	Total North America		6,140,000						
	Argentina	Hematite, Magnetite	30,000	41-55	3-19	0 -1.7	0.05-0.7	3-6	—
	Brazil	Mainly Hematite	2,690,000	40-68	—	up to 0.05	low	low	—
	Chile	Hematite, Magnetite	260,000	68	—	0.03-0.07	0-0.03	—	—
	Peru	Magnetite	100,000	56	—	—	0-0.5	—	—
Total World	Venezuela	Mainly Hematite	920,000	55-65	up to 1	0.02-0.13	0.04-0.64	0-4	—
	Total South America		4,000,000						
			35,185,000						

* Data from sources believed to be reliable

** Mainly natural Fe

x-Concentrates

ore, black-band ore, etc. Usually, carbonate ores are calcined before they are charged into the blast furnace. Frequently, they contain enough lime to be self-fluxing.

The Silicate Group—Iron silicates are of more interest as original sources of iron ores than as sources for iron. Like the carbonates, the silicates may be double salts in which iron forms only a portion of the base. Silicates in which iron forms the principal base include *chamoisite*, *chloropal*, *cronstedtite*, *hisingerite*, *stilpnomelane*, *greenalite*, *minnesotaite*, *grunerite*, *daphnite*, *thuringite*, and others. Silicates are not mined as a source of iron except when they occur associated with other iron-bearing oxides.

Distribution and Supply of Iron Ores—Table 6—II is an attempt to indicate the wide distribution of iron ores, the kinds of ore and the estimated available supply for each country of the world based upon data from many sources believed to be reliable. Available ores include those which can be mined and used in their natural state, and those amenable to beneficiation by present methods at a cost to compete with natural ores. In addition there is an even greater potential supply that includes ores which, on account of their geographical location, chemical composition, or physical condition, cannot be commercially exploited in competition with available ores by present methods of beneficiation, transportation and smelting. The table shows that the world supply of available ore is roughly 35,000,000,000 tons and that while about 1/14 of this supply lies within the United States, other countries are also well supplied; some so well that strong competition in the iron and steel business is assured. However, the business does not rest upon ore alone, but also upon coal, fluxing materials, refractories and water, all of which are required in the manufacture of iron and steel. Even with all these raw materials available or close at hand, much depends on the ingenuity, resourcefulness, and

intelligence of the men who make the products and manage the business, and especially of the people of the nation so favored. Table 6—III summarizes the average proportions of annual production of iron ore by continents.

Due to various causes, such as interruption by war, change of political control, change in share of world markets, depletion of deposits, and others, in certain countries, there have been shiftings of the above proportions among the nations of the earth. For example, United States production was half of the world's total for the year 1947.

Geologic Range of Iron Ores—Just as iron ores have a wide geographical range, so do they have a wide geological range. They are found in the oldest rocks of the earth's crust (Pre-Cambrian), as well as in the most recent, and in rocks of most of the geologic periods between these two extremes. Even today, iron-ore minerals are being formed in some of our existing bogs, lakes, and oceans, though these have no commercial importance at present. A summary of a few of the world's important iron-ore occurrences, but including a few little-known deposits for purposes of illustration, together with their location and the geological age of the rocks in which they are found, is shown in Table 6—IV. The largest deposits presently producing are those of the Lake Superior Region, the Bihar and Orissa Provinces in India, and Quebec-Labrador, all of Pre-Cambrian Age.

Formations Associated With Iron Ores—Besides showing a wide distribution geographically and geologically, deposits of iron are found associated with almost all the principal types of rocks. That is, they occur in formations of igneous, metamorphic, and sedimentary origin. The form and shape of the deposits are, to a certain extent, governed by the types of rocks in which they occur. Deposits exist generally as irregular and lenticular shoots in schists and igneous rocks and as flat-lying, tabular beds in the sedimentary formations, as, for example, the Wabana ores of Newfoundland.

The Lake Superior ores, the Alabama ores, and the Minette ores of Luxemburg, France, and Germany, are in sedimentary rocks, the Lake Superior ores being found in jaspers, cherts, and slates, and the other two in calcareous sandstones and shales. On the other hand, the magnetic ores of Kirunavaara, Sweden, are found in syenites and quartz porphyries, while the ores of Chile occur in diorite and diabase, both occurrences being in igneous rocks. Metamorphic rocks in the form of granitic and syenitic gneisses enclose the ores of Northern New York and New Jersey.

Origin of Iron Ores—Naturally, the form and mineral content of iron ores are closely allied to their rock associations and to their mode of origin. By the latter term is meant how the iron-bearing minerals as now found were introduced or concentrated in the rocks now enclosing them. It is a curious fact that although deposits of these ores have been extensively studied and worked for years, and though they rank among the most common of metallic ores, opinions as to the mode of origin of numerous deposits are wide and varied.

This diversity of opinion is, in part, due to the ever-changing and increasing knowledge of geology and its allied sciences, principally physics and chemistry, for the natural concentration and development of iron-bearing minerals are dependent upon the physical and chemical laws of nature.

The methods of formation and natural concentration of iron ores may, for simplicity, be divided into two classes:

Table 6—III. Average Proportion of Annual Production of Iron Ore By Continents.

Continent*	Per Cent	
	Average of 1935 to 1938, Inclusive	Average of 1946 to 1951, Inclusive
Africa	3.2	3.0
Asia	3.4	2.4
Europe	63.4	43.1
North America	27.9	45.6
South America	0.9	1.9
Other Sources (Australia, Etc.)	1.2	4.0
Totals	100.0%	100.0%

* The percentages of world production of iron ore produced by several selected countries or political groups were as follows:

	Per Cent	
	Average of 1935 to 1938, Inclusive	Average of 1946 to 1951, Inclusive
United States	26.9	44.1
Great Britain	7.4	6.1
France	20.0	11.6
Germany (excluding the Soviet Zone after 1945)	5.0	3.6
U.S.S.R. and Soviet Sphere (calculated from sources believed to be reliable)	15.8	15.6

Table 6—IV. Geological Range of Iron Ore Deposits

Geological Period	Deposit	Location
Quaternary		
Recent	Taza Hematite & Limonite St. Lawrence Magnetite placers	French Morocco Quebec, New York
Tertiary		
Pliocene	* Kerch Oolitic Limonite El Tofo Magnetite	Crimea, Russia Chile, South America
Miocene	Honshu and Hokkaido gravel placers	Japanese Archipelago
Oligocene	Cheikh-Ab-Charg Hematite	Persia
Eocene	Upper Assam Clay Ironstones	India
Mesozoic		
Cretaceous	* Salzgitter Limonite & Hematite Bilbao Hematite Algerian & Moroccan Magnetite & Hematite	Germany Spain North Africa
Jurassic	* Minette Limonite & Hematite Iron Springs Magnetite	France, Germany & Luxemburg Utah
Triassic	Kashmir Calcareous Iron Ore (Hematite)	India
Paleozoic		
Permian	Damuda Sandstone (Hematite)	India
Carboniferous	* Black Band Ironstones Ohio Siderite Ores	British Isles Ohio
Devonian	* Seigerland Siderite Oriskany Limonite & Hematite	Germany Virginia
Silurian	* Clinton Hematites	Alabama
Ordovician	* Wabana Oolitic Hematites	Newfoundland
Cambrian	Residual Limonites of the Appalachians	Georgia, Virginia Alabama, Tennessee
Pre-Cambrian		
	* Minas Gerais Hematite * Krivoi Rog Hematites * Bihar, Orissa & Bastar Hematites * Labrador Hematite * Lake Superior Taconites and Jaspilites, Hematites & Magnetites * Cerro Bolivar & El Pao Hematites * Kirunavaara Magnetite	Brazil Ukraine, Russia India Quebec & Labrador Michigan, Wisconsin, Minnesota, Ontario Venezuela Sweden

* Well known, important deposits.

1. Ores concentrated through the action of mechanical forces.
2. Ores formed by chemical action and physical change.

In the first group the iron minerals, together with other detrital material (sands, etc.), were set free by the weathering and disintegration of rocks formerly containing them. This detrital material was then transported mechanically by streams or the waves of the sea and redeposited in layers or beds richer in iron content than the rocks from which the iron minerals were derived. The concentration thus brought about is made possible by the fact that the higher specific gravity of the minerals containing iron caused their deposition sooner than the other lighter sands, which were carried farther on. Thus originated most of the stream and beach placer deposits of magnetite sands, as well as some of the sedimentary hematite deposits of iron ore now mined. The deposition in these cases generally occurred in stream beds, lakes, bogs, and along the seacoast.

In the second group, two methods of concentration are involved:

- (a) Crystallization of the minerals from a silicate melt or magma.
- (b) Precipitation from solution carrying iron, by reaction within the solutions or between the

solutions and the rocks penetrated by them (replacement).

In the case of crystallization from a magma, the iron minerals, being heavier, tend to be segregated by gravity during the process of their crystallization from the cooling of the silicate melt, with the result that the final igneous rock may contain certain areas richer in iron minerals than others. It is in this manner that the magnetite deposits of Kirunavaara, Sweden, and the Adirondacks Region of New York, are supposed to have been formed.

Where precipitation from solution is involved, the iron ores are either introduced or concentrated later in the rocks in which they are now found, or formed contemporaneously with them. The solutions may either be descending ground waters or ascending hot waters and gases, the latter of which accompany or follow some period of igneous activity. These waters, in passing through the rocks, enrich them in iron materials, where iron ore is being formed, either by precipitating these minerals in the rocks, or by removing the gangue or waste minerals from a rock formation already containing some iron minerals so that the latter are thereby concentrated into an ore. An example of enrichment by introduction of minerals to a formation is the deposition of magnetite in limestone as at Cornwall, Pa. Enrichment by leaching of gangue material is exempli-

fied by ores of the Lake Superior District, where the silica of the iron formation has been removed, leaving the hematite behind to form ore.

Where the silica has not been removed by natural leaching, the iron formations still exist in their original, metamorphosed state. These unleached formations, commonly referred to as *taconite* in Minnesota and *jaspilite* or *Jasper ores* in Michigan, are now the subject of intense metallurgical research directed towards methods of economically removing the silica and concentrating the iron minerals.

Besides the formation of iron materials through the agency of ground waters, they are also precipitated from the waters of lakes, streams, and oceans, the precipitation being either chemical or through the action of minute bacteria. In these cases, the iron is generally accompanied by other sedimentary material such as sands or calcium carbonate, and the result is beds or layers of iron-rich sandstone or limestone. Thus have originated many of our sedimentary iron ore deposits such as those of Birmingham, Alabama, or the Minette ores of Luxemburg, France, and Germany.

SECTION 2

VALUATION OF ORES

The Mineralogical Make-Up of Iron Ores—As was indicated at the beginning, an ore deposit at best represents but a mixture of different minerals, only a part of which will contain the element or elements sought. All iron ores, then, may be looked upon as being made up of these two parts: One part is composed of the iron-bearing minerals, which represent definite compounds of iron; the other part includes all the other substances mixed with these compounds, and is known as the gangue of the ore. Evidently, the richness of the ore, by which term is meant the proportion by weight of iron to all other elements in the ore, depends on the composition of the iron-bearing minerals it contains and upon the amount of gangue associated with them. In working up the ores, their physical condition must also be taken into consideration. In this respect, they are subject to the widest variation, ranging from soft claylike or earthy matter to hard, compact masses. Both extremes tend to give trouble in the blast furnace. Thus, the soft, fine ores are so apt to choke up a furnace not designed to use them that they were once considered practically worthless. The successful smelting of these ores represents one of the great achievements of American furnacemen. One objection to very fine ores, and one that has not yet been overcome, is that they give rise to large amounts of flue dust, which interferes seriously with the economical utilization of the furnace gases. On the other hand, the very hard and dense ores, which enter the furnace in the form of comparatively large lumps, are difficult to reduce and require an excessive amount of fuel for two principal reasons: first, the lumps make an open charge above the zone of fusion, which generally leads to channeling; second, the large dense lumps offer so small a surface to the gases that there is very little reduction of ore by the CO in the gas. Both conditions permit the ore to descend to the fusion zone before much reduction occurs. Here the temperature is so high that reduction must be effected by direct action of carbon upon the oxide with the formation of carbon monoxide, a process that requires about twice as much fuel as the indirect reduction with CO. To improve the reducibility, the ore must be crushed to a maximum size of 2 inches and screened, as described in Section 4.

Factors in the Valuation of Ores—Omitting relative property valuations, prices of competitive ores, costs of transportation, and other considerations of a purely business nature, the chief factors that determine the value of an ore are its richness, its general chemical composition, and its physical make-up. The last factor has been sufficiently discussed above. The richness of the ore will, of course, be made the chief basis for the valuation. For this purpose a unit system is employed, a unit of iron corresponding to one per cent. But the

prices of ores, while generally following closely in proportion, do not rise and fall parallel with the number of units of iron they contain, because the gangue to be disposed of must also be considered. For example, suppose two hematite ores containing 63 per cent and 42 per cent iron are being considered. In the first, 90 per cent of the ore is iron oxide, leaving only 10 per cent as gangue to be disposed of, but the second represents only 60 per cent iron oxide with 40 per cent of its weight as gangue to be fluxed and transported. Next to richness comes the consideration of the chemical composition of the ore as a whole, for certain impurities, such as sulphur, when present in only relatively small amounts, may make a rich ore worthless. Without taking the time to consider all the possibilities in this connection, the more common impurities in ore may be classed as follows:

1. Those impurities that are never reduced in the blast furnace and so do not affect the composition of the iron are alumina, Al_2O_3 ; lime, CaO ; magnesia, MgO ; oxides of sodium (Na_2O), potassium (K_2O), and the other alkali metals; and rare-earth-metal oxides. Most of these substances, it will be observed, are strong bases, with the exception of alumina which may be either an acid or a base. Therefore, the presence of these substances in the ore may not be objectionable, for the lime and the magnesia, in particular, are valuable as fluxes. Alumina also, up to about 5 per cent, may play a useful part in regulating the blast furnace. The alkalies for the most part are driven off with the flue dust, and with modern appliances they may be recovered as a by-product, when present in sufficient amount to justify the installation of the necessary equipment. Usually, the alkalies are not worth the cost of recovery.

2. Those impurities that may be partially reduced in the blast furnace and give elements that enter the pig iron are silica, or the silicates, the sulphates, the selenates, titania, and compounds of manganese, chromium, vanadium, cobalt, columbium, zirconium, and other of the more rare elements. Of these, the silica, which term includes both the free silica and the silicates, constitutes a large part of the gangue of most ores, and as it requires an equal weight of lime or magnesia to flux it, it must be considered in fixing the value of an ore. Owing to the fact that the amount reduced in the blast furnace is subject to control to a considerable extent and that the element is readily removed during the process of purifying the pig iron, it is not considered of much importance from the standpoint of its effect on the steel produced from the iron. This attitude toward silica is just the opposite of that displayed toward the sulphur compounds. All these compounds are reduced in the furnace to sulphides, in which form the sulphur enters either the metal as ferrous or manga-

nese sulphides or the slag as calcium sulphide. Now, there is a limit to the quantity of sulphur a given slag can absorb, less than 5 per cent with the usual slags, and, naturally enough, the nearer this limit is approached, the more difficult it becomes to keep the sulphur out of pig iron. Since more than comparatively small amounts of this element are undesirable in most steels, and it can be removed in steel making only partially and with much difficulty, the importance of this element in fixing the value of an ore is evident. As to the manganese compounds, the amount of this element that enters the iron varies with the manganese content of the ore and takes place to the extent of nearly 75 per cent of the manganese charged. The per cent of this element is, therefore, considered in its relation to the iron content. An ore is available for the manufacture of the ordinary grades of pig iron when the manganese content does not exceed 2 per cent of the iron content, and for steelmaking grades it is preferably kept below 1.5 per cent iron with less than 0.75 per cent being most desirable for the bessemer process. With manganese between 2 per cent and 10 per cent, calculated on the same basis, it is necessary (except for a few grades of merchant iron) to mix the ore with others containing little of this element; but if the manganese content is 15 per cent to 35 per cent of the iron content, then the ore becomes available for the manufacture of spiegel. Since manganese is oxidized in the manufacture of steel, and its reduction consumes fuel, the reasons for these limits are self-evident. Therefore, ores containing 2 to 10 per cent manganese should be classed as **manganiferous iron ores**; those with 10 to 35 per cent manganese, as **ferruginous manganese ores**; and those containing more than 35 per cent manganese, as **manganese ores**, though 48 to 50 per cent is necessary to produce standard 80 per cent ferro-manganese, which represents the form of this element most valuable in the manufacture of steel.

3. **The impurities that are always reduced in the furnace and alloy with the iron** are all the compounds of arsenic and phosphorus. Fortunately, arsenic is not common in iron ores, but phosphorus is always present. While this element is easily removed from the metal by basic processes, none is eliminated by the acid processes, with the result that acid steels contain a higher percentage of this element than the average of the charge from which the steel is produced. This element, therefore, is the basis for the separation of all ores into the two great classes, known as **bessemer** and **basic**. This division, like that for manganese, is made on the basis of the relation of the phosphorus content to iron content of the ore. Since it is desirable to produce bessemer steel that will contain not more than 0.100 per

cent of its weight as phosphorus, a true bessemer ore would be one, the phosphorus content of which, plus the phosphorus content of the coke and limestone required to smelt and flux it, would produce a pig iron with a phosphorus content not exceeding 0.090 per cent. Allowing 10 per cent for conversion loss, such a pig iron would give a steel containing less than 0.100 per cent of its weight of phosphorus. Other elements, the compounds of which are reduced in the blast furnace, include such metals as nickel, copper, zinc, cadmium, indium, tin, etc. Metals like zinc and molybdenum, the oxides of which are volatile at temperatures slightly above or below their reduction temperature, may not all remain in the iron, and may be very objectionable in the furnace, as is zinc, which destroys the furnace linings.

Water or moisture is another factor to be considered in the valuation of ores, because it adds to the weight of ore to be handled and transported. The importance of this matter in fixing the value of an ore is seen at once when it is pointed out that many of the soft ores of the Lake Superior Region carry as much as 12 per cent of their weight as hygroscopic water, and a few as much as, or more than, 15 per cent. Some ores, such as the bog ores, may contain as much as 25 per cent water. This moisture content for any particular ore is much more nearly constant under varying weather conditions than might be expected; but in the case of different ores there is a wide variation, ranging from 0.40 per cent in some of the hard hematites to 16.80 per cent in a few of the soft red ores. Admixtures of certain soft red hematites and limonites sometimes contain in excess of 20 per cent moisture. These points are well illustrated by Table 6—V, the examples of which have been selected because they show about the same iron content when dry.

The marketing of the ores and all the metallurgical calculations involving them are based on the composition of samples dried at 100°–105° C (212°–221° F). It will be observed that drying at this temperature may not drive off water of crystallization and that in the case of the brown hematites a much higher temperature than the drying temperature is required to drive off all the combined water.

Accessibility—It is evident that the economic importance of an ore deposit depends to a great extent upon its size and its location, both geologic and geographic. Thus, an ore, that is very desirable from the standpoint of chemical composition and physical condition, may be so located as to be practically inaccessible; or, granting it can be made accessible, the amount of ore in the deposit may not justify the expense of opening it up. On the other hand, a poor ore may be so con-

Table 6—V. Compositions of Ores, Illustrating Dry and Wet Basis

ORE	STATE	% Iron, Fe	% Phos., P	% Silica SiO ₂	% Mang., Mn	% Alumina Al ₂ O ₃	% Lime, CaO	% Magnesia MgO	% Sul., S	% Ignition Loss	% Moisture H ₂ O
A. (Marquette Range)	Dry	57.36	0.137	15.62	0.08	1.26	0.68	0.33	0.007	0.03
	Natural	56.82	0.135	15.47	0.08	1.25	0.67	0.327	0.007	0.03	0.94
B. (Mesabi Range)	Dry	57.03	0.042	12.48	0.56	1.69	0.21	0.32	0.010	2.80
	Natural	52.54	0.039	11.50	0.52	1.56	0.19	0.29	0.009	2.58	7.87
C. (Mesabi Range)	Dry	57.06	0.081	7.33	1.72	1.00	0.30	0.40	0.010	2.00
	Natural	47.47	0.067	6.09	1.43	0.83	0.25	0.32	0.008	1.66	16.80

veniently located that it may be smelted in its natural condition or concentrated at a profit. A thorough discussion of this topic cannot be undertaken in the brief space allotted to this chapter. Suffice it to say, that the working of any ore body under modern conditions

presents difficult engineering problems, both in mining and in transportation, and to these may be added one or more beneficiation operations which include mixing, crushing, screening, concentrating, and agglomerating in various combinations.

SECTION 3

IRON ORE DEPOSITS OF THE UNITED STATES

The Main Producing Districts—Just as a wide diversity is found in geographical and geological distribution of the iron ores from a world viewpoint, so are like conditions found in the United States, for ores of all types and geological associations are encountered here. In two of the main producing districts the ores are found in sedimentary formations. Of these, the one is centered around Birmingham, Alabama, and the other is in the Lake Superior Region, including parts of Northern Minnesota, Wisconsin, and Michigan. The first district produced, in 1953, about 7 per cent of the iron ore of the United States, while the second, the largest iron ore region in the world, produced about 81 per cent (see Table 6—VI). A more detailed description of these areas is given below.

Besides the Alabama and Lake Superior deposits, several other regions contribute their share of the total tonnage. Among these are the magnetite ores of Pennsylvania which occur in Paleozoic limestones cut by diabase dikes of Triassic Age. They contain also a small percentage of copper in the form of chalcopyrite. Another interesting area is the Adirondack Region of New York which also produces a magnetite ore. The latter occurs as irregular masses and lenticular shoots in granites, syenites, and gneisses of Pre-Cambrian Age. Titaniferous magnetite ores of similar association and occurrence also are found here, but they are worked at only one place and there principally for recovery of titanium with iron as a by-product. The magnetite ores of Northern New Jersey and Southeastern New York are also similar to those of the Adirondack Region and occur in granitic, syenitic, and dioritic gneisses in the form of podlike shoots which

are more or less parallel to and interfoliated with the banding of the gneisses, and pitch in the same direction as the minerals of the gneisses themselves. The western occurrences of iron ore are principally magnetite, though those of Sunrise, Wyoming, are hematite. The western magnetites come principally from Iron County, Utah, their occurrences being more or less similar in that they are found in Jurassic limestone at or near the contact of the latter with monzonitic intrusives. Table 6—VI lists the iron ores produced in the United States during 1953, together with the percentage of the total each district produced:

**Table 6—VI. Iron Ore Mined
in the United States During 1953**
(Gross Tons of 2240 Lbs.)

District	Type of Ore	Tonnage Produced	Per Cent of Total
Lake Superior	Hematite	95,655,105	81.1
Southeastern U.S.A.	Hematite	7,691,745	6.5
Northeastern U.S.A.	Magnetite	5,161,813	4.4
Western U.S.A.	(Magnetite) (Hematite)	8,868,658	7.5
Undistributed (by-product ore)	Pyrites Sinter, etc.	617,448	0.5
Total		117,994,769	100.0

SECTION 4

THE BIRMINGHAM DISTRICT

LOCATION, GEOLOGY AND GENERAL DESCRIPTION

Hematite—The most important iron-ore deposit of the Birmingham District is the red hematite ore of the "Big Seam," which occurs in the Clinton formation of Silurian Age. The Clinton formation is approximately 200 feet thick and, in addition to the Big Seam, consists of shales, ferruginous sandstones and some ferruginous limestone. The Big Seam outcrops near the crest of Red Mountain and is of workable thickness (8 to 16 feet) and grade for a distance of about fifteen miles, or from Birmingham to about two miles southwest of Bessemer (Figures 6—1 and 6—2). Near the surface, the ore dips about 22° to the southeast, but, at present mining depths, levels off to about 10°. The leached, or oxidized, ore near the surface has been exhausted, and current mining operations are in the dense, hard ore, under from 1000 to 2000 feet of cover. The ore continues for an undetermined distance under the Cahaba Coal Field.

The ore is an original sedimentary deposit of both fossiliferous and oölitic hematite. The iron content of run-of-mine ore varies from approximately 34 per cent in the southwest end to about 40 per cent. The alumina (3.0 to 3.5 per cent), magnesia (0.40 to 0.80 per cent), phosphorus (0.25 to 0.35 per cent) and manganese (0.14 to 0.16 per cent) do not vary appreciably, but the silica and lime change progressively from about 10 per cent SiO₂ and 18 per cent CaO in the southwest to 21 per cent SiO₂ and 11 per cent CaO in the northeast. Thus, some of the ore is more than self-fluxing but most of it requires additional flux. The relatively low iron content of the ore is partially offset by the presence of the lime, and smelting is economically feasible because of the proximity of suitable coking coals and fluxes. The phosphorus is not high enough for producing iron to use in basic Bessemer converters, but much steel is produced by an acid-Bessemer—basic-open-hearth duplex process in which a high-phosphorus slag is formed that is prepared for use as an excellent soil conditioner as discussed in Chapter 7.

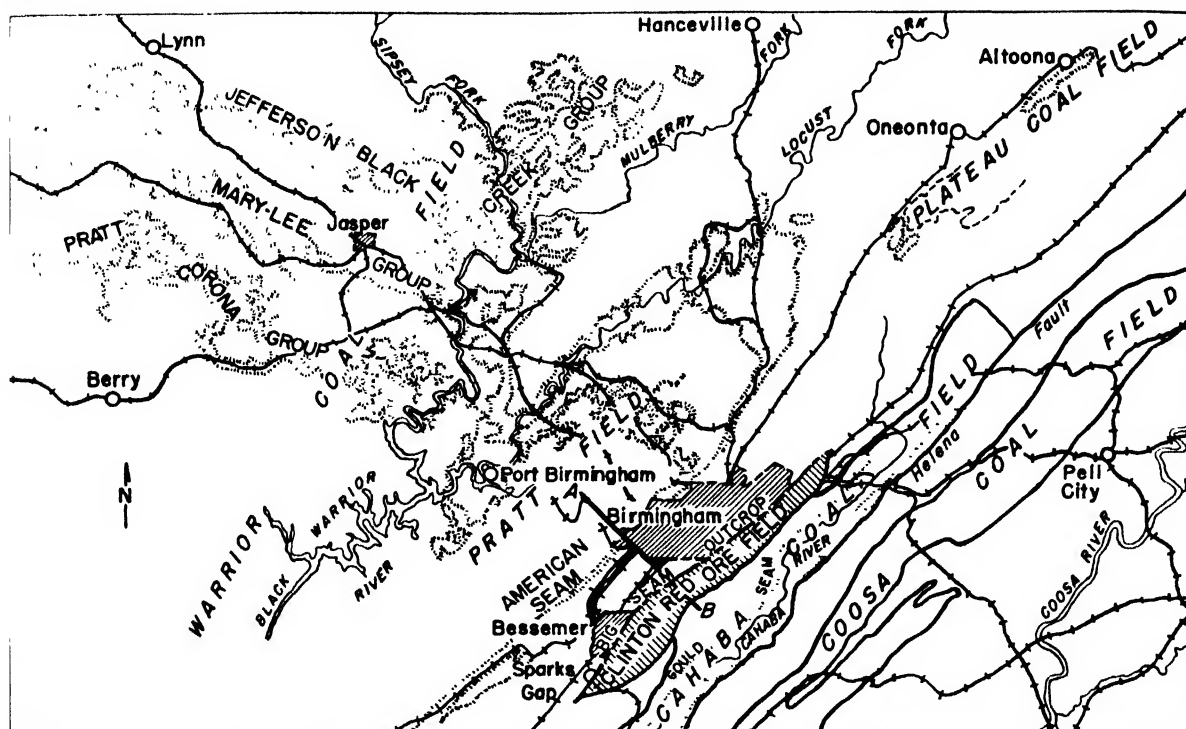


FIG. 6-1. Geological map of the Birmingham, Alabama, mineral area. (See Figure 6-2 for Section AB.)

This ore is mined underground by mechanized room-and-pillar methods generally, and in some operations by stoping methods.

In the mechanized room-and-pillar operations, panel sections approximately 450 feet in width and in varying lengths up to 2000 feet or more are mined. Triple parallel entries are turned off main haulageways at intervals of 450 feet between each set of entries. Haulage track is laid in the middle entry for the operation of mine cars. Rooms are driven from the upper entry into the ore bed at varying angles to afford favorable grades for the mining equipment. Drilling is done at the working faces with air-operated long-feed drills mounted on rubber-tired drill carriages equipped with movable arms to which the drills are attached. Positioning and moving of the arms is by hydraulic control. The broken ore from blasting is loaded with mobile loaders equipped with caterpillar treads. The gathering arms of the loading machine sweep the broken ore to a flight conveyor back of the arms from which it is discharged into rubber-tired shuttle cars maneuvering at the rear of the loader. The shuttle cars transport the ore over varying distances up to 500 feet to discharge points at the panel entry track where mine cars are filled.

In the stoping operations, the method used at mines where the ore bed is too steeply pitching for mechanized room-and-pillar methods, working levels or headings are driven at approximately 230-foot intervals, then "stopes," or rooms, are driven between the levels to recover the ore. Sufficient ore is left in place to form pillars for roof support. After the ore is shot down in the stope, cable-drawn scrapers drag it down the stope, over a steel ramp, and into mine cars. Electric locomotives haul the cars to collecting pockets above the main slope, whence it is drawn into 16-ton skips and pulled to the surface by steam- or electric-driven hoists.

Due to the low iron content of the local ores, pro-

visions have been made for the receipt and storage of foreign ores having high metallic values, for mixing with the local ores to provide richer metallic feed to the blast furnaces, thereby increasing iron production. The characteristics of these ores vary a great deal, but generally the iron content ranges from 61 to 64 per cent, phosphorus from 0.04 to 0.09 per cent, silica from 0.8 to 7.0 per cent, alumina from 1.3 to 1.6 per cent and manganese is fairly constant at 0.04 per cent. The moisture content varies from less than 1.0 to 12.0 per cent and the fines ($-\frac{1}{4}$ inch) from 30 to 70 per cent. Foreign ores are received by vessel at the Port of Mobile and transferred by rail or river transport to the Birmingham district. These ores are processed directly through the ore-conditioning plant in the same manner as the local ores, or sometimes stored for later use. The storage area is provided with a railroad-car shake-out, disposal conveyor and transfer tower, from which 23-ton diesel haulage trucks can carry the ore to assigned stockpile areas, or the ore can be diverted to a surge pile adjacent to the recovery hopper. Recovery of ore from "live" storage is by bulldozer, pushing into the recovery hopper, feeding an inclined conveyor which enters the main plant at a point where secondary crushing is performed. Recovery from "dead" or remote storage is by dragline or shovel loading into the large trucks which dump into or near the recovery hopper.

Limonite—The limonites, or brown ores, of the district are of Cretaceous Age and lie unconformably in scattered deposits on top of Cambrian and Mississippian limestones. The most important of the deposits are those southwest of Birmingham near the Jefferson-Tuscaloosa County line, and northwest of Birmingham at Russellville. The deposits are near the surface and are worked by stripping and open-pit operations. The ore as mined is mixed with about three times its volume of sand, clay and rock and is separated from

this "muck" by log washers. These ores contain no lime. The iron content varies from 40 to 55 per cent, while the phosphorus or the manganese may be from 0.10 to over 1.00 per cent. The brown ores are not sufficient to support the district, but they do make an important contribution to the iron-ore tonnage. Furthermore, with proper selection, they are useful, within certain limits, in controlling phosphorus and manganese contents in pig iron.

Beneficiation of Red Ore (Hematite)—The Tennessee Coal and Iron Division of United States Steel Corporation operates from five to seven mines in the Red Mountain Formation, the product from each mine being of different chemical composition, as indicated above. This presents a problem in burdening blast furnaces. Furthermore, definite economies in operation of the furnaces by the use of uniformly crushed and sized ore have been established in the district. Accordingly, a Central Ore Conditioning and Sintering Plant, located near No. 7 Wenonah mine, about midway of the mining operations, was put in operation in 1940 (Figure 6-3).

The run-of-mine ore (in large lumps) from the various mines is brought in 70-ton cars to this plant and dumped, two cars at a time, by a revolving car dumper. The ore passes through one of two 42-inch gyratory crushers set at 5 inches, and then through one of four 7-foot Symons cone crushers set at approximately 1 inch on the close side. All the ore from these secondary crushers, 70 per cent of which is now minus one inch, is taken by one belt over a conveyor scale, then discharged into a receiving bin in the screening building. This bin contains dividing plates and "hog-backs," which, together with vibrating feeders, portion the ore out to six 6-foot by 14-foot double-deck vibrating screens, where it is screened to three sizes: coarse ($+\frac{3}{4}$ in.), medium ($-\frac{3}{4}$ in., $+\frac{1}{4}$ in.), and fines ($-\frac{1}{4}$ in.). This screening is possible because the ore is dense and usually contains only about $1\frac{1}{2}$ per cent moisture. The product is about 55 per cent Coarse, 20 per cent Medium and 25 per cent Fines.

Each size is sampled by an automatic sampler, the sample being about 0.5 per cent of the total ore. Each sample is prepared for the laboratory in this same building by a series of crushers and cutters, and the three samples from each mine are analyzed promptly so that the results may be used in properly blending the ore.

The coarse and medium ores are weighed on conveyor scales and taken by belt conveyor to the silo storage bins, of which there are forty-eight of 800 tons capacity each, twenty-four being for coarse and twenty-four for medium. The ore from each mine run is kept separate in the silos, and after the results of analyses are received, it is blended with ore from the other mines. This is done by proportional feeders, which always carry a uniform load in terms of pounds per foot of belt, but which can be run at various speeds, so that ore from several silos may be run onto the same belt to produce one blend of uniform excess acidity (excess acidity equals silica plus alumina minus lime). The coarse and medium ores are discharged to separate belt conveyors, which transport them to railroad cars for shipment to the furnaces, the outgoing ore being automatically sampled in the same manner as the incoming ore to furnish a sample for check analysis. Shipment of ore is made on a regular schedule to conform to blast-furnace consumption, the ore being hauled by the company's own railroad over the "High Line," which traverses the valley at a fairly uniform slope from Wenonah to Fairfield and Ensley.

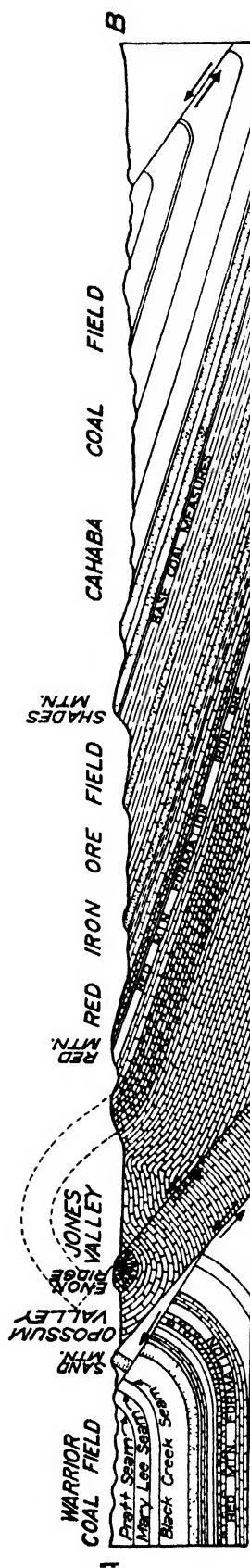


FIG. 6-2. Cross-section of Birmingham Valley iron ore fields (Section AB of Fig. 6-1).

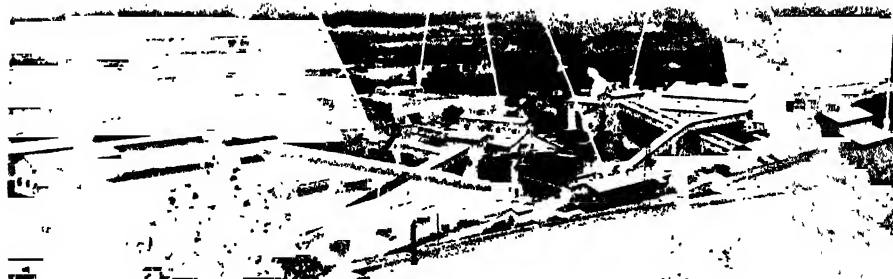


FIG. 6.—3. Central ore-conditioning and sintering plant, Tennessee Coal and Iron Division of United States Steel Corporation.

Sinter Plant—The fine ore from the screening plant is taken by belt conveyor over scales to the storage bins at the sinter plant. There are 33 of these bins, each of 350 tons ore capacity, sixteen bins being used for the ore from the screening plant, ten for flue dust (150 tons per bin), coke breeze (80 tons per bin), or other materials received from outside the plant, and seven which will receive either outside or intra-plant materials.

The fine ore and flue dust are fed from these bins by proportioning feeders to conveyor belts for transporting the proper mix to the sinter plant. Arrangement is also made so that the fine ores alone may be blended and loaded out for stocking purposes without interfering with the sinter-plant operation.

The flue dust in this district contains roughly 30 per cent iron and 15 per cent carbon, which is the chief source of fuel in the sintering operation, although coke breeze and quenching-pit cleanings are also used. The feed must be proportioned to contain the proper amount of carbon, and this varies with different ores, type of sinter desired, etc. At this plant the carbon in the mix is usually about 4.5 per cent.

The conveyor carrying the feed to the sinter plant first passes the discharge end of the sintering machines, where hot undersize sinter, that has been screened out of the sinter discharge chute by a $1\frac{1}{4}$ -inch grizzly, is charged on top of the feed. These return fines amount to about 25 per cent of the feed to the machines and are necessary in the operation to help maintain the porosity of the sintering bed. The proper amount of this circulating load is determined by experiment and may sometimes be as high as 40 per cent.

The sinter feed and the superimposed return undersized sinter are delivered to a paddle-type double-shaft pug mill wherein the revolving blades pre-mix the feed materials and then pass them on by conveyor to the distributor.

The sinter plant proper consists of three Dwight-Lloyd type continuous-grate machines, so that it is necessary to split the feed three ways. This is done by a revolving distributor which gives a full flow of the belt momentarily to each machine. This avoids the segregation that would occur if it were allowed to split itself naturally in three directions. A small receiving hopper and vibrating feeder eliminate the surges that would otherwise occur.

From the distributor, the feed goes to a revolving drum-type pug mill (one for each machine) which thoroughly mixes and aerates it. Water is also added at

this point, if necessary. A certain amount of water must be in the mix in order to maintain the bed porosity, and the proper amount is determined by experiment. About 6 to 8 per cent total moisture is required here, but in plants using ores with higher natural moisture, the figure would, of course, be considerably higher.

The mix feeds out of the pug mills continuously over 24-inch belts to swinging spouts over the machines. The mix strikes a wear plate first, slides off to a feeder plate and then falls to the advancing hearth of the sinter machine evenly, being distributed over the width of the hearth by the swinging motion of the spout.

The sinter machines are 72 inches wide. The one first installed is 89 feet, 3 inches long, with a rated capacity of 50 tons per hour, and the other two machines are each 102 feet long and rated at 60 tons per hour. The machine consists of a series of "pallets" or metal frames, 72 inches by 24 inches mounted on four wheels. Grate bars cover the bottom, and sides of steel plate 12 inches to 14 inches high are provided on the short side, but the other two sides are open so as to form a continuous hearth as the successive pallets push each other along the guide rails. The pallets are picked up from the bottom, or return side, by a large sprocket which then forces them to pass the length of the machine. As they pass under the swinging spout, the feed mix is distributed over the hearth, the larger pieces of feed rolling off by a plate to the desired depth, usually 12 to 14 inches. The pallets then pass under the ignition furnace, which consists of a burner extending the width of the machine. Wing plates of fire brick also serve to reflect the heat down to the bed. Natural gas with forced air is used here for ignition, but other fuels such as blast-furnace gas, coke-oven gas or fuel oil are also satisfactory.

The pallets then travel over the wind boxes, which make up most of the length of the machine, the ignition furnace being over the first wind box. Air is sucked through the bed and the wind boxes by a centrifugal fan of 144,000 c.f.m. capacity at 24-inch static water pressure. The fans, one for each machine, are driven by 1000-h.p. synchronous motors.

Ignition of the sintering bed occurs only at the surface, and as the pallets progress over the wind boxes, this burning, or sintering, zone moves downward through the bed until it reaches the grate bars. The speed of the machine is so regulated that, when the bed is sintered through, the pallet is at the discharge end

of the machine, where it goes over a circular guide, dropping the sinter onto the discharge chute, which contains the grizzly for removing the necessary return fines. The sinter going over the grizzly is discharged directly into railroad cars and is taken to the furnaces as scheduled along with the coarse and medium ore.

The actual sintering process lasts for a few moments only and can occur at the temperature of incipient fusion, which causes the fine particles to agglomerate. Higher temperatures cause a slagging action or positive fusion and result in "hard burned" sinter. During the

process, the carbon is burned off, the CO_2 associated with the lime is driven off, and some of the Fe_2O_3 is reduced to Fe_3O_4 . This results in an increase in iron content from 36.25 per cent in the fine ore alone to about 41 per cent in the sinter, with a yield of about 80 per cent. The action of the combustion air and gases causes a porous structure in the sinter. The relative merit of sinter and natural ore as a blast furnace charge material has been the subject of much discussion, but there is no doubt that sinter is superior to the fine ore from which it is made.

SECTION 5

WESTERN IRON ORES

Iron ore deposits in the western United States exist in Utah, Colorado, Nevada, Wyoming, New Mexico and California. Deposits of commercial significance, with respect to the manufacture of pig iron, are at present known to exist in Wyoming, Nevada, Utah, California, and New Mexico. Western iron-ore mining by United States Steel has been confined to deposits in Utah and only these Utah deposits are discussed herein.

Presently known Utah ore deposits of substantial size are located in the southwestern section of Utah some fifteen to forty miles west of Cedar City and some two hundred miles south of the Geneva and Ironton plants. The most extensive commercial deposits are in the Iron Springs Mining District (now considered as the Pinto-Iron Springs District.) There are

also extensive showings in the Bull Valley District. The general locations of southern Utah ore deposits are indicated in Figure 6-4, on which the Pinto District is identified as "Iron Mountain Deposits" and the Iron Springs District as "Granite Mountain and Three Peaks Deposits."

To date, most of the exploratory work on Utah deposits and practically all of the development work has been confined to the Pinto-Iron Springs District.

Origin of the Utah Ores—In the Iron Springs District, neither the ores nor the country rock show the features characteristic of contact metamorphic deposits. One explanation of the origin of the deposits is that during the intrusion of monzonite porphyry stocks, the hoods and overlying rock were fractured, causing a sudden

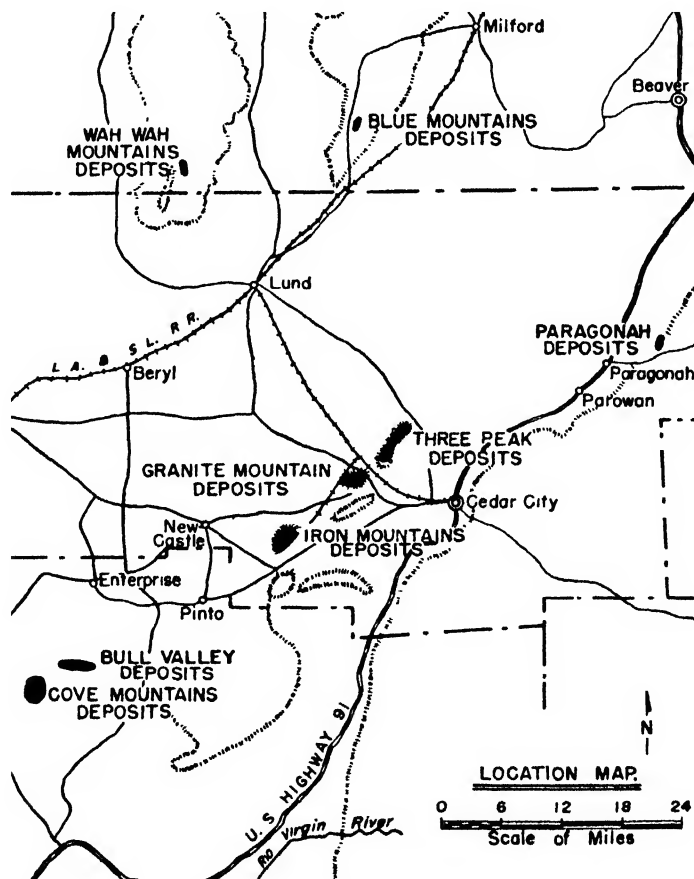


FIG. 6-4. Locations of Southern Utah iron-ore deposits.

release of pressure and liberating FeCl_2 and H_2O gas from the stocks which were still at a high temperature. These gases found ready egress to the surface along tension fissures without appreciably heating the country rock. They deposited magnetite and hematite in the fissures and replaced the contiguous fractured limestone. The period of gas emanation was brief and the hydrothermal stage probably evolved during the later magmatic history never reached this area. The deposits were formed under a cover of less than 5000 feet.

General Geology of the Utah Mining Districts—In the Pinto-Iron Springs District sediments of Jurassic, Cretaceous, and Tertiary ages have been intruded (in early Miocene) by large masses of monzonite taken to be laccoliths. Erosion following the intrusion developed mountains out of the laccoliths, with surrounding rings of outward dipping sediments on the lower slopes. Later extrusion of lavas locally covered the eroded laccoliths and sediments. The lower slopes and flats adjacent to the laccoliths are covered by the usual unconsolidated deposits of the Great Basin country. Faulting, principally of the tension type, is prevalent. The sedimentary sequence in the area includes, from the base upward, the Homestake limestone, Entrada sandstone shale, Iron Springs sandstone, and the Claron sandstone, limestone and conglomerate.

The Homestake limestone is a dark, bluish-gray limestone of a dense texture with relatively uniform characteristics throughout its entire extent, except near the laccolithic contact. The bedding of the Homestake limestone is very indefinite and is easily confused with secondary fracturing; where it is well defined, the limestone is generally thin-bedded.

The Entrada formation is about 250 feet thick and is composed of interbedded shales, sandstones and arkosic sandstones.

The Iron Springs formation is a sandstone probably more than 3000 feet thick which is composed of sandstone, shales and some limestones.

The Claron formation is a pinkish limestone overlain by pink to red shales and conglomerates.

The essential geological features of the Bull Valley District are believed to be the same as those in the Iron Springs District—a series of laccoliths with sediments dipping in every direction away from them, in the manner of a dome surrounded and overlain by flat-lying lavas, the whole being bounded on north and west by later flows of basalt. The contour of the district is rougher than that of the Iron Springs District and the evidences of volcanism are more conspicuous on account of the presence of basalt flows and cinder cones. The mineralized zones are less regular than in the Pinto-Iron Springs District and consist of fissure fillings and irregular replacements.

Occurrence of the Ores "In Situ"—In the Pinto-Iron Springs District, the iron ores occur in disconnected masses within a general area about three miles wide by twenty-three miles long, running northeast and southwest through the district. Figure 6-4 shows the outcroppings of ore (as dark irregular spots) on Iron Mountain, Granite Mountain, and Three Peaks.

The ore deposits for the most part lie at or near the contact of the monzonite laccoliths and the Homestake limestone, which occurs primarily on the eastern, western and southern slopes of foothills of the Three Peaks, Granite Mountain and Iron Mountain, but some of them, as on Iron Mountain, appear at or near the tops of the mountains. Some of the ore does not appear at the surface at all, being covered by sediments and/or monzonite detritus washed from the upper slopes.

In rare instances, replacement is noted in the shales and sandstones of the Entrada and Iron Springs formations which overlie the Homestake limestone.

Ore bodies in the district range in size from small pods of little consequence to bodies containing many millions of tons. The greater part of the ore of the district occurs as limestone replacement deposits on the contact of the monzonite. Vein-type deposits are small. The vertical range of known ore bodies is from surface

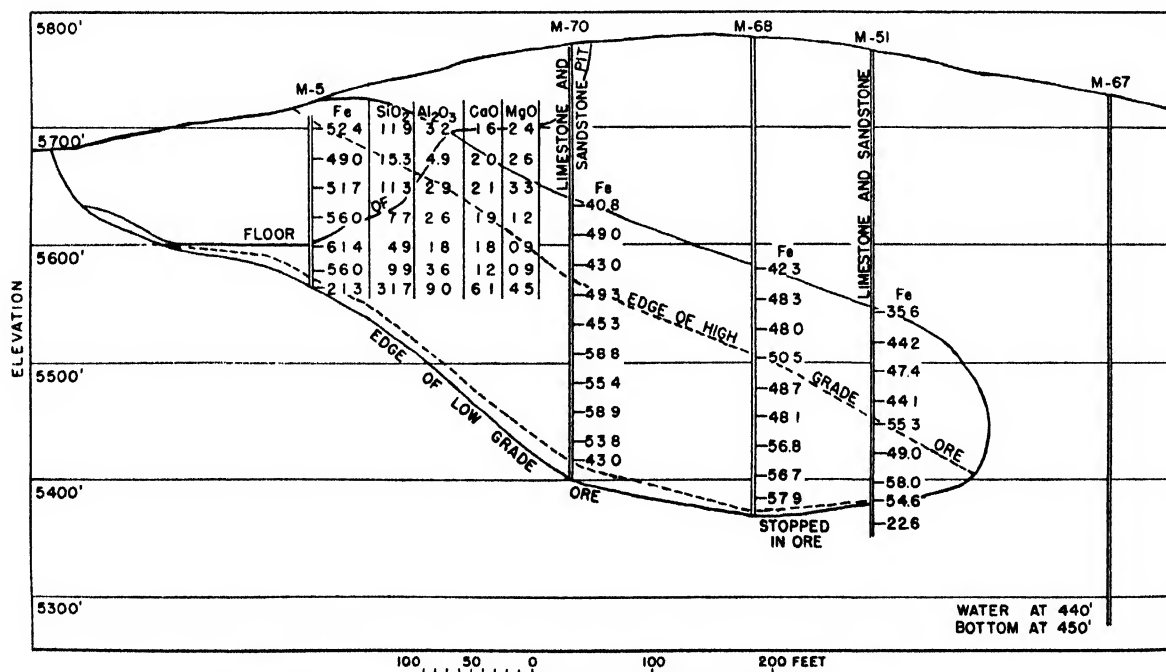


FIG. 6-5. North-south section through an open pit, Pinto-Iron Springs District. Figures show composition of samples taken at 25-foot vertical intervals.

exposures to bodies overlain by over 1300 feet of overburden. Figure 6-5 shows a north-south section through one of the pits in the Granite Mountain area.

In the Bull Valley District, the ore outcroppings are found in an elongated tract about three miles long and a mile and one-half wide. The ore deposits are discontinuous and are similar in some features to those of the Iron Springs District. The principal deposits lie within the monzonite associated with limestone fault blocks, and subordinate ones follow the main contact of the monzonite.

Ore Types—Two distinct types of ore occur in the Pinto-Iron Springs District, namely, the replacement type and the vein type.

The **replacement type**, which includes almost all of the large ore bodies, consists of hematite associated with magnetite and minor amounts of unreplaced limestone, quartz, chalcedony, calcite, barite, apatite, and pyrite. In texture, it is a fairly soft, black, fine granular mixture with occasional ribs of hard, dense material. It is usually magnetic in varying degrees. Ore bodies, where tested, contain 10 to 50 per cent magnetite. Wide variation in magnetite content within the ore body is common. The silica content of replacement ores is relatively low. Local variation of silica within the ore body is more pronounced than the variation in average contents of different ore bodies. Alumina, magnesia, and lime are fairly evenly distributed and of a grade to be ideally suited for fluxing. Sulphur is erratically distributed. It is concentrated locally in the form of pyrite and barite. However, almost all ore bodies in the district are very low in average sulphur content. Phosphorus is found in varying amounts in all ore bodies of the district. Future grading of these ores is likely to be based upon iron and phosphorus contents. The phosphorus, like the sulphur, is also distributed erratically throughout the ore body, seemingly concentrated near faults or fissures. A marked increase in the phosphorus content is noted as the northeast end of the district is approached.

Ores mined in the Iron Springs District in the period 1923 to 1953 averaged 53.99 per cent Fe (weighted average) on a natural weight basis.

Manganese, arsenic, titanium, and zinc are present only in very minute amounts. One ore body in the district, where sampled, contains about 1 per cent copper (from chalcopryite). Other bodies show copper in very minute amounts.

Ore bodies of the replacement type generally are homogeneous masses of a fairly even grade of iron. In some, the periphery is a zone of varying thickness of lower-grade iron, but in most cases the mineralized contact is sharp and distinct. Some areas exist wherein mineralization grades into the Entrada formation above and into the monzonite below. Inclusions of unreplaced limestone occur within some ore bodies, but these are not common. The ore is soft and can be easily crushed.

Vein type ore can be considered as ore from true fissure veins in the monzonite and as ore from contact veins between the monzonite and the sedimentary rocks. Contact veins are rare. They usually extend as replacement ore bodies into the limestone with characteristic changes in mineralization. Vein-type ore is predominantly magnetite with some hematite, limonite, and minor amounts of quartz, apatite, and calcite. The ore is hard, dense, crystalline, massive, and highly magnetic. It is very high in grade and exceptionally low in impurities. It has been used very successfully as charge and feed ore in open-hearth steel manufacture.

The ore in the Bull Valley District is both magnetite and hematite, as in the Pinto-Iron Springs District, but

the hematite on the lower slopes takes on a fine granular texture and a steel-blue color which is not seen in the Iron Springs District.

Development and Mining Practice—The existence of extensive iron-ore deposits in southern Utah was discovered in 1849-1850 by an exploratory party of Mormon pioneers. Soon after discovery of the ores, construction was commenced on a stone blast furnace near Cedar City. This furnace produced small quantities of iron in 1855; however, the fuel, transportation, and labor problems prevented continuation of the operation. A small furnace was erected in the Iron Springs area in 1874 and operated until 1876 on ore mined at Iron Mountain. This operation was economically unsuccessful. Except for the mining and shipment of fluxing ore for non-ferrous metal smelters, no further development work was done until the 1920's.

Near the turn of the century a considerable amount of exploratory work was done on all Southern Utah ore bodies by various individuals and firms, and in 1905 an extensive survey of the Pinto-Iron Springs Area was made by the United States Geological Survey.

Emphasis must be placed on the fact that essentially all development work on Utah ore bodies to date has been confined to the Pinto-Iron Springs District. Development of the Iron Springs District was begun in earnest in 1923 by the Columbia Steel Company (now Columbia-Geneva Steel Division of United States Steel Corporation) preparatory to operation of the Iron-ton blast furnace south of Provo, Utah (blown in May 1, 1924). A spur was built into the Iron Springs

Table 6—VII. Iron Ore Produced in Iron County
1923-1953

Year	Ore Mined (Gross Tons)	Natural Iron Content (%)
1923	2,700
1924	148,278
1925	265,322	53.00
1926	275,567	55.00
1927	221,009	53.00
1928	320,655
1929	320,960	51.68
1930	277,774	52.00
1931	183,668	53.00
1932	136,874	52.60
1933	95,129	52.20
1934	161,009	52.30
1935	161,010	52.30
1936	153,923	57.33
1937	190,908	54.50
1938	167,933	54.11
1939	262,087	53.14
1940	326,500	54.97
1941	355,006	54.47
1942	321,034	53.49
1943	922,959	53.69
1944	1,542,284	52.58
1945	1,931,749	52.84
1946	1,317,176	53.73
1947	2,823,853	53.34
1948	3,233,413	53.93
1949	2,712,390	53.66
1950	3,139,926	54.41
1951	4,726,159	55.60
1952	4,060,003	53.97
1953	4,838,983*	54.15*
Total	35,596,241	
Weighted Average		53.99

* Preliminary data.

area from Lund, Utah, by the Los Angeles and Salt Lake Railroad (Union Pacific). Loading trestles and crushing equipment were installed by the Columbia Steel Company. Tunnels were driven into the Pioche and Vermilion claims on the northeast slopes of Granite Mountain and raises were run to the surface. The ore was then mined and passed through these raises in a modified glory-hole system. In 1925, a steam-shovel open-pit operation was commenced at Desert Mound on the west of Granite Mountain. In 1935-1936, open-pit operations were moved to Iron Mountain (Pinto Mining District) and in 1937 some vein mining was commenced on the south slopes of the Three Peaks (Iron Springs Mining District) for open-hearth ore. World War II, with its expansion of the western iron and steel industry of which the construction and operation of the Geneva Plant was a part, saw a large increase in mining and development of the ore bodies, principally those on Iron Mountain.

The growth of the district can readily be seen by a study of Table 6—VII, which gives statistics of tonnages of iron ore mined in Iron County from 1923 to 1953.

At present the major portion of the ore mined in Utah is mined by the open-pit method with electric shovels. Transportation is by truck from shovel to crusher, and by railroad from crusher to the consuming plants.

Common practice is to crush ore to a top size within

the range of 1 inch to 1¼ inches. The crushed ore at the Geneva plant is blended to uniform composition by bedding with the use of stackers. It is then reclaimed from the beds, screened into two sizes, a coarse product (plus ¼-inch) and a fine product (minus ¼-inch). The fines are then sintered to form a coarser product (plus ¼-inch) which is proportioned with the original coarse fraction of the ore for use in the blast furnace.

Ore Reserves of Utah—Bureau of Mines data as to reserves of iron ore in Utah are summarized in Table 6—VIII. Of the ore reserves presently known, it is felt unofficially by various observers that approximately 200,000,000 long tons are so situated and of such composition as to be commercially mineable.

Table 6—VIII. Estimated Iron-Ore Reserves of Utah *

	Deposits	
	Pinto-Iron Springs	Bull Valley
Estimated Grade of Ore—% Fe....	45-50+	52
Estimated Reserves--Long Tons:		
Measured and indicated	100,000,000
Inferred	200,000,000
Unclassified	949,200
Totals	300,000,000	949,200

* Based on estimates by United States Bureau of Mines.

SECTION 6

THE LAKE SUPERIOR DISTRICT

Location—The great iron-mining region known as the Lake Superior District is part of the southern border of a vast area of Pre-Cambrian rocks geologically termed the Canadian Shield. The district includes several separate mining centers or ranges, as they are

called, some of which lie within the United States, and others in Canada (Figure 6—6). Among the Canadian ranges, which are found in the southern part of the Province of Ontario, are the Michipicoten, Gunflint, Atikokan, Animikie, and Kaministiquia. Those occur-

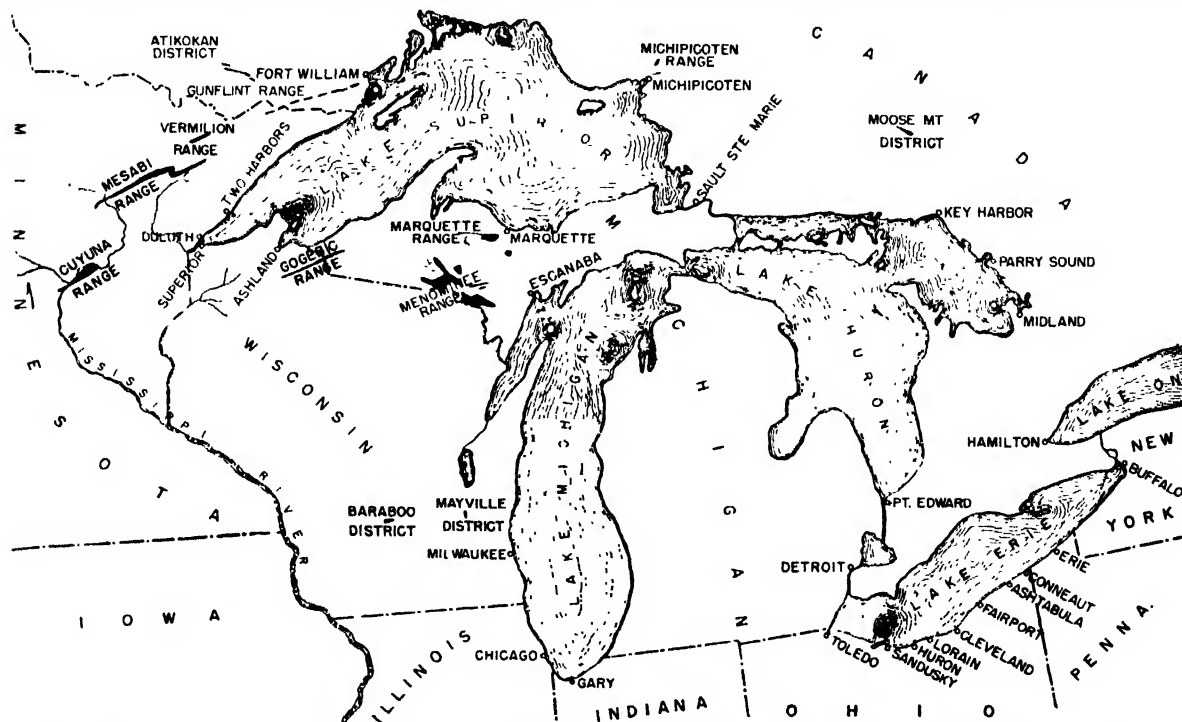


FIG. 6—6. Map showing location of iron-ore ranges, Lake Superior district, and of upper and lower lake ports.

ring in the United States are six in number, including the Cuyuna, Mesabi, and Vermilion in Northern Minnesota, the Penokee-Gogebic in Northern Wisconsin and western part of the Upper Peninsula of Michigan, and the Menominee and Marquette ranges of Northern Michigan.

Importance—The Lake Superior District, the greatest iron-mining region in the world, started production of ore in the year 1854, the first shipments coming from the Marquette Range in that year. To 1954, the Lake Superior district had shipped a total of 2,931,690,134 tons, the maximum during any one year being 99,033,748 tons in 1953.

General Geology—Geologically, this area is one of great complexity and is composed principally of rocks of Pre-Cambrian Age, numbered among which are some of the oldest known geological formations. The Lake Superior District lies within what is termed in geology a geosyncline, that is, a great basin-shaped depression or fold within which occur many minor folded areas or basins, the latter being the location of many of the iron ranges.

The rock formations found in this district include all types—igneous, metamorphic, and sedimentary—but the iron-ore deposits are found chiefly in the sedimentary strata occurring in what are called "iron formations." The latter term includes rocks comprised of banded ferruginous cherts, jaspers, slates, graywackes, and conglomerates which are components of the Archean and Huronian series of this great Pre-Cambrian mass. Besides the Pre-Cambrian rocks, some isolated areas of Cambro-Ordovician limestones and sandstones occur within the district and on the Mesabi Range local patches of Cretaceous sediments are also found. The region has been heavily glaciated and a thick mantle of glacial detritus covers a considerable portion of the area, making outcrops scarce, so that on many of the ranges the ore has been found only by dip-needle surveys and diamond drilling. A more detailed account of each of the ranges, together with some of the geological facts concerning the association, occurrence, and origin of the iron ores, follows.

The Marquette Range—This range is located in the northern part of the Upper Peninsula of Michigan. It covers an area about 30 miles long and 6 miles wide striking in an almost east-west direction, and includes from east to west the towns of Negaunee, Ishpeming, Palmer, Humboldt, Republic, and Michigamme. However, the towns of Ishpeming and Negaunee are the center of the main iron ore producing area. This range is the oldest of all the United States ranges in the Lake Superior District, the first ore being shipped in 1854, and it is still active. From its initial production to 1954, it had yielded 273,365,217 tons, while its greatest annual output was 6,540,731 tons in the year 1942. The ore on this range is extracted principally by underground-mining methods, but, in the vicinity of Palmer, some mining by open-pit methods is employed.

Hard and soft hematites comprise most of the ores mined on the Marquette Range, though some limonite and magnetic varieties are also found, splendid masses of needle limonite being common, especially in vugs in the soft hematite ores. The ores are of hard and soft varieties and are blue, red, brown, and yellow in color. The hard ores are blue, the other colors being common to the softer grades. The texture of the ores also varies, some being fine-grained or massive, some porous, some conglomeratic, and others micaceous in character. Both bessemer and non-bessemer types are mined, though high-silica ores are also produced, especially at Palmer and Michigamme.

The deposits occur in what might be termed four horizons which are included in three geological formations, all of which are Upper Huronian in age. These horizons, the geological formations in which they occur, and the ores derived from them are indicated in Table 6—IX.

Table 6—IX. Geological and Mineralogical Features of the Marquette Range.

Ore Horizon		Geological Formation	Derivative Ores
Basal Portion.		Michigamme Slates	Soft red and yellow hematites and limonites.
Top of Series	Conglomerate at base, lying at contact with Negaunee.	Goodrich Quartzite	Hard blue and red hematite and martite ores.
Bottom of Series	1. Top of formation at or slightly below contact with Goodrich Quartzite.	Negaunee Iron Formation	1. Hard, massive, and micaceous, blue and red hematite at or near contact. Soft red and yellow hematite and limonite lower.
	2. Bottom of formation.		2. Soft red, brown, and yellow hematites and limonites.

The major part of the ore produced on the Marquette Range comes from the Negaunee Iron Formation. The conglomerate ores of the Goodrich quartzite consist of a rather thin fragmental layer, while the ores in the Michigamme slates are found in comparatively small lenses or belts of iron formation surrounded by slates.

The Negaunee Iron Formation is now believed to be correlated with the Biwabik, Ironwood, Vulcan, and Deerwood series of the other ranges here described as occurring in the Lake Superior District. It consists of banded ferruginous, carbonate, and siliceous cherts, and slates, together with quartz, actinolite, magnetite schists and massive and banded jaspers. The jasper phases are the most characteristic rocks associated with the hard ore. The jaspers are found in the top portion of the Negaunee, the cherts are in both top and bottom, while the slaty and schistose phases lie between. The whole formation has a thickness well over 1,000 feet, possibly being 2,000 feet thick, though the latter figure may include beds which have been repeated by folding.

Underlying the Negaunee is a series of gray and greenish-gray graywackes, and slates, known as the Siamo slates. Directly overlying the Negaunee, unconformably, is a quartzite and arkose series called the Goodrich quartzite. At the base of this latter formation locally occurs a coarse conglomerate containing, among other materials, iron formation and ore fragments. This conglomerate is the horizon of the hard blue conglomeratic ores.

Still higher in the geological scale and overlying the Goodrich is the Bijiki schist, a quartz-magnetite-grunerite (amphibole) schist, while overlying this formation is a thick slate series (the Michigamme) containing ferruginous, graphitic, and carbonate slates with local lenses and belts of banded chert. In these cherty portions the soft brown, yellow, and red hematite and limonite ores occur.

Besides the sedimentary formations above enumerated, both acidic and basic intrusives are found on this range. These consist of dikes, sills, and stocks of diabase, diorite, and some quartz porphyry.

Structurally, the Marquette Range is very complicated, due to excessive folding and the presence of numerous igneous intrusives. Faulting is also present locally. Taken as a whole, the range represents a gently westward pitching synclinalorium, or trough, containing many minor folds within its limits, the latter of which have approximately the same pitch as the major syncline. Locally, as at Republic, where the strike and pitch are northward, these minor folds vary from the general structure of the range. Part of this folding is due to earth movements, while some is due to pressures caused by the igneous intrusives. The ore bodies occur as irregularly shaped shoots or lenses generally found in the following structural locations:

1. Occupying synclinal folds in the Michigamme Slates or Negaunee Iron Formation.
2. In the bottom of troughs formed by the intersection of the Negaunee and the diabase and diorite dikes and sills.
3. Along the contact between the Negaunee Iron Formation and Goodrich Quartzite.
4. Along the contact between the Negaunee Iron Formation and the Siamo Slates.
5. Within the Negaunee Iron Formation itself.

Where the concentration occurs in closely folded areas, the deposits are generally steeply dipping and pitching, whereas in the shallower folds and troughs, the dip and pitch is flatter; of these two types, the former is more commonly found.

The ores of the Marquette Range were formed during at least two periods of concentration, the hard ores antedating the soft ores. Their origin has been described as due to downward percolating ground waters, but evidence is at hand which suggests that the hard ores at least may be connected with the presence of hot waters and gases accompanying some of the intrusives found in the district.

The Menominee Range—This range, as a whole, includes five mining districts located in Northern Wisconsin and the southeastern part of the Upper Peninsula of Michigan. These five districts are the Iron River, Crystal Falls, Felch Mountain, and Menominee in Michigan, and the Florence District in Wisconsin, the total area of all being 54 miles in length and 15 miles in width. The Menominee and Iron River-Crystal Falls areas are the only ones now operating, the first extending from the town of Iron Mountain on the west to Waucedah on the east, while the other two are centered around the town of Iron River and Crystal Falls, respectively. The discovery of iron ore on the Menominee Range dates from 1873, but actual shipments began in 1877 and have been continuous ever since. The total shipment up to 1954 had amounted to 244,059,350 tons and the largest shipment, that for the year 1916, was 6,562,106 tons, 1926 being next with 5,946,377 tons.

For convenience, it is best to consider the Menominee District separate from those of Iron River and Crystal Falls, since their ores are of different character and are

found in two different geologic horizons. The Menominee District ores are generally blue and reddish-brown soft, porous and lumpy hematites, often high in lime and magnesia. The gangue minerals are quartz, calcite, dolomite, pyrite, and chlorite. Both bessemer and non-bessemer grades are obtained, as well as some ores of siliceous and manganiferous character. The ores of the Iron River-Crystal Falls vicinity contain hematite and the hydrated oxides of iron, such as limonite. They are blue, red, and brown in color and are also soft, porous, and lumpy. The gangue minerals here are much the same as in the Menominee District, except that gypsum is rather abundant locally. Manganiferous ores are also produced from some of the mines in this district.

All the deposits on the Menominee Range occur in sedimentary formations of Upper Huronian Age, but in two different horizons. Those of the Menominee District are in what is known as the Vulcan Iron Formation, while the ores of the Iron River-Crystal Falls, and Florence Districts are found in the basal portion of a thick slate series locally termed the Michigamme slates. The Vulcan Iron Formation is from 600 feet to 800 feet thick and consists of banded ferruginous, carbonate, and siliceous cherts and slates. It is divided into four members, which from top to bottom, are:

Formation	Average Thickness
Curry Chert or Jasper.....	150 feet
Brier Slates	300 feet
Traders Chert or Jasper.....	125 feet
Traders Talc Slate and Quartzite	75 feet

The Vulcan series is underlain by quartzite about 70 feet thick and a siliceous dolomite known as the Randville, the latter belonging to the Lower Huronian. It is overlain by the Michigamme, or Hanbury, slates, a ferruginous, siliceous and carbonaceous graywacke, slate and quartzite series which, in other parts of the range, also contain belts or lenses of banded cherts productive of iron ore. In the Menominee District the ores occur in the Traders jasper and Curry jasper members of the Vulcan Iron Formation, most of the production coming from the Traders. The footwall is generally the Talc slate where the ore is in the Traders with a hanging wall of Brier slate. Where the ore occurs in the Curry member, Brier slate may be the footwall and Hanbury slate the hanging wall. In some instances, the ore bodies lie entirely within the jasper.

The deposits of the Iron River-Crystal Falls Region lie in the basal part (the lower 400-500 feet) of the Michigamme slates. They are found in banded, ferruginous and carbonate cherts occurring as lenses in the slates. Generally, either the foot or hanging wall is a black graphitic and pyritic slate, but siliceous slates, graywackes and even iron formation sometimes form the enclosing walls. In some instances greenstone volcanics (the Hemlock series) form the footwall. Basic intrusives also have been encountered locally.

Structurally, both the Menominee and Iron River-Crystal Falls areas are complicated by drag folding and faulting, close folding being especially predominant in Iron River and Crystal Falls, while faulting is more pronounced on the Menominee. As a result, most all of the ore bodies on this range are steeply dipping (from 45° to 90°) and have a definite pitch, especially where they occupy the troughs of pitching folds. Throughout the whole range the ore deposits are generally found in irregularly shaped lenses or shoots, sometimes reaching 250 feet in width and as much as 1600 feet in length. As a rule, they are found in one of five locations, namely: along a slate-jasper contact;

along a fault or at the junction of faults; in the bottom of V-shaped dragfolds; within the chert or jasper member itself; in locations containing a combination of these factors.

The ores are supposed to have been concentrated by ground waters circulating downward along the faults or bottom of folds, or along the slate-jasper contacts, leaching the silica from the cherts, adding oxides of iron partially through oxidation and partially by replacement, besides concentrating the hematite already present.

The Gogebic Range—This range is situated in Northern Wisconsin and the western end of the Upper Peninsula of Michigan. In all, it is about 80 miles in length, strikes almost east and west, and includes the towns of Mellon and Hurley, Wisconsin, and Ironwood, Bessemer and Wakefield, Michigan. The district was first opened in 1884, and to 1954 had shipped 288,297,060 tons of ore, shipping in 1929, 7,624,085 tons.

The ores are, for the most part, soft, greasy, red hematite with minor amounts of limonite. There are also some hard blue ores as well as local areas high in manganese content.

The iron formation, locally called the Ironwood series, is Upper Huronian in age and is equivalent to the Biwabik formation of the Mesabi Range. It consists of banded ferruginous and carbonate cherts and slates having a total thickness of 600 to 900 feet, and is divided into five members known from top to bottom as:

Formation	Average Thickness
Anvil Cherts	130 feet
Pence Slates	85 feet
Norrie Cherts	125 feet
Yale Slates	140 feet
Plymouth Quartzite, Slate and Cherts....	150 feet

All of these members yield ore in one place or another, but the cherty members are the most consistently producing ones. On the western end of the range, from just east of the town of Upson westward, the character of the Ironwood formation changes and the cherts become recrystallized into finely-crystalline quartz, actinolite, magnetite rocks. This change in character of the iron

formation may be due to the influence of igneous intrusives nearby.

Underlying the Ironwood series is what is known as the Palms quartzite and quartz slates, while overlying the iron formation is the Pabst fragmental (conglomerate) and Tyler slates of Upper Huronian Age.

Over most of the range, the structure is simple, all of these sediments dipping to the north at from 60° to 70° angles. Only at the eastern and western ends is any great complexity encountered, and this is mainly in the form of drag folding, although at the eastern end intrusives also are found. However, faulting is met in most of the mines, and this occurs both transverse to and along the strike of the formation, one major strike fault in the Yale slate horizon extending for many miles along the range.

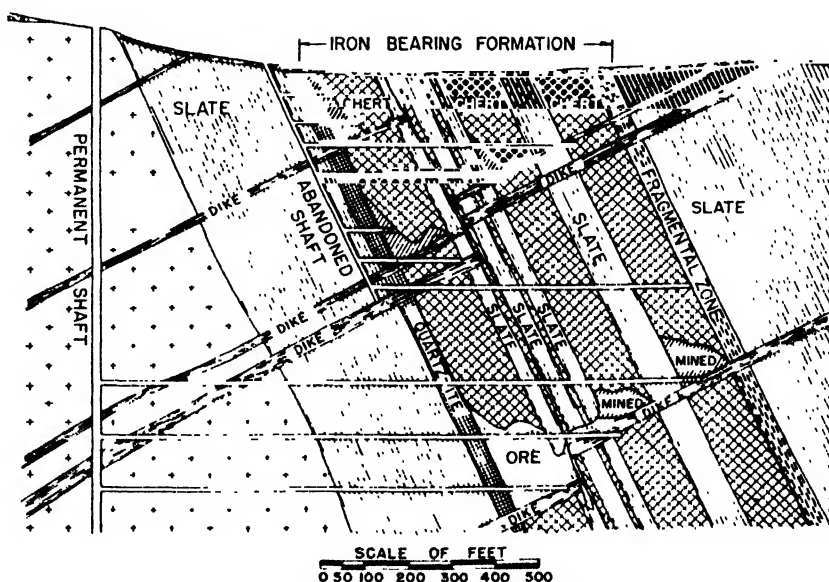
A further geologic feature of importance is the large number of basic dikes (diabase, basalt) which cut the iron formation in numerous places (Figure 6-7). These dikes are of Keweenaw Age. They are found in various thicknesses and generally have a south dip so that their intersection with the iron formation forms a V-shaped trough. These troughs are the location of most of the ore concentration, the pitch of the ore bodies being determined by the pitch of the dikes which, on the Gogebic, is generally eastward, though some westward pitching dikes also occur.

The ore bodies occur as irregular or lenticular-shaped shoots which occupy the troughs of the dike-iron formation intersection, pitching in the direction of the pitch of the dike. In some instances, however, ore has been found at the Palms quartzite (footwall) and Pabst fragmental (hanging wall) contacts with the iron formation, irrespective of dike intrusives.

The origin of these ores has been ascribed to downward-percolating ground waters causing the leaching of the silica and oxidation of the carbonates of the cherts and slates with the resultant concentration of iron oxides into ore as it is now found. These ground waters are assumed to have followed down from the surface along the V-shaped troughs on the impervious basement formed by the dikes penetrating the more or less porous cherts, leaching the silica and enriching the formations in iron oxide to form ore.

The Vermilion Range—This Minnesota range lies

FIG. 6-7. Diagram illustrating cross-section of Gogebic iron-ore range.



about 100 miles north of Duluth. It is about 30 miles in length, striking in a northeast-southwest direction, with the towns of Tower on its southwest and Ely on its northeast end. The district first started producing in 1884 and contains the oldest mine in Minnesota, the Soudan, which began operations in that year. Up to 1954, 89,530,403 tons of ore had been shipped, the largest shipment being 2,084,263 tons in 1902, while the shipment for 1953 was 1,613,155 tons.

The ores consist of blue and red hematite, largely very hard and massive, but in the eastern end of the range at Ely a great part of the ore is soft and porous. The chief gangue minerals are quartz, chlorite, pyrite, calcite, aragonite, and siderite, the carbonates being rarer than the other minerals. Native copper and copper sulphides are also found in some of the area.

The iron formation, known as the Soudan, is Archean in age. It consists of belts and disconnected lenses of massive and banded jasper, which occur in an earlier greenstone complex. The jasper is composed of recrystallized chert (now quartz), interlaminated with iron oxides in the form of both hematite and magnetite, and at times also contains amphiboles. Outcrops are scarce, since the region is more or less heavily covered with glacial detritus. The greenstone complex, known as the Ely greenstone, comprises chloritic and sericitic schists, the altered equivalents of former basic lava flows, tuffs and volcanics. Some black carbonaceous slates are also locally encountered in this formation. The Ely greenstone is also Archean in age.

The whole district is one of great geological complexity, for the two formations described above are in turn cut by numerous basic and acidic intrusives in the form of basalt, diabase and quartz porphyry dikes and sills, all of which are of Pre-Cambrian time. Besides the complications due to these igneous intrusives, the rocks are also intensely folded, often in three planes, so that the narrow belts of iron formation are now found in a steeply dipping position. Faulting is also present, being both postmineral and premineral in age.

The ores on this range lie entirely within the Soudan formation with enclosing walls of greenstone or sericitic schist. It is readily seen that with the complex geology here present, the ore bodies would probably be of irregular shape, and such is the case. They occur as irregular and podlike shoots with a steep dip and pitch, sometimes occupying the trough of folds and are generally less than 200 feet in width, though this width is exceeded where the iron formation has been repeated on itself by folding.

In this district, the deposits have resulted through the leaching of the silica in the jasper, leaving the hematite in more concentrated form. They have been further enriched through the addition of hematite from outside sources. The factors, which have been instrumental in bringing about this leaching and enrichment, have been ascribed by some to downward-circulating water following the contact between the jasper and the greenstone. However, from later studies made, there is also abundant evidence that hot ascending gases and waters, which accompanied or followed the intrusives so commonly found associated with the ores, may also have been instrumental in their concentration.

The Mesabi Range—The Mesabi Range (formerly spelled Missabe) of Minnesota comprises a belt of iron formation 120 miles in length, with a strike of about N. 70° E., and is located at distances varying from 60 to 120 miles W., N.W., and N. of the city of Duluth. It includes the towns of Grand Rapids, Coleraine, Hibbing, Chisholm, Buhl, Virginia, Eveleth, Biwabik, and Aurora, and contains the greatest open-pit mining op-

erations in the world. This range was first opened in 1892. Up to 1955, it had shipped 1,973,000,000 tons of ore, and in 1953, the year of its greatest production, over 75,000,000 tons were shipped. Several mines or open pits in this region have yielded over 1,000,000 tons annually, and one, the Hull-Rust Group, in 1942, shipped an unsurpassed total of 21,401,123 tons. During the war years, 1941-1944, inclusive, 76,267,731 tons were shipped from this one mine.

The ores mined on the Mesabi Range are mainly soft, porous blue, red, yellow and brown hematites, with minor amounts of magnetite, and some of the hydrated oxides of iron. They are of both Bessemer and non-Bessemer grades. Some siliceous ores are also produced, and others are concentrated before shipping by screening, crushing and washing. The Mesabi ores are mined principally by open-pit or quarrying methods, but in some localities where the ore is overlain by great thicknesses of glacial overburden and rock, underground-mining methods, such as top-slicing and caving systems, are employed.

The Mesabi deposits occur in what is known as the Biwabik Iron Formation, which is Upper Huronian in age. This formation consists of a series of banded ferruginous cherts, slates, graywackes, and conglomerates. The total thickness of this formation, to which has been applied the generic term "taconite," ranges from 400-750 feet. Besides the oxides of iron, the chert contains carbonates and several iron silicates. It is characterized by many different phases, some being sugary, some granular, while other types show oölitic and algal structures. The Biwabik series has been divided into four members, each of which is recognizable in the field and characterized by the ores which it produces. These divisions, from top to bottom, as well as the ores derived from them, are listed as follows:

Formation	Thickness (Feet)	Derivative Ores
Upper Slaty	0-100	"Paint Rock" Ores
Upper Cherty	175-200	Blue and Brown Ores
Lower Slaty	60-220	{ Fine Blue and Yellow Ores, Paint Rock at base
Lower Cherty	60-250	{ Yellow Ores—Top Blue Ores—Middle Brown Ores—Base

Overlying the iron formation conformably is a thick slate series, the Virginia slates, also of Upper Huronian Age, while underlying is a vitreous quartzite known as the Pokegama.

While the character of the iron formation is fairly uniform over most of the range, it shows a distinct change at its eastern end, where it has been altered by a great gabbro intrusive (the Duluth laccolith) into a series of quartz, amphibole magnetite schists and slates.

The geologic structure of the formations of the Mesabi Range is fairly simple. They are practically flat-lying, having only a shallow 10° dip to the south. As the ore has been formed chiefly through the leaching of the silica content of the cherts, and some oxidation of the carbonates and silicates of iron, the formations in the ore areas are more or less porous. In this case, the weight of the overlying rocks causes slumping in the ore areas with a down-dropping of the layers in which concentration has taken place (Figure 6-8). Local folding and faulting is encountered in some of the mines, but for the most part, the folding is shallow and gentle and not in any way as intense as on the other ranges of

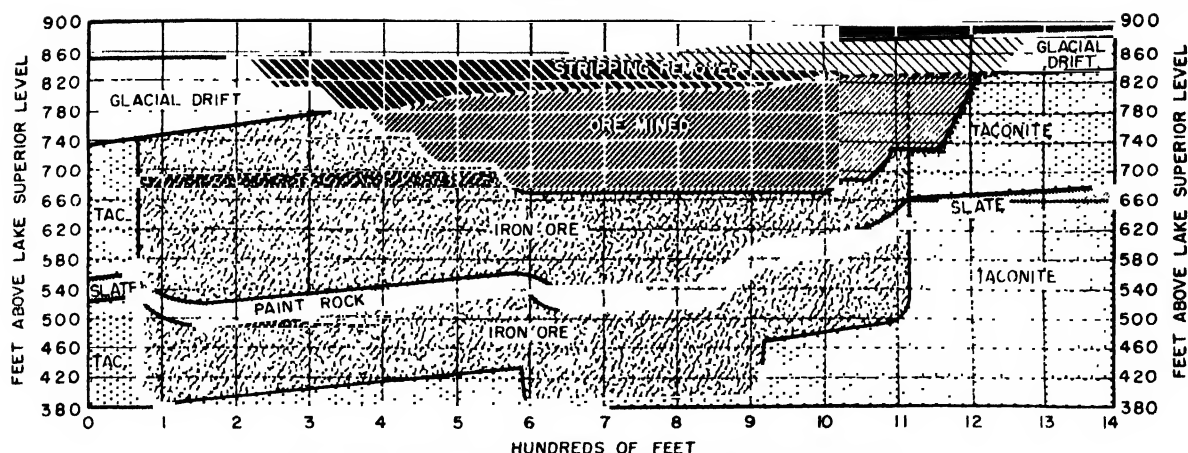


Fig. 6-8. Typical cross-section of Mesabi Range ore body.

the Lake Superior District. One large drag fold of great magnitude occurs in the vicinity of Eveleth and Virginia, causing the eastern extension of the range to be thrown to the south. This is termed "the horn," the mines of Virginia lying in its synclinal portion, while those of Eveleth are on the anticline.

The shapes of the ore bodies are of three types; namely, (1) Those occurring in shallow folds or troughs; (2) The fissure type; (3) The flat-lying or bedded ore body. The trough ore body generally owes its location to a broad shallow fold. The fissure type is smaller than the trough ore bodies and has resulted from the leaching of the iron formation and concentration of the hematite along sets of joints or fracture planes. The flat-lying ore bodies are wide and generally shallow, concentration being extensive and fairly uniform through the deposit. These three types of ore bodies often grade into one another and one may contain representatives of the other two. The concentration here, as on most of the other ranges, has probably been caused by downward-circulating ground waters.

Shipments of the depleting natural ores of the Mesabi Range will gradually be replaced by rich iron-bearing materials resulting from the beneficiation of taconite. Processes for beneficiating taconite are passing from the experimental stage into large-scale commercial operation. The term "taconite" embraces a wide variety of iron-bearing rocks (containing an average of about 30 per cent iron) found in the Lake Superior District. At the present time, commercial operations on the Mesabi Range have been restricted to the variety of taconite in which a major proportion of the iron is present as the mineral magnetite, which can be separated from the gangue by low-intensity magnetic separation. In the other types of taconite, most of the iron is present in such non-magnetic minerals as martite, hematite, iron silicates and iron carbonates. Of these types, those containing hematite and martite are the predominant iron minerals (when the minerals have a sufficiently coarse grain size) constitute a reserve of material amenable to concentration processes. The concentration of the non-magnetic minerals requires processes more complex than those required for magnetite. The taconites containing magnetite comprise the bulk of iron formation on the eastern portion of the Mesabi iron range, but very little of this kind is found in the western portion where hematite and martite are the principal iron minerals. Potential reserves of several billion tons of commercially-usable taconite are available on the Mesabi Range.

The Cuyuna Range—This is the most-recently opened of all the iron ranges in the United States portion of the Lake Superior District. It is located in Crow Wing County, Minnesota, the principal towns in the area being Deerwood, Crosby, and Brainerd. The strike of this range is in a northeast-southwest direction, more or less similar to that of the Mesabi, lying to the northeast of it. The late discovery of the deposits in this region is due, in a great degree, to the lack of outcrops, since the area is heavily covered with glacial drift. Most of the ore bodies have been disclosed by careful magnetic (dip-needle) surveys and diamond drilling. The Cuyuna was first opened in the year 1911, and to 1954 had shipped 80,264,536 tons of ore, the largest shipment being 3,714,684 tons in 1953.

The ores here are red, yellow, and brown hematite and limonite, the blue varieties being less common. As a rule, they are soft, porous, and fine grained. Some are high in phosphorus, others siliceous. Manganese occurs rather abundantly on this range, being over 15 per cent in some locations.

The ore bodies are generally irregular and lenticular in shape with a rather steep dip and pitch, often having basic intrusive rocks as a footwall. The deposits occur in two geological formations, both of which are of Upper Huronian Age. These are the Virginia Slate series and the Deerwood Iron Formation which underlies them, the latter in turn resting upon a siliceous slate and quartzite formation known as the Pokegama. The Deerwood Iron Formation is from 1000-1200 feet in thickness, and consists of banded ferruginous and manganiferous cherts, slates, graywackes, and quartzites, and can be divided into four horizons similar to those occurring in the Mesabi Range. Intrusive and extrusive rocks of both basic and acidic character are common in the region, while close folding and faulting also add to the complexity of the geological structure.

The Virginia slates yield the brown ores of what is locally termed the South Range of the Cuyuna, these being narrower than those of the North Range, which are found in the Deerwood formation and, in places, attain a thickness of 500 feet, the ores being red, blue, and brown hematite and manganiferous types. Little work has been done in the investigation of the origin of the Cuyuna ores, but from what has thus far been done, their occurrence and concentration are believed to be due to the same agencies which formed those of the Mesabi Range, viz., leaching of the silica and concentration of iron and manganese by downward circulating surface waters.

SECTION 7

MINING THE LAKE ORES

Prospecting and Exploration—Since the Lake Superior ores occur in pockets or distinct bodies and vary much as to character and location, the actual mining of the ores is preceded by much work of an exploratory character. This work includes prospecting and exploration.

Prospecting is the term generally applied to the quest for surface indications of ore, or the conditions which would warrant the expectation of finding ore in the vicinity. It includes such quest operations as geological examination, dip-needle work, shallow test-pitting, and trenching. The ore bodies of the Mesabi Range are slightly magnetic, and the Magnetometer or Hotchkiss super-dip needle sometimes are of use in outlining areas for drilling. On the Cuyuna Range, magnetic attraction, as evidenced by the dip needle, has been extensively employed as a guide to the location of ore deposits; in other localities also the dip needle has found limited application.

Drill Exploration—After the presence of an ore deposit is known or suspected, resort is generally had to exploration by means of diamond or churn drills. On the old ranges, geological conditions generally make this manner of ascertaining the exact limits of an ore deposit impracticable; so, if two or three adjacent drill holes develop considerable depths of ore, the sinking of a shaft for underground exploration, development and mining is generally considered warranted. On the Mesabi Range, however, the flat-lying and comparatively shallow characteristics of the ore formation warrant much more extensive drill explorations. On this range, then, an ore body is almost invariably followed out with the drills, and its limits are determined to the point where the complete plan of development can be worked out in advance of actual mining operations.

Methods of Mining—Both open pit and underground methods of mining are employed in the mines of the Lake Superior District. On the old ranges, where the ore bodies often extend to great depths and usually lie at angles so steeply inclined to the horizontal that the surface exposures, or outcrops, are small, underground-mining methods are employed almost without exception. On the Mesabi Range, the ore bodies are, as a rule, flat-lying with relatively large areas of outcrop, and open-pit mining is, therefore, general. Of course, there are a few deposits on this range that, on account of limited operating area, excessive depth of overburden, or for other reasons, must be mined by underground methods. But by far, the greater part of the tonnage produced from the Mesabi Range comes from open pits.

The mining of taconite poses problems because of its extreme hardness, requiring considerable additional drilling and blasting, and, because of the adverse concentration ratio, requires the handling of tremendous quantities of material.

OPEN-PIT MINING

Before deciding whether an ore body should be mined by underground methods or as an open pit, a detailed operating analysis is made of the property to determine by which method the ore can be mined most economically. Estimates are made determining the yardage of overburden, or "stripping," that must be removed to uncover the ore body, the tonnage of ore which can then be mined by power shovels and the additional

tonnage which can be "scrammed" in the pit after the limits of shovel operation have been reached. Then the cost of the entire operation, including interest charges on the necessarily large investment in stripping removal, is calculated and reduced to a final cost per ton of ore recoverable. If this figure is less than the probable cost per ton of underground mining, and if the other operating conditions are satisfactory, open-pit operation is, of course, deemed advisable. If mining by open pit methods is indicated, a decision must be made as to whether rail, truck, or belt-conveyor haulage is to be used. The planning of an open-pit mine involves the following engineering problems; first, outlining the area of ore which it will pay to strip, i.e., considering the two factors of depth of ore and thickness of overburden; second, planning the disposal of stripping which it will be necessary to remove to uncover the ore body, for this material must often be hauled considerable distances from the pits to dump grounds; third, locating the haulage systems outside the pit for the transportation of waste material and ore; fourth, designing the system of railroad tracks or roadways within the pit that will make available the maximum quantity of ore accessible by power shovels, which designing generally involves a series of switchbacks on limiting operative grades and curvature; fifth, providing for draining of the pit; and sixth, planning in advance for the mining of the ore that cannot be mined by power shovels.

Haulage Systems—Three haulage systems are utilized in open-pit mining on the Mesabi Range; i.e., railway, truck and belt conveyor. The haulage system employed is dependent upon the shape of the ore body, the quantities of stripping and ore to be moved, length of haul necessary to dispose of waste material and to deliver the ore to the beneficiation plant, and the equipment available. A rail haulage system is generally restricted to track grades of 3 per cent or less and from 12° to 16° curves, and its use is, therefore, generally limited to large, regularly shaped ore bodies. Truck haulage systems are more flexible than rail systems, and can be adapted to deep, irregularly shaped ore bodies. Road grades of 8 per cent are common in "truck pits" and steeper grades can be employed over short distances.

Removal of dirt stripping and ore from open-pit mines by belt conveyors is often the most economical and convenient means of transportation. It is well adapted to comparatively small, deep ore bodies with a limited area for the operation of power shovels and to larger open pits where conditions are suitable. Belt-conveyor systems can be either underground or surface installations. In an underground installation, an inclined belt-conveyor drift is driven from a location on the surface, where railroad facilities are available, to a carefully selected location as near as possible to the lowest elevation and center of the open-pit ore body. A shaft is driven from the top of the exposed ore in the pit to the lower end of the conveyor drift. Trucks are used for gathering the ore to a truck station or pocket at the collar of the shaft from where the ore is dumped through the shaft to the lower end of the conveyor unit. As mining and consequent deepening of the pit progresses, the collar of the shaft and truck station are lowered. In a surface installation, ore haulage is accomplished in the same manner as an underground



FIG. 6-9. Open-pit iron-ore mining.

installation, but the belt conveyors and truck station are installed on the surface of the pit profile and no shaft is necessary. As the ore is mined, the truck station is lowered and the belt conveyor is extended down to the new elevation of the pit bottom. A modification of that system is often used wherein the truck station is permanently installed at the bottom of the overburden near the edge of the ore body, with the ore hauled by trucks up to the truck station and by belt conveyor from that point to the railroad facilities on the surface.

Drilling and Blasting—The banks ahead of the power shovels, in preparation for loading, are shaken and loosened by blasting. To accomplish this, horizontal holes are sometimes drilled at the foot of the bank, but the common practice is to use vertical holes. These holes are drilled by mobile, self-contained units of the well-drilling type and are equipped with either electric, gasoline, or Diesel power. The holes are 6 to 9 inches in diameter and as deep as the height of bank desired. This type of drilling has the distinct advantage of column loading for blasting purposes, obviating the necessity of chambering the hole by springing prior to loading.

Waste Dumps—Waste dumps are located on barren ground, often at some distance from the open pits. Sites are selected for their accessibility and for quantity of material to be stored with minimum grades to secure desired results. Should the surface be nearly level, dumps for rail haul are started by building a grade fill, often twenty to thirty feet high, using trucks and power shovels. Tracks are then laid on this grade fill and, after the dump is established in this manner, the tracks can be fanned out to the right and left. When a dump track is completely filled, a spreader plow is used to level the dirt, following which the track is moved over by a machine known as a track-shifter. With use of trucks, advantage of rise in elevation of ground is usually sought to start a dump. However, if necessary, the approach to the dump can be built up by successive dumping on the roadway until the desired elevation is gained. Equipment such as described above makes it possible to dispose of vast quantities of waste material at low cost.

Scramming—This is a term applied colloquially on the Mesabi Range to the operation of recovering shallow pockets and hummocks of ore left unmined in and around the open pits following the period of power shovel mining. It is a general term inclusive of hand work, scraper work, mining with dragline excavators, bull-dozers, etc., and is applicable generally to the operation of "cleaning up" a pit after its period of real production has passed.

Drainage—Whenever an open-pit mine is deepened to the point where water interferes with operations, drainage must be supplied. This is accomplished in several ways, such as sinking shafts and driving laterals to intercept the flow of water, or by sinking wells 26 to 32 inches in diameter in which are installed deep-well pumps of sufficient capacity to dispose of the flow of water encountered. This type of pump has the advantage of keeping the motive power on surface while the pump is submerged at any desired depth. All such equipment is electrically operated. These wells or shafts are so located as to extend into the deepest areas of the mine, and at the same time so placed as to cause the least possible interference with mine operations. Frequently, a sump is constructed at the lowest elevation in the pit, and a centrifugal pump is installed on a raft to dewater the mine. All such equipment is electrically operated.

Transporting and Sampling of Ores—Aside from mer-

chantable ore, all material removed in open-pit mining is stored in waste or other dumps, as conditions may require. The physical properties of the ore are usually such that beneficiation is necessary to produce a desirable ore for furnace use, and it is loaded into steel dump cars and transported to crusher, screening or concentration plants. After beneficiation, the ore is loaded into ore cars, stored in adjacent railroad yards provided for that purpose, and from that point, transported to upper port docks by railroad. Sampling the ore for analytical and grading purposes is usually performed at the time the ore is delivered to railroad yards and built into trains for transportation to the docks. The usual method is to take one sample from units of five cars each. Unusual conditions might require closer sampling or other variations from standard practice. The usual method permits securing results of analyses in time to effect grading of the ore prior to its arrival at upper lake shipping points.

Crushing and Screening Ore—During the loading of ore in the open-pit mines of the Mesabi Range by power shovels, many large pieces of hard and chunky ore are encountered. These large masses must in some way be reduced to such a size that they will not interfere with the ore being quickly discharged from the dock pockets into boats, or be too large for successful blast-furnace operation. Very often large pieces are handled through a shovel dipper into railroad cars entirely unobserved by the workmen, and their presence is first discovered when unloading the hopper-bottom cars at the docks. Here these chunks obstruct the car openings and cause serious delays, and if they leave the cars, they are likely to obstruct the pocket opening.

In order to reduce the size of the lump material several large crushing and screening plants have been constructed. A crusher-screening plant helps to speed up shovel work by making it possible to load all ore, whether chunks or fines, and acts as a mixer to produce a more uniform grade of ore. Before the construction of crushing plants, all large chunks, which did not escape the notice of workmen, had to be reduced by blasting or with hand hammers. Shovels were delayed in picking out these chunks and laying them aside until they could be broken.

The Sherman crushing and screening plant, installed in 1950 with a daily capacity of approximately 50,000 tons of ore, illustrates a typical installation of a plant for crushing and sizing ore.

The plant is composed of two units, each unit having four receiving bins, four double-deck screens and one jaw crusher. The ore is hauled to the plant from the mines in trains of five to ten 30- and 40-cubic yard air-operated side-dump cars. The cars are pushed up a three per cent grade to the top of receiving bins which are 32 feet above the shipping-track grade. The cars are dumped, one at a time, into the receiving bins which have a capacity of 650 tons each (four bins for each crusher). The ore is moved from the bottom of the bins by 72-inch by 27-foot pan feeders which discharge the ore onto 60-inch by 12-foot double-deck vibrating screens. These screens are set in an inclined position. As the ore moves over the vibrating screens, it is separated into three sizes (+5-inch, -5-inch + $\frac{3}{4}$ -inch, and - $\frac{3}{4}$ -inch). The - $\frac{3}{4}$ -inch material goes through both screens and onto a 30-inch belt and is conveyed to a transfer where it goes onto a 36-inch belt running up to the loading pocket. The -5-inch and + $\frac{3}{4}$ -inch material goes through the upper decks of the screens and rolls off the top of the lower deck and onto a 36-inch belt that bypasses the crusher. The +5-inch material rolls over the upper decks of the

screens and goes onto a 72-inch by 70-foot pan feeder and is fed into a 48-inch by 60-inch jaw crusher and crushed to -5-inch size. The +5-inch material can also be rejected by discharging the material from the 72-inch by 70-foot pan feeder onto a 60-inch by 10-foot reversible chain belt that conveys the ore to a 54-inch by 70-foot waste-rock pan feeder. This waste rock is conveyed up and discharged directly into dump cars and hauled to the rock dump. The crushed material joins the -5-inch + $\frac{3}{4}$ -inch material on the 36-inch belt and is conveyed to a transfer where it is carried by another 36-inch belt up to the loading pocket. The two products, coarse (-5-inch + $\frac{3}{4}$ -inch) and fine (- $\frac{3}{4}$ -inch), are loaded separately from the loading pockets into standard-gauge railroad cars for shipment to the docks.

All machinery in the plant is electrically driven, and the capacity is normally limited to the crusher's capacity to handle the chunks. The crusher operators are stationed directly over the crushers with the necessary control equipment to regulate the operations of all conveyors and crusher. A signal system makes it possible for each workman on a unit to know at any time the grade of ore being run, and from what property it came.

OPEN-PIT MINING EQUIPMENT

For the operation of a mine by open-pit methods, the major items of equipment consist of electric- and diesel-powered shovels, diesel-electric locomotives, belt-conveyor systems, trucks of from 15- to 50-ton capacity, tractors equipped with blades, and churn or rotary drills. Other equipment used includes ore and stripping dump cars, flat cars, locomotive cranes, track shifters, spreader plows, automatic tie-tamping machines, tripod and jack-hammer drills, etc.

Shovels—Full-revolving, caterpillar-traction, electric-powered shovels (Figure 6—10) and walking draglines have almost supplanted other types, except in isolated locations where small diesel-powered shovels are generally used. Present shovel and dragline equipment ranges in weight from 90 to 350 tons, and from 2½ to about 10 cubic yard dipper capacity. Dirt, rock and ore banks are drilled and blasted ahead of the shovels to put the material in condition for maximum shovel output. With some of the larger equipment, ore-bank blasting is minimized.

Locomotives—In recent years, 1000- to 1600-h.p., 240-ton diesel-electric locomotives have come into use, supplanting 60- to 125-ton steam locomotives. Tractive effort required for starting loads on heavy grades and length of haul are among the problems in mind when choosing the type of power plant for haulage purposes.

Stripping Cars—Stripping cars are of steel construction, two-way, air-dump, side-hinged type, ranging from 30 to 40 cubic yard capacity, and are used for removing stripping, as well as ore that is to be hauled to beneficiation plants.

Tracks—In providing standard-gauge tracks for open-pit use, the present-day practice is to build up sections on wood ties of one rail length (39-foot standard) at points where track material is stored, making the pit installation as needed by transporting these sections on flat cars, using a locomotive crane for loading as well as laying them in position in the pit. Likewise, as it becomes necessary to move tracks from one point to another in the pit, the same method is followed. Rails used weigh 100 pounds per yard.

Belt Conveyors—Removal of dirt stripping and ore from open-pit mines by belt conveyors is often the most economical and convenient means of transportation. This is becoming increasingly true for ore as the mines

attain greater depths, requiring longer hauls with objectionable grades for locomotives or trucks. The tensile strength of rubber-covered fabric belts has increased in recent years to the extent that they are supplanting similar belts that had required reinforcing by a layer of steel cables placed between the several plies of woven fabric. A 36-inch rubber-coated fabric belt may have a capacity up to 1000 tons per hour. A popular width of belt is 36 inches, speed 500 feet per minute, with about 18° incline.

Electric Power Distribution—Electric power service and distribution for open-pit mines is either at 2400 volts or 4160 volts, 3-phase, 60 cycles. Power shovels and large pumps are operated at these voltages. The present trend is towards 4160 volts, because of the relative increase in mining area which can be economically served.

Shops—The present-day use of large-unit equipment makes imperative the installation of adequate shop facilities adjacent to the operation, such as machine, blacksmith, boiler, electrical, car, truck repair, and carpenter shops. These are provided with modern equipment necessary for the purpose intended.

Safety—In open-pit as well as underground mining, safety is a first consideration. To counteract man failure, as well as that of materials and machines, safety appliances and educational work are both necessary.

Advantages of Open-Pit Mining—It is very apparent that open-pit mining, when feasible, offers decided advantages as compared with underground methods. Probably the most evident of these is the possibility of big production; in 1942, the Hull-Rust Mine alone shipped 21,401,123 tons of ore—more than 20 per cent of the total mined in the United States during that year which, according to the U. S. Geological Survey, amounted to 104,883,184 tons. Where the overburden is light in comparison with the depth of ore, and stripping charges are not heavy, open pit mining produces low-cost ore. It accomplishes a great saving in labor; the output per man per day from the open-pit mine is many times that from the average underground mine.

BENEFICIATION OF ORES

Owing to changes in the character of available ores, most of the tonnage mined must be treated in some way in order to supply the furnaces with ore that can be



Fig. 6-10. Electric power shovel on caterpillar tracks, making a "sinking cut" and loading ore into steel dump cars for haulage from an open-pit mine to a beneficiation plant. Diesel-electric locomotive handles the dump cars. Cable carries power to shovel.

used to best advantage in burdening the furnace. Hard and lumpy ores must be crushed, fines must be agglomerated, rock lumps excluded and lean ores concentrated to a standard iron content. The quantity of beneficiated ores, exclusive of the substantial amounts beneficiated only by crushing or screening, is indicated in Table 6—X.

The flow sheet in Figure 6—11 is typical of a Mesabi

Range concentrating plant using a simple washing process. Additional methods are being used and experimented with for improvement in per cent of iron recovery and cost. These additional methods include heavy-media, jigging, fine grinding, drying, flotation and magnetic concentration. The iron-ore-mining industry in the Lake Superior District is presently involved in the development of improved methods

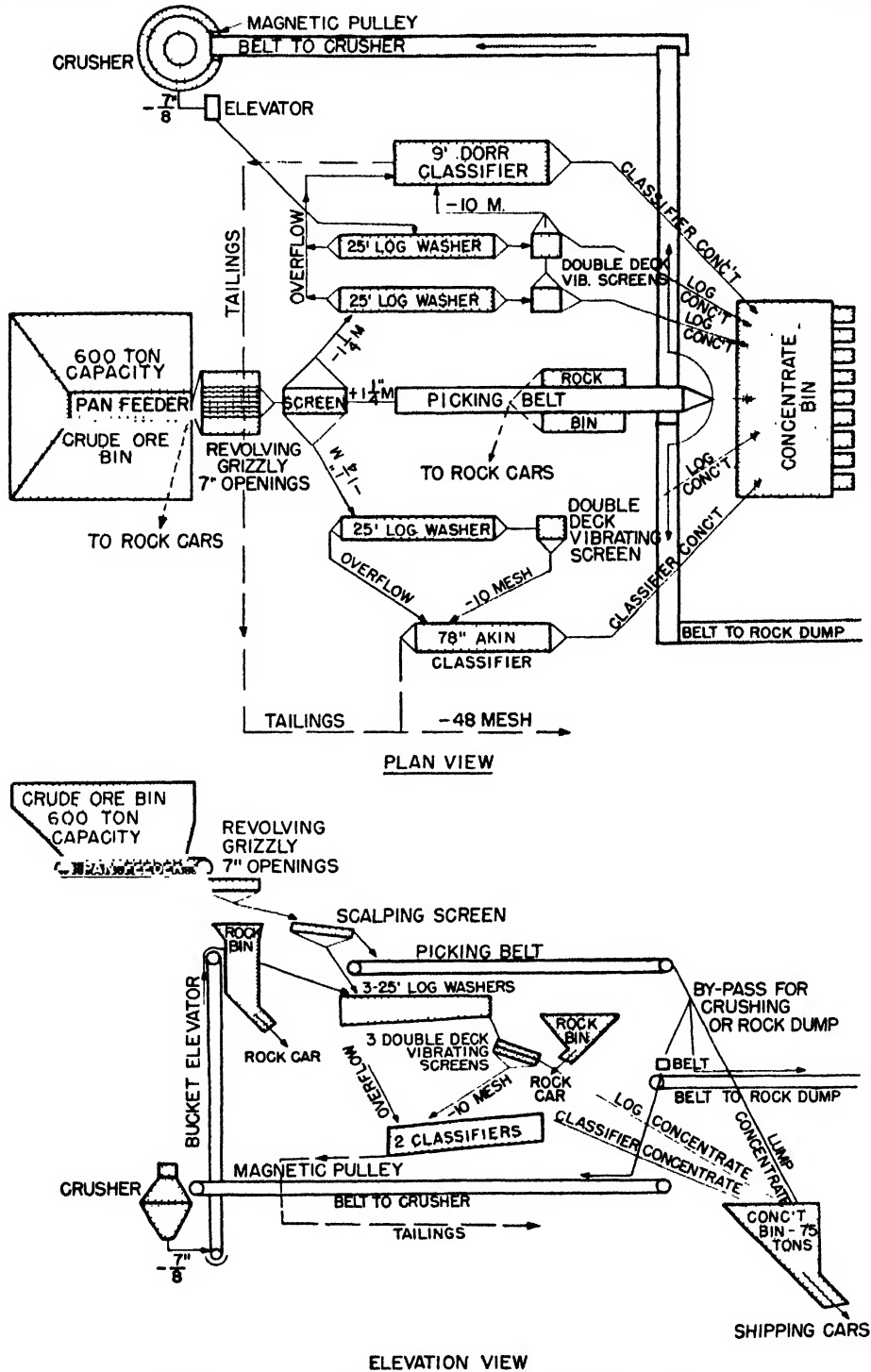


FIG. 6—11. Flow-sheet of a typical Mesabi Range concentrating plant using a simple washing process.

looking to commercial means for the utilization of the low-grade ore and so-called taconite. Taconite, or iron-bearing rock, occurs in immense quantities in the Lake Superior District, and on the Mesabi Range it can be mined or quarried, by open-pit methods.

Processing taconite to produce a usable iron ore requires grinding of the rock finer than 200-mesh to free the iron-oxide particles from the silica and other waste materials. The ore is concentrated by use of magnetic separators which remove the ore particles and reject the silica. These fine iron concentrates must then be agglomerated either by sintering, nodulizing or pelletizing to make a product for furnace use. More than three tons of taconite rock must be processed to produce one ton of usable iron ore. The investment in plant and equipment to produce iron ore from this source is very high. There were three taconite plants in operation on an experimental basis in 1955. These were located at Mountain Iron, Minnesota (Oliver Iron Mining Division, United States Steel Corporation), Aurora, Minnesota (Erie Mining Company), and Babbitt, Minnesota (Reserve Mining Company). There were two commer-

cial-size plants under construction: (1) a 3,750,000 tons per year plant built by Reserve Mining Company at Silver Bay on Lake Superior to grind, concentrate and agglomerate taconite shipped from Babbitt, Minnesota, which started production of iron-ore concentrate pellets for shipment, in October, 1955, and (2) a 7,500,000 tons per year plant scheduled to be in operation by early 1957, being built by Erie Mining Company at Aurora, Minnesota to beneficiate taconite mined in the Aurora

UNDERGROUND MINING

Importance—Underground mining in the Lake Superior Region predominates over open-pit operation on the Michigan ranges and is used exclusively on the Vermilion Range in Minnesota. It is still practiced on a limited scale on the Mesabi and Cuyuna Ranges, but is less important than formerly because long-term cost trends have favored open-pit extraction.

Shafts—In most cases, access to underground mines is obtained through vertical shafts sunk adjacent to the deposit but far enough away to avoid the effects of

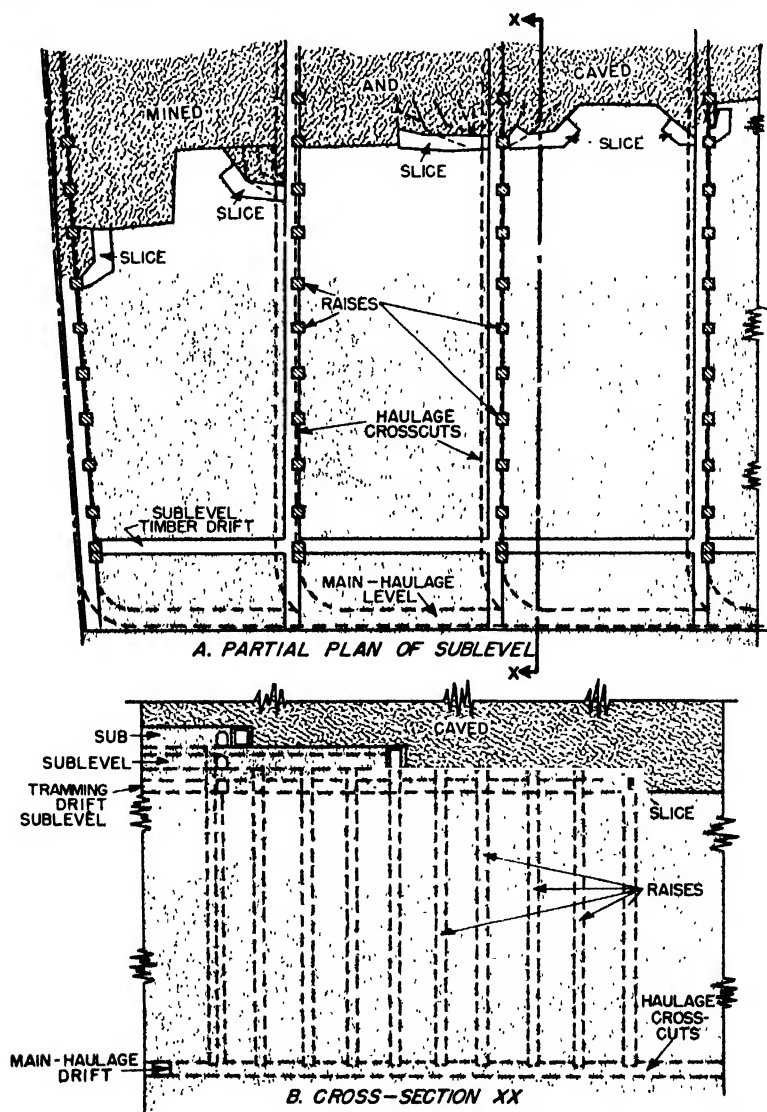


FIG. 6-12. Radial top slicing to chutes at a Mesabi Range mine; thick ore body. (From U. S. Bureau of Mines Bulletin No. 390.)

surface subsidence resulting from mining operations. Some inclined shafts are still in use and some ore is removed through adits driven in from open-pit banks. The depth of underground mines varies from as little as 300 feet in the Mesabi District to nearly 4000 feet on the Gogebic. Shafts are divided into several compartments, usually two for hoisting ore in skips (ore conveyances) in balance, one or two for men and materials, and others for ladders, air, water and power lines, and other miscellaneous facilities.

Underground Haulage—On the Mesabi Range, some of the deposits are in the form of thin, gently-dipping beds. In such a situation it is only necessary to establish one main level at or slightly below the elevation of the ore, but in most mines several levels at vertical intervals of from 100 feet to 200 feet are necessary to develop the deposit. On each level, or sometimes only on occasional levels, storage pockets are installed at the shaft for transferring ore from the underground haulage system to the hoisting facilities, and the shaft station is connected with the orebody by a system of haulage drifts. The usual method of moving ore to the shaft is by electric locomotive and tram cars operating on narrow-gauge tracks, but in several instances belt conveyors are in use.

Mining Methods—A variety of mining methods are applied in underground mines of the Lake Superior District, due mainly to the wide range in competence of the ore and enclosing formations and in the dimensions and attitude of the deposits. Cost of mining in relation to the value of the ore is also very important where there is a choice of methods.

Caving Methods—The majority of underground operations are in deposits where the ore and superincumbent rock are too weak to stand unsupported in large openings. Under such conditions, three methods—top-

slicing, sub-level caving, and block-caving—may be applied. In these methods, ore is removed in such a manner that gradual settling of the overlying material is induced, with little or no opportunity for sudden falls of large masses of ground.

In top-slicing, raises are driven vertically from the haulage level to the top of ore at intervals of from 30 to 120 feet for the purpose of transferring broken ore from the working place to the level (Figure 6-12). A timbered room or slice, about 10 feet wide and from 8 to 18 feet high, is advanced on a sub-level directly beneath the rock capping (or broken material resulting from prior operations) from a raise out to the rock walls or predetermined mining limit. Broken ore is drawn by an electrically driven scraper from the face of the slice and dumped down the raise (Figure 6-13). When a slice is completed, it is allowed to cave under the weight of the mass above, or the supporting timbers may be blasted to hasten the process. Succeeding slices are driven, each immediately adjacent to the one preceding in either parallel arrangement or in a radial pattern centering at the raise, until the area tributary to the raise is mined out. This procedure is repeated on successive sub-levels below until the bottom of the deposit is reached. This method also finds application in thin, flat-lying seams on the Mesabi, some of which are only slice-height in thickness. Top-slicing has the advantages of clean ore recovery, high percentage of extraction, and selectivity (important in orebodies highly irregular in outline or badly broken up by waste rock); but is a high-cost method because every ton of ore must be drilled and blasted and large amounts of timber are required. As a result, its use is declining.

Sub-level caving (Figure 6-14) was evolved from top-slicing in order to increase ore-breaking efficiency and decrease timber consumption. It is probably the

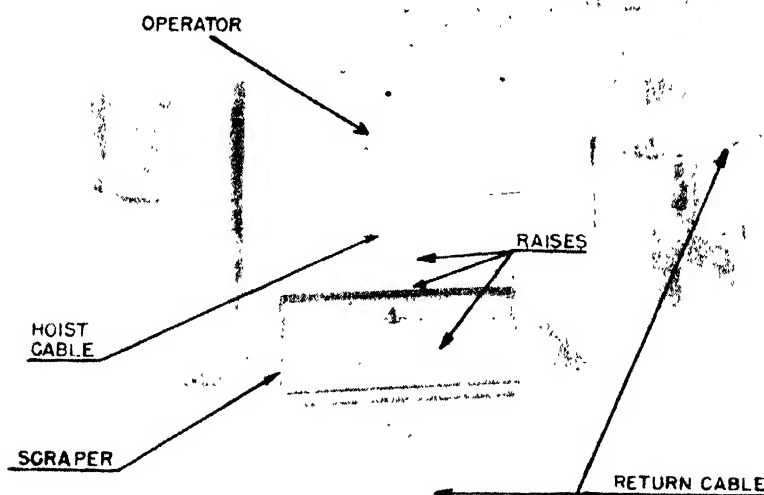
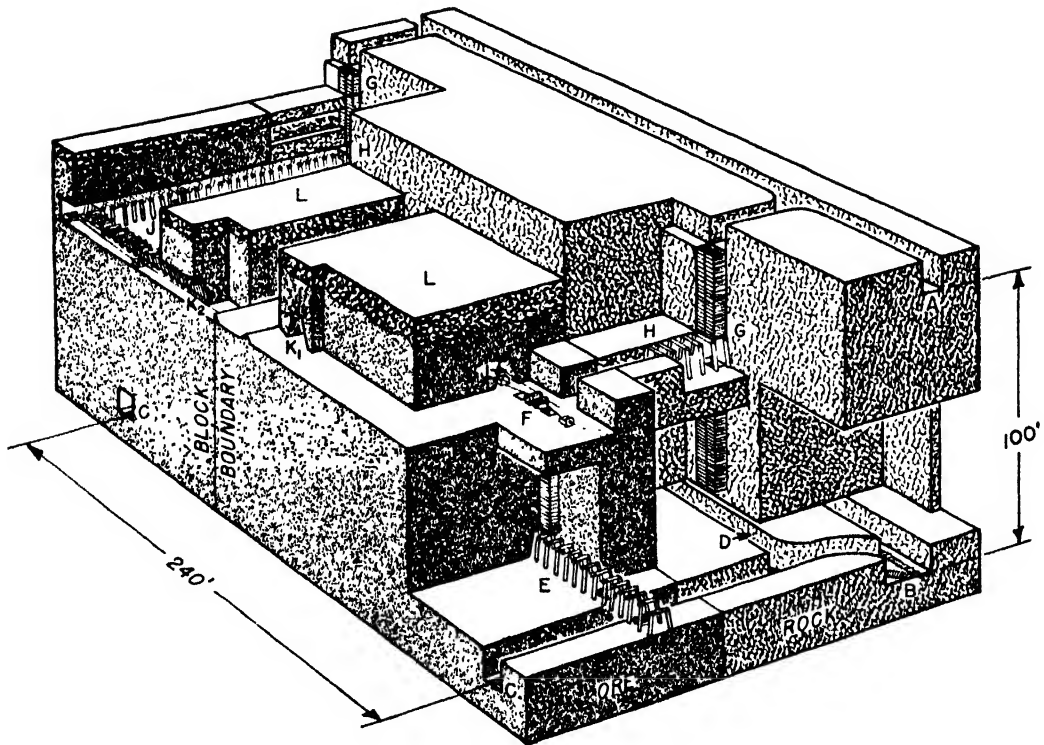


FIG. 6-13. Miner operating double-drum, electrically driven "slusher" hoist, scraping iron ore into "raises" or chutes.

SUB-LEVEL CAVINGLEGEND

- | | |
|---|--|
| A - OLD HAULAGE LEVEL USED FOR VENTILATION | G - SERVICE, VENTILATION AND MANWAY RAISES |
| B - MAIN HAULAGE LEVEL, ROCK DRIFT PARALLEL TO ORE BODY | H - SUBLEVEL SERVICE AND VENTILATION CROSSCUTS |
| C - MAIN HAULAGE LEVEL CROSSCUTS | J - SUBLEVEL TRANSFER SCRAPING DRIFTS |
| D - MAIN LEVEL SUPPLY DRIFT | K ₁ - CAVE DRIFT IN CROSS-SECTION (CROSS SLICING) |
| E - TRAM CAR LOADING DRIFT | K ₂ - CAVE DRIFT IN LONG SECTION (LONGITUDINAL SLICING) |
| F - ORE CHUTES | L - CAVED GOB |

FIG. 6-14. Diagrammatic representation of the workings in an ore body being mined by the sub-level caving method.

most extensively used mining method in the Lake Superior District, particularly on the Gogebic and Vermilion Ranges. In this method the sub-level height is increased to 20 feet or 30 feet, and the width of ore recovered in each slice-block to 20 feet or 30 feet. A timbered drift, about 10 ft. by 10 ft., is driven at the base and on the center line of each successive block. When this opening reaches the rock wall or mining limit, the ore over the back of the drift and on both sides is blasted or caved into the drift in a retreating sequence. One variation of sub-level caving utilizes blocks 50 feet high and 50 feet wide, but requires the use of two bottom scraping drifts plus a third drift above them in order to drill ore in the upper half of the block. Raises for dropping ore from the mining elevation to the haulage level are usually spaced much farther apart than in top-slicing (up to 300 feet) and transfer scraping is necessary to move ore from the slice-drift entrance to the raise (up to 150 feet).

In both sub-level caving and top-slicing, maximum production obtainable is to a considerable extent a function of the horizontal area of the deposit, since within each unit area, the ore on any sub-level must be largely exhausted before extraction can begin on the next one below.

Block caving is a method in which maximum use of the force of gravity is employed to break the ore. It is applicable mainly in clean, massive or tabular deposits which if flat-lying must be at least 50 feet thick, or if steeply dipping must be wide enough in relation to the hardness of the ore to cave or crush readily when undercut. If these prerequisites are not met, other methods will generally prove more economical since they require less preparatory work. In this system, blocks of ore up to 250 feet long and from 50 to 150 feet wide are developed by a series of parallel scraping drifts 25 to 35 feet apart, driven at or near the foot-

wall contact of the orebody or a short distance above the haulage level as the situation requires. The block is undercut at an elevation slightly above the scraping drifts by long holes drilled and blasted either from them or an auxiliary drilling drift located at the undercut elevation. Short, closely spaced **finger raises**, also called **mills**, are put up and "coned" out from the scraping drifts to enable ore caving into the void created by the undercut to be drawn off. Close control of the drawing operation is essential to obtain high recovery of ore with minimum contamination from rock capping or waste from overlying mined-out zones. Recent application of block caving in the Lake Superior District dates only from about 1947 and its current use is mainly on the Marquette Range.

Open-Stoping Methods—In a number of mines, large-sized rooms called **stopes** can be opened up without immediate danger of collapse. In steeply-dipping orebodies having widths of up to 100 feet, the stopes may be arranged so that their long axes coincide with that of the deposit, but in wider orebodies, transverse stopes, separated by vertical columns of ore called **pillars**, are employed. The pillars may be left in place permanently or may be recovered at a later date as dictated by the physical or economic situation. When the walls of the stopes are strong enough to stand unsupported for heights equivalent to the level interval, a method called **sub-level stoping** (Figure 6-15) is commonly employed. In this method one or more scraping drifts, 10 to 25 feet above the haulage level, are driven at the base of the proposed stope. At the same time, sub-level drifts at vertical intervals of 25 feet or more are driven on the stope centerline. From the scraping drifts "mills" are driven to the first sub-level and funnelled out in the shape of an inverted cone. Actual mining is started at one end of the stope by opening a slot across the full width from the first sub-level to

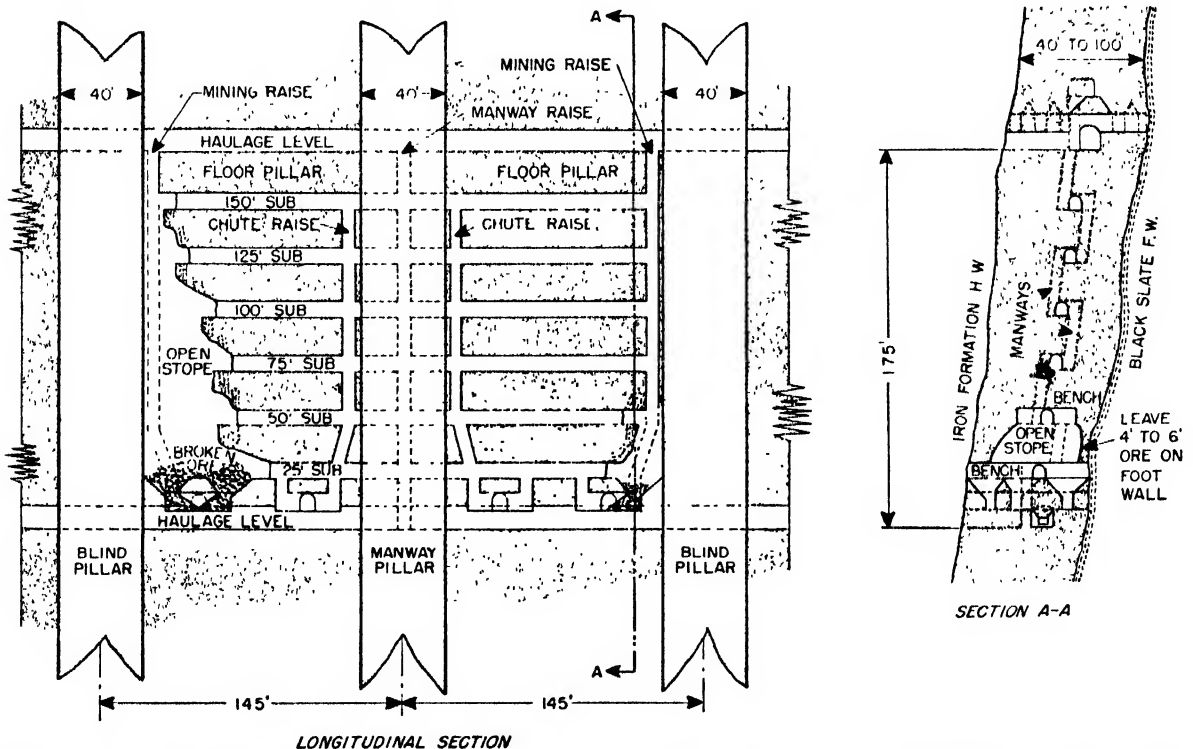


FIG. 6-15. Typical sub-level stoping using scraper system. (From A.I.M.E. Transactions, Vol. 163, "Mining Practice")

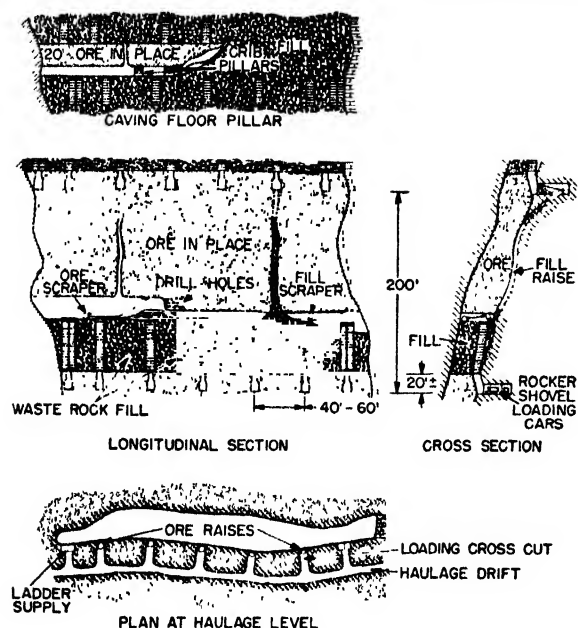


FIG. 6-16. Schematic representation of horizontal cut-and-fill stope.

within 20 or 30 feet of the main level above. Operations are then continued by retreating to the other end of the stope, ore being broken by long holes drilled radially in

vertical planes from the various sub-levels or by shorter holes drilled from benches cut across the working face. Broken ore falls to the bottom of the stope, is drawn through the mills to the scraping drift, where any required secondary breaking is accomplished, and then moved by scraper to a raise leading to the main level. Sub-level stoping is an efficient, low-cost method, and is the most widely used open-stoping system in the region. Its use is particularly widespread in the semi-hard ores of the Iron River and Crystal Falls Districts of the Menominee Range, but has been frequently used in narrow sections of soft orebodies on the Marquette, Gogebic and Cuyuna Ranges.

Several other open-stoping methods are in use where rather unique conditions exist. In each case the ores happen to be quite hard, but other factors have as much to do with the choice of mining method. At one mine, the orebodies are narrow, elongated, and steeply dipping. The rock walls have a tendency to slab off when unsupported for a height of as little as 20 feet. To circumvent this condition, a method called *cut-and-fill stoping* is used (Figure 6-16). The ore is mined between levels 200 feet apart by taking a series of horizontal cuts about 12 feet thick in an ascending sequence. At the completion of each cut waste rock is run into the stope to fill the void left by the removal of the ore. Cribbed chutes are carried upward as filling proceeds so that ore can be dumped down to the haulage level. A somewhat similar system is used at another mine where the deposits have about the same shape with somewhat more competent walls. In this method, called *shrinkage stoping*, broken ore is left in the stope as mining progresses upward. The mill or chute arrangement is

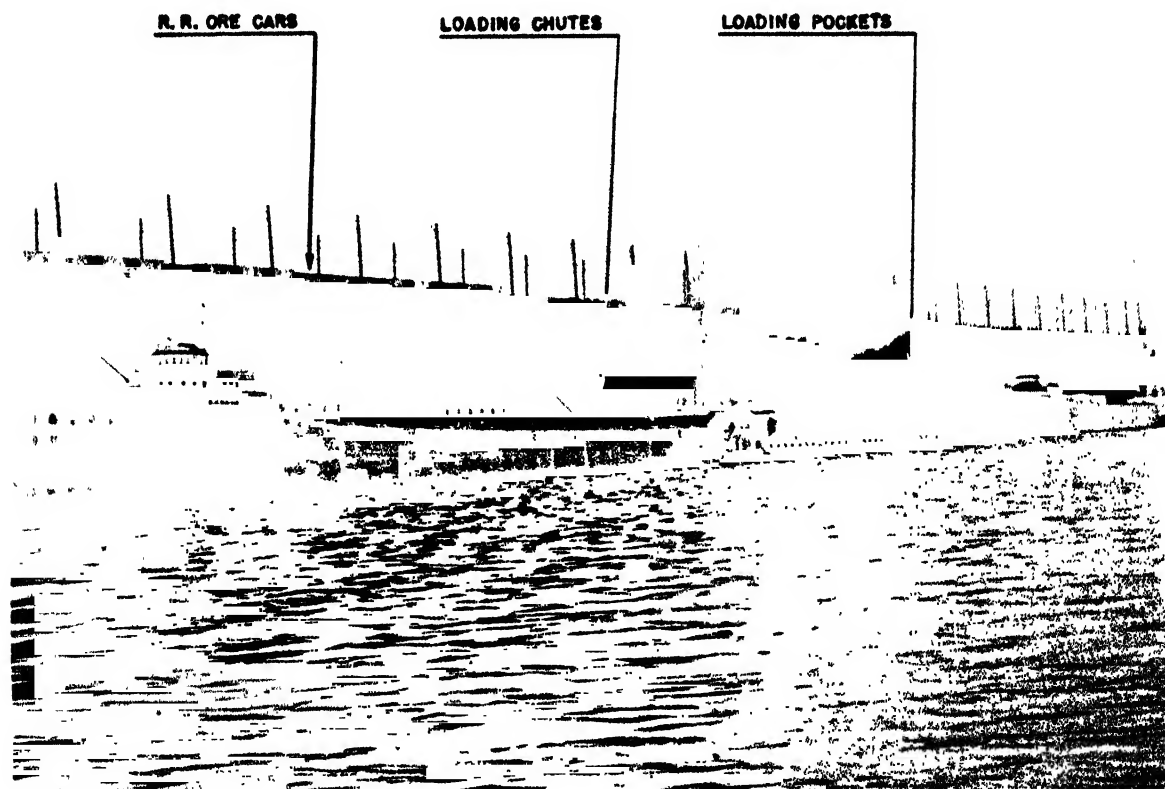


FIG. 6-17. Loading iron ore into steamer.

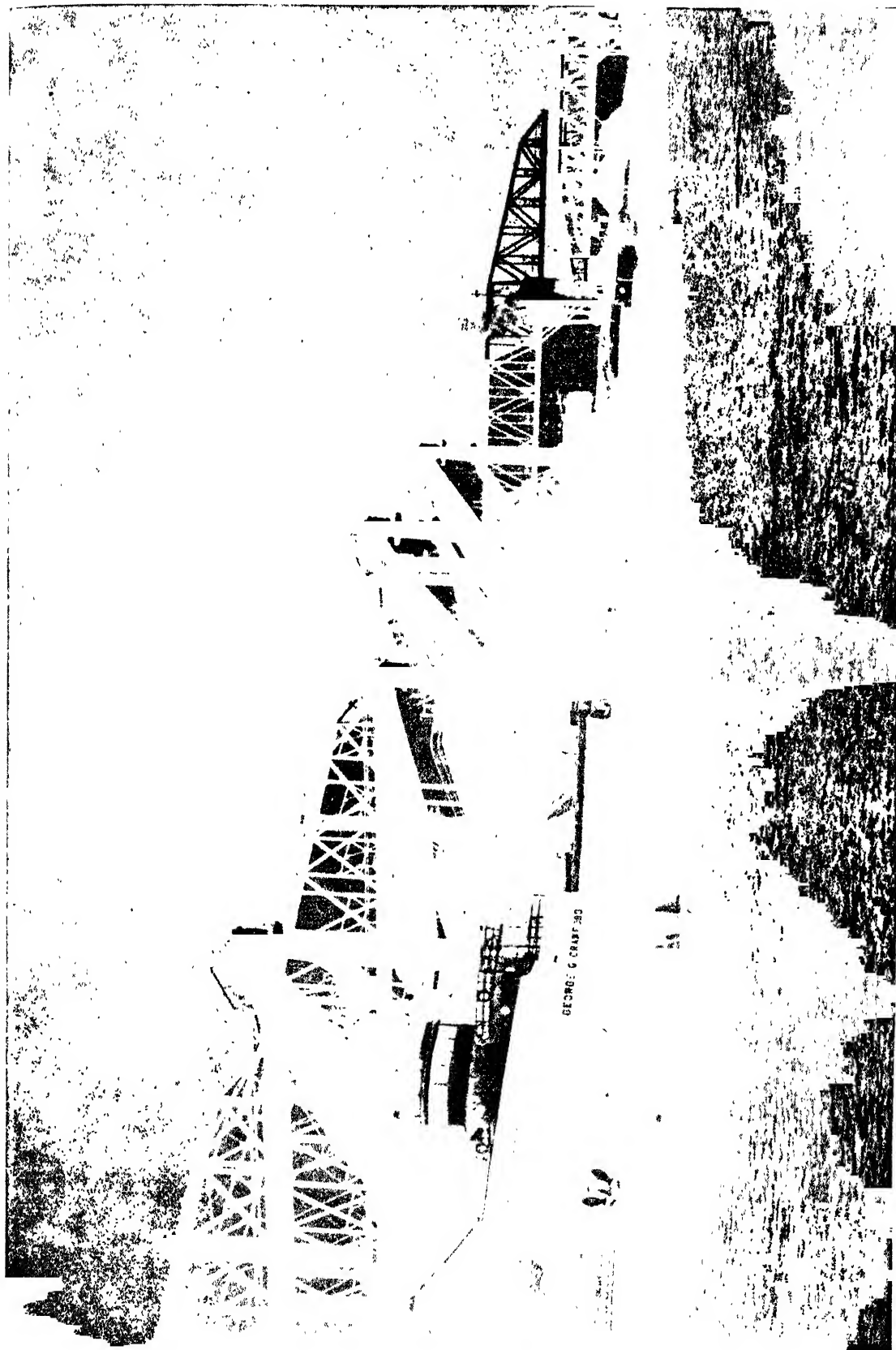


FIG. 6-18. Unloading ore from boats, showing Hulett electric unloaders in operation, with ore bridges in background.

similar to that used in block caving and sub-level stoping, only enough ore being drawn to offset the expansion in volume due to breaking. The balance is not removed until the stope is completed. At another hard-ore operation, the orebodies are a series of thick, tabular masses originally part of a flat-lying formation but now folded and faulted into numerous segments, dipping at predominantly flat angles. Levels are only 50 feet apart due to the irregularity of the deposit. The total horizontal area to be mined is too great to allow complete removal of the ore so that pillars approximately 25 feet in diameter are left unmined at fairly regular intervals for roof support. The mining system is a combination of methods, the first operation on each level being the opening of rooms about 25 feet high with pillars left as noted above (room-and-pillar stoping). If the ore does not extend to the level above, the back of the stope is mined by drilling from the top of a pile of broken ore (overhand stoping). Where the ore is continuous between levels the floor of each room is broken downward into a central raise leading to the level below (underhand stoping).

GRADING THE ORES

In the early days of iron ore mining, no grading of the ore from composition, such as prevails today, was made, and the ore was known by the name of the mine that produced it. Then, the number of mines was small, and the ore from any one mine was fairly uniform. As the production increased, however, and the field of available ore was broadened to include deposits previously regarded as unprofitable, it became necessary, in order to simplify shipping, to grade ores according to their composition and, further, to mix ores differing in composition to produce certain grades. Finally, it became quite common for one mine to ship several different grades, and for the ore from several mines to be grouped under one name. These conditions brought about a necessity for knowing the exact composition of the various ores, and whether or not, in the case of mixed ore, each cargo was of the grade guaranteed. This grading is done by sampling the ore in the cars as fast as they are loaded at the mine or beneficiation plant in lots not exceeding five, and making a rapid but accurate analysis of the determining elements. From the results of this analysis, the class or grade of the ore is fixed, and its allotment into a certain group can be made. This is the work of the grader, who, from the analysis of the cars as submitted to him, makes a theoretical cargo in which the contents in silica, iron, phosphorus and possibly manganese, the determining factors in the value of an ore for its particular purpose, must fall within certain predetermined limits. It is very difficult to grade to more than two components, and usually the grading is done to iron and silica, or to iron and phosphorus, unless the deposits have been separated by nature into bodies of basic and bessemer grades, when the grading may include iron, phosphorus, and silica.

TRANSPORTING THE ORES

The lake ores now supply all the furnaces in Western New York, Western Pennsylvania, Ohio, Illinois, and Indiana, as well as those in the ore producing states of Michigan, Wisconsin, and Minnesota. In order to reach these markets, the ores must be transported for distances varying from 300 to more than 1000 miles, depending upon the locations of both the mine and the furnaces. The cost of transporting this ore by rail alone would be a serious handicap to some furnaces, but fortunately the chain of Great Lakes affords a cheap mode of transportation for the greater part of the long

as well as the short distances. Nearly all the ore mined in the ranges, then, goes first by rail to a harbor on Lake Michigan or Lake Superior, where it is loaded on ore-carrying boats that carry it either down Lake Michigan to Chicago or Gary or through Lake Huron and Lake Erie to ports farther south. For most of the ore, even these lower lake ports are not ultimate destinations, and another haul by rail is required to place it at the furnaces.

Now, to return to the grading of the ore and what was there referred to as a theoretical cargo or boat load which may weigh from 3000 to 21,000 tons, depending upon the size of the boat. When the cars containing this theoretical cargo reach the dock at the shipping port, they are unloaded into the dock pockets, one on top of the other, three to six cars to a pocket, in such order as to mix the ore as much as possible. The ore is then allowed to flow from these pockets into the hatches of the steamer (Figure 6—17), thus, again mixing the ore. Then the boat proceeds to her destination, where the ore is unloaded by electrically or otherwise operated grabs (Figure 6—18), which process of unloading still further tends to mix the ore and make it uniform. All this mixing of the ore is not to be thought of as merely incidental to the operations, but as a necessary course of procedure, for uniformity in the ore is a very important requirement in the successful operation of the blast furnaces. If the ore is unloaded at the works located on the lakes, it is sampled for analysis during the unloading by an elaborate system and is dumped upon its appropriate stock pile; if for inland works, such as Pittsburgh, it is placed in cars and shipped to the works, where it is thoroughly sampled. This sampling and analysis is common to all consumers, in fact, and is employed as a check on the sampling, analysis, grading, and mixing at the mines, and on all the operations of handling involved in delivering each grade of ore to the furnaces. The cars are then unloaded upon a stock pile from which the ore can be used as needed, or directly into the furnace bins, if the ore is needed for immediate use.

Mining and Grading in Winter—In winter the procedure as outlined above has to be changed somewhat. During a part of November, and all of the winter months of December, January, February, and March, the ore cannot be transported over the lakes because of the ice. On this account, operations in the open pit mines of the Mesabi District are suspended in winter; but in all the underground mines, both of the old ranges and the Mesabi, mining is continuous throughout the year, and the ore mined during the nonshipping season must be stock-piled. As this ore is removed, it is carefully sampled, and average samples are analyzed daily. The results of these analyses, supplemented by those made in the work of exploration that is constantly carried on in advance of the mining, make it possible to calculate the average composition of each stock pile at the beginning of the shipping season in the spring. This stock, therefore, may be combined, if necessary, with the ore direct from the mines to make up cargoes of definite and known composition.

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Chapter 7

FLUXES AND SLAGS

SECTION 1

FLUXES

Function of Fluxes—Any metallurgical operation in which metal is separated by fusion from the impurities with which it may be chemically combined or physically mixed (as in ores) is called **smelting**. Since in iron smelting both these conditions with respect to impurities are always present, the production of crude iron involves two processes: (1) the reduction of the metal from its compounds and (2) its separation from the mechanical mixture. Many of the impurities associated with iron ores are of a highly refractory nature, that is, they are difficult to melt. If they should remain unfused, they would retard the smelting operation and interfere with the separation of metal and gangue. *To render such substances more easily fusible is the primary function of a flux.* Some elements, being reduced almost simultaneously with the iron, dissolve in it, or even combine chemically with it. Some other compounds, already combined with the metal in the raw materials, refuse to be separated from it unless there be present a substance for which they have a greater chemical affinity under the prevailing conditions. *To furnish a substance with which these elements or compounds may combine in preference to the metal is the secondary function of a flux.*

Chemistry of Fluxes—The selection of the proper flux for a given process is chiefly a chemical problem requiring a knowledge of the composition and properties of all the materials entering the process. With this knowledge at hand, the selection will be governed by the well-established physical and chemical laws which apply at smelting temperatures. These laws are not unlike those which govern chemical reactions at ordinary temperatures. Of most importance are the laws concerning the formation of salts from the interaction of acids and bases. Practically all of the slag-forming compounds that enter into a smelting or refining process may be classed as either "acids" or "bases" by virtue of the fact that they will react with each other to form compounds which are analogous to the salts formed in reactions taking place in water solutions. In like manner, stronger or more active "acids" and "bases" will replace weaker or less active ones in slag compounds. The substances which are conventionally considered to be "bases" at the high temperatures encountered in smelting and refining are those which are compounds of the elements forming basic compounds in ordinary chemical reactions in water solutions, such as calcium, magnesium, sodium, etc.; while the most important acid impurities are compounds of silicon and phosphorus. These latter elements normally form acids in water solutions. In addition to these are compounds which are analogous to the amphoteric compounds or elements which are capable of acting as acids or bases depending on the conditions imposed. Since one of the functions of

a flux is to neutralize or form slag with unwanted impurities, it will naturally follow that to remove "basic" impurities, an "acid" flux will be required and to remove "acid" components, a "base" will be used as the flux. It is fortunate that, generally speaking, the slag compounds formed by reactions between acid and basic materials have lower melting points than the reacting compounds, so that the primary function of a flux, rendering impurities more fusible, is simultaneously fulfilled, although there are occasions where a "neutral" material may be used to obtain slag fluidity. In most ores the impurities will be both acid and basic, with the acid materials usually predominating. In certain Southern ores, however, the acids and bases are so well balanced, or can be made so by mixing, that the ores are classed as **self-fluxing**. A brief discussion of the more important fluxes used in the iron and steel industry follows.

Acid Fluxes—Silica (SiO_2) is the only substance that may be classed as a strictly acid flux. For this purpose it is available as sand, gravel and quartz in large quantities and in a sufficiently pure state. In acid-steelmaking processes, silica seldom is added as a flux, for the lining of the furnace supplies what may be required. On occasion, silica may be used in basic processes where excess lime has been charged, or where the raw materials are too low in silicon. Acid-Bessemer slags may be charged in the blast furnace for their beneficial fluxing or scouring effect on accumulations on the furnace bosh, as well as, incidentally, to recover the iron and manganese they contain. Gravel is used frequently for the same purpose. Siliceous iron-bearing materials are charged in making blast-furnace grades of ferrosilicon, but the function of silica in this instance is to provide a source of silicon rather than to act as a flux.

Basic Fluxes—While in acid processes iron and manganese oxides act as bases, they are not normally added for this purpose and cannot strictly be classed as fluxes. The chief natural basic fluxes are limestone, composed principally of calcium carbonate, CaCO_3 , and dolomite, composed principally of calcium-magnesium carbonate, $(\text{Ca,Mg})\text{CO}_3$. Either dolomite or limestone may be used as a blast-furnace flux, but the former is not so satisfactory as limestone in cases where it is desired to minimize the volume of slag produced, or where large amounts of sulphur must be removed by the slag. Availability and cost are important factors when choosing between them. A high-magnesia slag is not desirable for cement manufacture, but for ballast or concrete aggregate produces a much harder product which is less subject to slaking. Representative compositions of limestone and dolomite are given in Table 7—I, together with typical compositions for calcined or "burnt" limestone and dolomite. Lime is usually added as calcined or

Table 7—I. Representative Compositions of Limestone, Burnt Lime, Dolomite and Burnt Dolomite (Natural State Determinations).

Ingredient	Limestone	Burnt Lime	Washed Dolomite	Burnt Dolomite
CaCO ₃	95.06	54.74
MgCO ₃	0.54	0.76	39.61
Fe ₂ O ₃	0.70	0.93	0.43	1.57
Al ₂ O ₃	1.73	2.55	0.74	1.53
SiO ₂	0.049	0.07	0.026	0.037
Sulphur.....	0.020	0.03	0.006	0.009
Phosphorus.....	1.70	4.00
H ₂ O.....	81.36	56.35
CaO.....	38.65
MgO.....	14.00	1.60
Loss on Ignition.....

"burnt" lime in the basic open-hearth process where more than the amount furnished by the limestone charged is found to be needed at a later stage of the heat. *Dolomite is not used in the basic open-hearth process except as a refractory.*

The effectiveness of a natural basic flux is reduced by the chemically-acid impurities it contains, since these also must be neutralized by some of the chemically-basic compounds in the flux. The effectiveness of a basic flux is expressed in terms of "available base," by which is meant the amount of basic substance that remains in the raw flux after its own acid contaminants have been satisfied.

Alumina—Although alumina is seldom employed as a flux, it deserves mention at this point because it is present in a large number of raw materials and is therefore unavoidable. In slags it may function as an acid or as a base, depending on the conditions imposed. For instance, in highly siliceous slags it may form aluminum silicates, while in the presence of a large excess of a strong base such as lime, it may form calcium aluminates.

Neutral Fluxes—For the purpose of making slags more fusible without changing their acidity, basicity, or oxidizing properties, neutral substances having low fusion points may be added. For this purpose, fluorspar is the most commonly used substance. A typical composition of fluorspar, in which calcium fluoride is the active ingredient, is given in Table 7—II.

Table 7—II. Typical Composition of Fluorspar (Dry Basis)

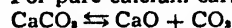
Ingredient	Per Cent
CaF ₂	81.0
SiO ₂	4.75
Al ₂ O ₃ , Fe ₂ O ₃	1.00
S	1.00
CaCO ₃	Remainder

Sources of Fluxing Materials—As mentioned earlier, sources of silica for fluxing purposes are sand, gravel, quartz, and, in some cases, such material as used brick from silica-brick lined vessels or ladles. Acid-Bessemer and acid open-hearth slags or siliceous ores may be used in special instances. Limestone of fluxing quality is distributed widely and underlies most of the area drained by the Mississippi and Ohio Rivers. The larger portion of the limestone for the Pittsburgh steelmaking district originates in Western Pennsylvania, while that for the Lakes district is obtained from the northeastern section of the lower peninsula of Michigan, and from northern Ohio. Dolomite is quarried chiefly in eastern

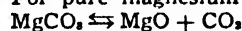
West Virginia for use in the Eastern district. Fluorspar for use in the steel industry comes primarily from western Kentucky and southern Illinois.

Preparation of Fluxes for Use—The fluxes previously mentioned usually require only drying and sizing before use. Those materials which may be added to a slag, such as fluorspar and burnt lime, are desired small in size so that they may rapidly react to produce the desired results. Limestone and dolomite for blast-furnace use and limestone for use in the basic open hearth require careful sizing because the rate of calcination of these materials is controlled primarily by the surface exposed. The chemical reactions involved are:

A. For pure calcium carbonate:



B. For pure magnesium carbonate:



Both reactions are endothermic and are affected by pressure and temperature. Reaction A comes to equilibrium under a pressure of one atmosphere of CO₂ at a temperature of about 1625° F, and it is evident that calcination will begin at temperatures much below steel-making temperatures. For a given limestone at a given calcining temperature the surface area (screen size and shape) determines the time for complete calcination. The times for complete calcination at 1780° F of different sizes of an ordinary grade of limestone are about as follows:

Screen Size of Stone (In.)	Calcining Time (Hr.)
1	1 to 1½
2	2 to 3
3	3 to 4½
4	4 to 6
5	5 to 7½
6	6 to 9

Thus, in a blast furnace, for example, it is possible through a sizing of the stone to have its decomposition completed at the position in the stack desired for most economical operation. In general, the preferred size for blast-furnace stone is 2 to 4 inches, while for open-hearth use the preferred size is 4 to 8 inches, although in the latter case the stone size may be varied more widely in order to cause the occurrence of the lime boil at the desired time.

Burnt lime is produced by the calcination of limestone in large rotary kilns of the general type described later in Section 5, which may be several hundred feet long. They are inclined at a slight angle and are fired from the lower end. The slow rotation moves the stone from the upper charge end and during its passage through the kiln

it decomposes to calcium oxide or burnt lime and is discharged at the fired or lower end, the carbon dioxide

passing out with the waste gases from combustion of the fuel.

SECTION 2

SLAGS

Slag is the name applied to the fused product formed by the action of the flux upon the gangue of an ore or fuel, or upon the oxidized impurities in a metal. As previously indicated, they result from the neutralization of bases and acids, hence, correspond roughly to the salts formed in water solution during chemical reactions at ordinary temperatures. The word cinder is used interchangeably with slag, but cinder is also applied to refuse in a solid form. The subject of slags in general is a very large one and the discussions in this section are intended merely as introduction to the subject.

Metallurgical Functions of Slags—On account of their fusibility, chemical activity, dissolving power, and low density, slags furnish the means by which impurities are separated from the metal and removed from the furnace in both iron and steelmaking processes. Incidentally, as in the open hearth process, for example, the slag performs other important functions. Lying upon the molten metal, it serves as a blanket to protect the metal from the action of hot gases, and as slag is a poor conductor of heat, it may sometimes prevent overheating of the metal and, at other times, conserve its heat. Since slags possess the power of dissolving oxides, they are used effectively as a means of controlling the composition of the bath of metal beneath. In the metallurgy of iron-making, the importance of slag cannot be overemphasized. In the blast furnace, the combination of the composition of the slag and the temperature of the metal and slag in the hearth furnishes the means of control of the sulphur and silicon contents of the iron produced, as discussed in Chapter 12. In open hearth steelmaking, particularly in the basic process, the slag is the only means by which the impurities in pig iron (excepting carbon)

are removed. To the metallurgist, a knowledge of the properties of slags, their chemical behavior, formation temperatures, fusibility and fluidity, and how to control these factors, is essential.

No attempt has been made in this brief statement to do more than call attention to the metallurgical importance of slag in blast furnace and in open hearth furnace operations, both of which are later covered in detail. Slag is of equal metallurgical importance in other processes such as the electric furnace, the Bessemer and other special processes, including duplexing. In each case, the metallurgical features of the slag formed are described under the individual process in later chapters.

Secondary Metallurgical Uses of Slag—In addition to its metallurgical functions in different processes, slag, when recovered from one process, may possess properties which make it useful or of value in other metallurgical operations. Thus, basic open hearth slag, with its high iron and manganese contents, is an excellent addition to the blast furnace burden, as these elements are reduced, and valuable metallics recovered which otherwise would be wasted. Open hearth slag of special composition, such as the tapping slag described later, is of value in making up the blast furnace burden. The iron units in this slag are recovered in the pig iron, the lime present in the slag aids as a flux, and the manganese is reduced and increases the manganese content of the pig iron.

Special methods or treatments have been developed or are in the course of development for the recovery of valuable alloys from electric furnace slag. The value of these processes, as in those mentioned above, is largely one of economics; i.e., does the value of the material recovered amount to more than the cost of its recovery?

SECTION 3

SLAGS AS BY-PRODUCTS

After performing their useful functions in a particular process, the slags produced often are of value for other purposes.

Blast furnace slag, depending upon its chemical composition and the physical form in which it is permitted to solidify, has many uses. Three general physical types are produced by different methods of cooling from the molten state; i.e., air-cooled, granulated, and expanded. **Air-cooled slag**, the "crushed slag" of commerce, is prepared by pouring the molten slag onto a slag bank or into a pit. After solidifying and cooling, the slag is excavated, crushed and screened. **Granulated slag** is prepared by three general methods: pit, jet, or dry granulation. **Pit granulation** consists of running the molten slag directly into a pit of water, producing a relatively coarse, friable product. In the jet process, a modification of the pit process, the stream of molten slag is broken up by a high-pressure water jet as it falls into the pit, and the granulated slag falls into the water in the pit to be quenched further. A product called dry granulated slag is made by a mechanical device using relatively small amounts of water. **Expanded or lightweight slag** is the

foamed product produced when molten slag is expanded by applying a limited quantity of water or with a controlled quantity of water and air or steam. The amount of water used under controlled conditions is less than that required for granulation; consequently, a relatively dry, cellular lump product is formed in the process. **Lightweight slag** is made commercially in the United States in three general ways, which may be designated as the machine process, the jet process and the bank process. The objective of the machine process is the expansion of the molten slag in a revolving device with steam generated by contact of the slag with a controlled quantity of water. In the jet process, one or more high-pressure jets of water impinge on a stream of molten slag, and the product falls into a dry pit. The bank process is the application of water under pressure as the slag flows from the ladle down the face of the bank. The water is controlled or regulated by the pitman in order to expand the slag to a clinker properly and to prevent granulation. Expanded slag has a relatively high structural strength with good insulating and acoustical properties. Nearly 2,300,000 tons were produced in 1953. The

material is normally produced in two sizes, the coarse ranging from $\frac{1}{2}$ " to No. 4 sieve and the fine from No. 4 sieve to dust. The cellular structure of the aggregate consisting of many nonconnecting cells surrounded by thin walls of slag is more pronounced than with the air-cooled slag.

Specific Uses of Blast Furnace Slag—Blast furnace slag has been very successfully put to many commercial uses, some of which are summarized in Table 7—III. The steps in processing the slag to convert it to useful forms are shown schematically in Figure 7—1.

Table 7—III. Principal Uses of Blast Furnace Slag

A. AIR COOLED—Crushed and Screened Slag

Ballast, Railroad
Concrete, Portland Cement
Concrete Units; Masonry, Hollow
Binder Course, Sheet Asphalt Pavement
Bituminous Concrete Base or Surface Course
Bituminous Macadam Base or Surface Course
Bituminous Surface Treatment
Cushion Course for Brick or Block Pavement
Sand for Sheet Asphalt or Bituminous Concrete
Waterbound Base or Wearing Course
Sewage Trickle-Filter Media
Roofing Granules (Bituminous Built-up Roofing)
Mineral Wool
Sub-bases and Special Subgrade (Insulations)
Traffic Bound Roads; Berms, Shoulders
Stabilized Roads and Bases
Porous Back-Fill and Underdrains
Lightweight Aggregate (Expanded Slag)
Embankments, Roadway Fills (Bank Slag)
Glass Sand, Ceramic Ware

B. GRANULATED SLAG

Slag Cements
Soil Corrective, Agricultural Slag
Roadway Insulation Courses
Special Subgrade and Sub-bases
Roadway Fills and Embankments
Ceramic Ware, Glass Sand
Concrete Units
Building Blocks

C. EXPANDED SLAG

Concrete Masonry Units
Structural Concrete
Fireproofing
Floor Tile
Precast Products
Floor Joists and Slabs
Building Blocks
Brick
Acoustical Tile
Curtain Walls; Back-Up, Insulation, Fire Resistive

Slag output in 1953, as reported by the Bureau of Mines, totalled 30,511,603 short tons, a new record high well above the previous record of 26,899,376 short tons turned out in 1951. The value of the 1953 output was \$40,067,294. The recovery of iron by slag processors during 1953 totalled 353,833 short tons.

For concrete making purposes, slag has many desirable features. The strengths, both compressive and flexural, developed in slag concrete are at least equal to those resulting from the use of other types of aggregate. Typical slag concrete weighs approximately ten pounds per cubic foot less than concrete made of other materials, this statement of course not applying to concrete made from special lightweight aggregate, of which lightweight slag is one.

Slag is especially adapted to macadam road construction for the particles, due to their shape, interlock in such a way as to form a very good mechanical bond. The slag fines used to bind or fill the voids of larger

macadam sizes are also very desirable because of the cementing value.

The characteristic rough, pitted surface of slag particles, together with the fact that it is hydrophobic in nature, makes it exceptionally suitable for bituminous construction of all types. These characteristics provide an excellent bond between the bitumen and aggregate, and also permit the application of a sufficiently heavy film of bitumen on the aggregate surface to assure durability and long life in the pavement. Slag sand is particularly valuable for use as fine aggregate in bituminous concrete of both the hot and cold-mix types.

Slag has proved to be an excellent aggregate for airport runway construction of the macadam, bituminous or concrete types.

As railroad ballast, slag has been used successfully by the leading railroads in the United States for many years. Many of the most heavily-traveled tracks in the country are ballasted with slag. More than 5,000,000 tons were so used in 1953. Another use for crushed as well as granulated blast-furnace slag is for agricultural purposes, in which it serves three functions: (1) to reduce soil acidity; (2) to alter and improve physical characteristics of soils; and (3) to supply essential plant nutrients which many soils lack. In regard to (3), however, blast-furnace slag is considered as a soil conditioner rather than as a fertilizer. It contains many of the mineral elements necessary for plant growth, notably manganese, boron, sulphur and iron. Its action as a soil conditioner is similar to that of basic open-hearth slags used widely in the South for this purpose, as discussed later.

Slag also has proved of value as trickle-filter media for sewage disposal. This is especially true in the northern states where the filter media is subjected to severe freezing and thawing as well as wetting and drying. Slag has withstood the extreme exposure in sewage-disposal filter beds to a remarkable degree with practically no breakdown, even after as much as thirty years of service.

Slag has been in great demand for many years for use as a covering for built-up bituminous roofs and as granules for covering asphalt shingles.

A demand for slag for use in the production of mineral wool in the past few years has materially increased. A large percentage of all mineral wool produced in the United States is made of slag. Slag is re-melted in a small cupola and the molten product, after reaching a desired temperature, is blown into wool by a jet of high-pressure steam or compressed air. The production of mineral wool from the molten slag as it comes from the blast furnace has, after much experimentation, not been found practical.

One of the earliest uses for blast-furnace slag was in the manufacture of cement. Slag with a low magnesia content is required for cement-making. In granulated form, such slag has been used in large quantities in the manufacture of portland as well as other types of cement. For portland cement it is used as a part of the raw material that is burned in the usual way to form clinker, which, in turn, is ground into cement by methods described in Section 5 of this chapter. Over 1,400,000 tons of blast-furnace slag were used for the making of cement in 1953.

Basic open-hearth slag has a more limited field of application as a by-product than has blast-furnace slag but, like the latter, it also has found an important use as a soil conditioner, especially in the southern states. In the northern states, with present steel-making practices, the phosphorus content of basic open-hearth slag is low compared to that of the basic open-hearth slags produced in the southern states where ores of a much higher phosphorus content are used.

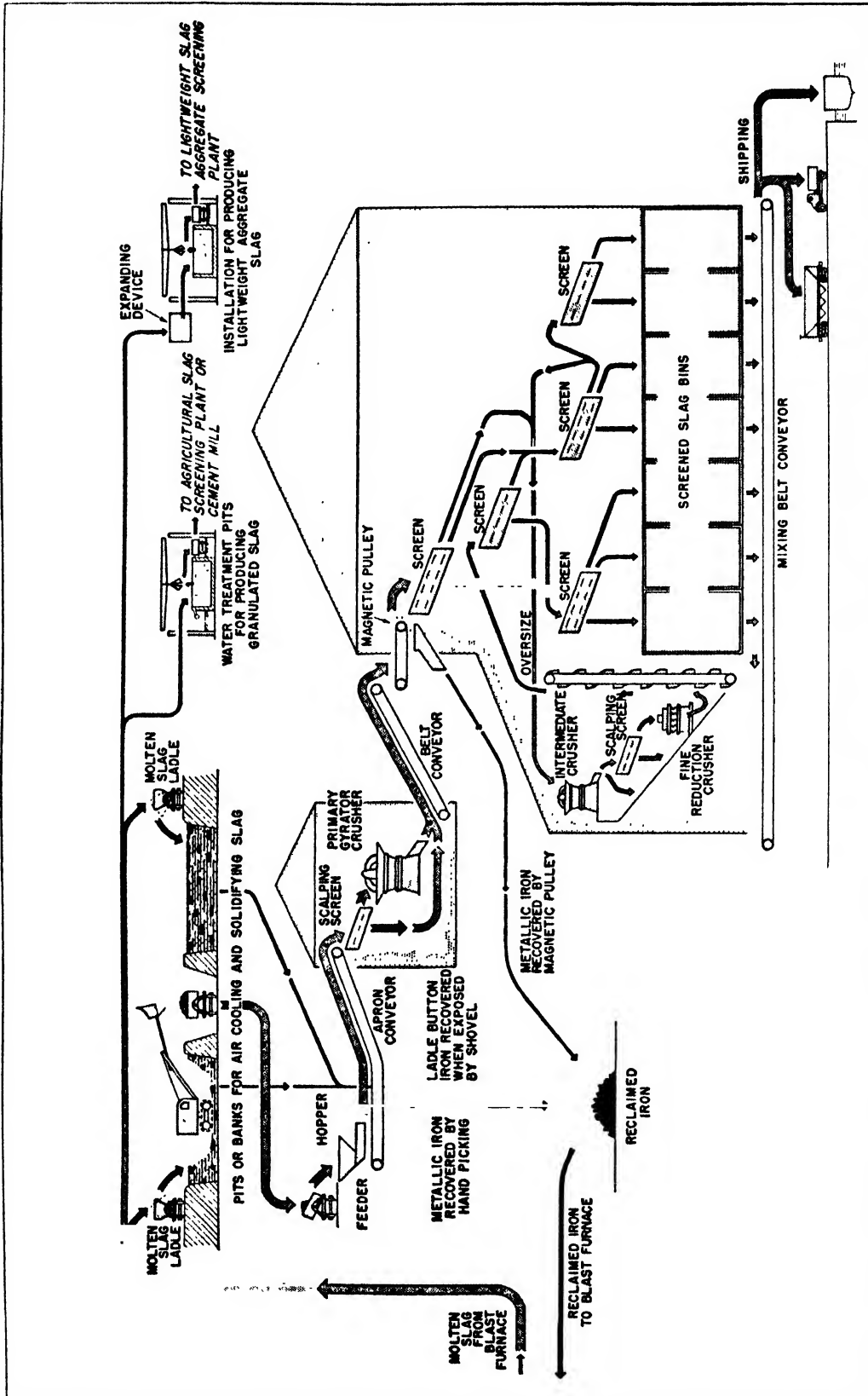


FIG. 7-1. Schematic representation of the steps involved in processing blast-furnace slag to convert it to useful forms. The purpose of the multiple screens preceding the screened-slag bins is to sort the crushed slag into appropriate sizes.

The greatest production of basic open-hearth slag for use as a soil conditioner is produced at a plant in which the steel is produced by the duplex process involving use of acid converters and tilting basic open-hearth furnaces maintaining a dominant pool. In this process approximately 80 per cent of the next succeeding charge in the form of blown metal is washed through the refin-

ing slag of the previous heat impregnating the slag with phosphorus. The slag is then poured from the furnace over the fore plate into slag pots beneath the furnace, as described in Chapter 17 entitled "The Duplex and Triplex Processes."

Flush slags from the conventional open-hearth process also may be used as a soil conditioner.

SECTION 4

FUNCTIONS OF SLAGS AS SOIL CONDITIONERS

The chief food elements that plant life or vegetation obtains from the soil are potassium, phosphorus, and nitrogen, but, as mentioned previously, others such as iron, sulphur, and manganese are essential. For plants to thrive, soil conditions must be favorable, particularly with respect to humus content and basicity. Thus, if the soil is in clods, or if it is too acid or basic, unsatisfactory crop results are obtained, irrespective of the amount of fertilizer applied. An important function of slags is to correct such adverse conditions.

The chemical bases in slags consist mainly of lime, with some magnesia and small amounts of other basic compounds. The lime in slag, however, is in loose chemical combination with silica, iron, and manganese, so that it does not "burn" like ordinary agricultural lime nor revert to carbonate, but remains in a stable, almost neutral form, available as needed over long periods of time until exhausted.

While slags used for agricultural purposes cannot be called fertilizer, they do contain some fertilizing elements and, if of a suitable grade and properly prepared and applied to the soil, do promote conditions essential to profitable farming. Hence, they rightly are called soil conditioners.

The first basic slag used as soil conditioner was from the Thomas-Gilchrist or basic Bessemer process of steel-making. The possibilities of this slag as a soil conditioner and stimulant to plant growth were investigated first in England in 1884. Its use was adopted in Scotland in 1890, and the practice spread rapidly throughout Europe in succeeding years.

As early as 1900, a German investigator reported on the use of blast-furnace slag as a fertilizer, with special reference to the beneficial effects of soluble silica in the material. Comparatively little comprehensive research work on the agricultural value of blast-furnace slag has been done in the United States, although coordinated test work has been undertaken in recent years and is still in progress.

In the South, the Tennessee Coal and Iron Division of the United States Steel Corporation has been preparing and marketing basic open-hearth slag from their duplex-

ing process, as a soil conditioner since 1915. The high-phosphorus content of the slag, previously referred to, justifies economically the cost of preparation and freight charges of the product for use in that area. Originally, it was the aim of the company to produce a slag for agricultural purposes in which the citric-acid-soluble phosphoric-acid content was high enough to meet fertilizer specifications. However, experiments established the fact that slags with a much lower phosphoric-acid content were so useful as soil conditioners that it appeared best to aim for greater tonnages containing between 8 and 12 per cent phosphoric acid, rather than a limited tonnage of product with an extremely high content of phosphoric acid. Steps in its preparation are as follows:

The high-phosphorus slag from the tilting open hearth, after cooling, is delivered to the soil-conditioner plant and mixed with flush slag containing less phosphorus. The large slag cakes are first broken by the use of a heavy steel ball, after which large pieces of steel are removed by the use of a conventional electromagnet. The slag is then transferred by a grab bucket to a roller crushing unit which breaks it down to 1½-inch maximum size. Elevating conveyors then deliver the slag over a magnetic roller and vibrating screens from which the larger pieces of slag are returned to be recrushed and shot-steel scrap is loaded for use in the blast furnaces. That portion of the slag which passes through the screen is then conveyed to a conventional tube mill for final grinding.

From the tube mill the slag is conveyed to storage bins and thence to a bagging unit where once again it is passed through screens to remove all tramp particles. Samples for chemical analysis and fineness testing are taken hourly from the product discharged from the tube mill. The product is packaged in paper bags, each containing 100 pounds net weight and marketed under the trade name of "Ground Open Hearth Basic Slag." Phosphoric acid content ranges from 8 to 12 per cent; fineness 70 per cent through a 100-mesh screen. Facilities are available for the loading of this slag in bulk not only in railroad cars but spreader trucks as well.

SECTION 5

MANUFACTURE OF PORTLAND CEMENT

Portland cement is a finely pulverized material consisting of certain definite compounds of lime (CaO), silica (SiO_2), alumina (Al_2O_3) and iron oxide (Fe_2O_3). Its most familiar use is in concrete. Concrete is a mixture in which a paste of portland cement and water binds fine and coarse materials (sand, pebbles, crushed stone, crushed blast furnace slag, etc.) known as aggregates into a rock-like mass as the paste hardens due to the formation of new compounds through chemical action

between the original compounds of the cement and the water. In simple terms, concrete is a mass of aggregates held together by a hardened paste of portland cement and water.

Raw Materials—Based on the 1949 "Minerals Yearbook" published by the Bureau of Mines, approximately 68,621,000 tons of raw materials (exclusive of fuels and explosives) entered into the manufacture of 209,727,417 barrels of portland cement in the United States in that

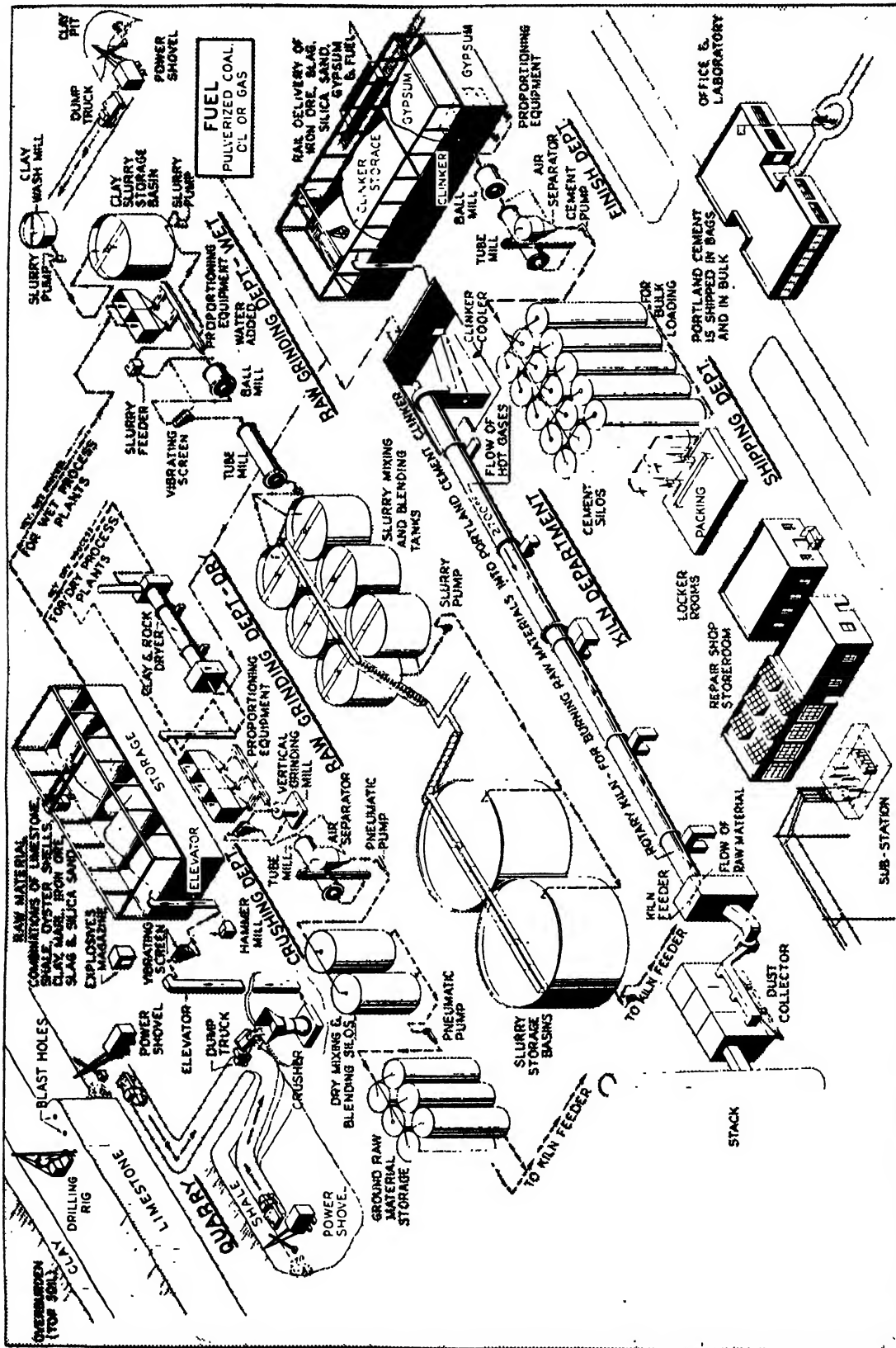


FIG. 1-2. Schematic representation of cement-plant layout and equipment for producing Portland cement by the "dry" and the "wet" methods. (Courtesy Portland Cement Association.)

year. This is an average of about 654 lb. of raw materials for each barrel of finished portland cement weighing 376 lb. Loss of weight is caused by the process of calcination in which moisture, carbon dioxide and other gases are driven off in the kilns. Materials are chiefly high-calcium limestone, clay or shale, argillaceous limestone, blast-furnace slag and marl. Totals were:

Raw Material	Short Tons
Cement rock (argillaceous limestone)	12,628,494
Limestone and oyster shells	44,968,739
Marl	722,606
Clay and shale	6,698,408
Blast-furnace slag	847,375
Gypsum	1,543,198
Sand and sandstone	724,624
Iron-bearing materials	346,542
Other materials such as:	
diatomite, fluorspar, pumicite, flue dust,	
pitch, red mud and rock, hydrated lime,	
tufa, cinders, calcium chloride, sludge,	
grinding aids, and air-entraining com-	
pounds	140,999
Total	68,620,985

Also according to the Bureau of Mines, the following quantities of fuel were used by the portland cement industry in the United States and its territories in 1949 (for all cement-mill fuel consumption including the process of burning in the kilns, independent power production and other uses):

Coal (tons)	7,987,560
Oil (gallons)	192,637,116
Gas (cubic feet)	84,689,284,000

Based on figures from this same source relating to cement production and fuel consumption in those plants that use only one fuel exclusively, the following amounts of fuel were required to produce one barrel of cement:

Coal	115.1 lb.
Oil	8.44 gal.
Natural gas	1,519 cu. ft.

Dry and Wet Processes—There are two processes which may be used to make portland cement (Figure 7-2). These are called the "dry" and the "wet" methods. In the dry process, after the crushed raw materials are taken from storage, dried and blended in the proper proportions, the mixture is fed into huge rotating ball mills or impact mills of various types for preliminary grinding. The ball mills get their name from the hundreds of large steel balls inside them which are carried on ribs up one side of the rotating mill and cascaded down onto the material being ground.

The mixture then goes to a tube mill, which is similar to the ball mill except that the material is more finely ground by 40 to 50 tons of smaller steel balls. Some plants use combination ball-and-tube mills, called compartment mills. Motors up to 1,000 horsepower are required to operate them. Grinding is frequently done in circuit with air separators which remove the fine material from the mill discharge and return the coarse material for further grinding, effecting economies in grinding.

After the dry raw material is ground, it is carried to storage. Samples are analyzed for chemical content to insure uniform quality. Accurately proportioned amounts of the dry raw material are then ready to go to the kilns for burning.

The wet process is very much like the dry except that, at the ball mill, water is added and the material is ground wet to give it a thick, creamy, soup-like texture. The

cement makers call this slurry. Sometimes, this wet grinding is done in closed circuit with classifiers to remove finished fine material as fast as produced and return the coarse material for additional grinding. In this case, the finished slurry has a very high water content which is reduced in large thickeners before using the slurry as kiln feed.

The slurry is then pumped into tanks where it is tested and blended to the desired chemical composition. From the blending tanks, the slurry is pumped to large storage tanks to be held until needed for burning.

The Burning (Calcining) Operation—The prepared and blended mixture of raw materials is fed into rotary kilns, where the mixture is calcined or burned to what is known as cement clinker. Briefly described, a rotary kiln is a nearly horizontal steel cylinder, lined with suitable refractory brick and 6 to 12 feet in diameter and from 60 to 500 feet in length. Such kilns are continuous in operation. The raw material is fed continuously into the higher end of the long, inclined cylinder which rotates continuously at a rate of about one revolution per minute. By reason of the inclined position of the kiln and its rotary motion, the charge moves gradually and continuously toward the lower end of the kiln where it is discharged.

Heat for calcining the charge is supplied by combustion equipment that fires axially into the kiln at the discharge end. The direction of flame travel, therefore, is opposite to the direction of progress of the charge, and the charge is heated hotter and hotter as it travels toward the firing end. Pulverized coal, natural gas, or fuel oil is used as fuel. A temperature of 2800° F is attained in the hottest zone at the discharge end of the kiln. The heat serves two purposes: (1) to cause the desired chemical changes to take place in the raw mixture, and (2) to form clinker by fusing the transformed materials into small masses about the size of marbles.

The clinker resulting from the burning of the raw material in this way is then cooled and pulverized in equipment similar to that used for dry raw material grinding, together with a small amount of gypsum added to regulate the setting time, and becomes the portland cement of commerce.

A barrel of cement weighs 376 pounds net and equals four sacks of 94 pounds each. Each sack contains approximately one cubic foot. Cement has been shipped in cloth sacks or paper bags, in bulk on large jobs, and for export in wooden barrels or paper-lined cloth sacks. Today, nearly all cement is shipped in bulk or in paper sacks.

Chapter 8

REFRATORIES

SECTION 1

CLASSIFICATION OF REFRATORIES

Refractories are the chief materials used in the steel industry in the construction of all furnaces and in the lining of retaining vessels, as well as in the flues or stacks through which hot gases are conducted. At the risk of over-simplification, they may, therefore, be said to be materials of construction exposed to temperatures above a dull red. This material is expensive, and any abrupt failure in the refractories results in a great loss of time, equipment and product. Therefore, the problems of obtaining refractories suitable for each branch of the industry and a refractory best suited to each specific purpose are of supreme importance. Economics figure largely in the solution of these problems, as a refractory best suited for an application is not necessarily the longest lived, but the one which provides the best balance between initial installed cost and service performance. These balances are never fixed, but are constantly shifting as a result of the introduction of new processes or new types of refractories. History reveals that refractory developments have occurred largely as the result of the pressure for improvement caused by the persistent search for superior metallurgical processes. The rapidity with which these ever-recurring refractory problems have been solved has been a large factor in the rate of advancement of the iron and steel industry. To discuss the many factors involved in these problems and to provide information helpful to their solution are the objects of this chapter.

Refractories may be classified in a number of ways, no one of which is completely satisfactory. From the chemical standpoint, refractory substances, in common with matter in general, are of three classes; namely, acid, basic, and amphoteric or neutral. Theoretically, acid refractories should not be used in contact with basic slags, gases or fumes, while basic refractories should be exposed to no other conditions. Actually, for various reasons, these rules are continually violated. Also, with the possible exception of carbon, the existence of a strictly neutral refractory may be doubted. Hence, the time-honored chemical classification is primarily academic, and of little value as a guide to actual service possibilities. Classifications by use, such as blast-furnace refractories or open-hearth refractories are generally too broad and are constantly subject to revision. Mineralogically, refractories may be classified two ways; namely, with reference to the raw materials, and with reference to the minerals predominating after processing for use. For the present purpose, the following raw-materials classification is believed to offer the best possibilities for a clear understanding of the origin and nature of steel-plant refractories.

A. SILICEOUS GROUP

Quartzite—Quartzite or ganister is the most commonly used, and the purest, of the siliceous raw materials. Massive rock forms analyzing over 98 per cent SiO_2 , are

the chief raw materials for silica-brick manufacture, and are mined in Pennsylvania (Medina quartzite), Wisconsin (Baraboo quartzite), Alabama, Utah and California. Pure quartz pebbles are sometimes found loosely bonded by a matrix of fine silica and mica as a natural rock known as a conglomerate. The Sharon conglomerate in Ohio is of this type and is being increasingly used for silica-brick manufacture, as the impurities may be readily removed by washing. In recent years, it has been clearly demonstrated that the total Al_2O_3 and alkali content of silica brick must be kept under one per cent for satisfactory performance in open-hearth roofs, and preference is being given to even purer brick containing less than 0.5 per cent total impurities. For this reason, much care is taken to select and wash all silica raw materials used in producing open-hearth roof brick.

Sandstone—Sandstone, or firestone, is a sedimentary rock consisting essentially of bonded sand grains, and usually analyzing 90 to 93 per cent SiO_2 , 3 to 5 per cent Al_2O_3 , and some iron oxide and lime. It is relatively soft and often striated, thus permitting easy cutting or splitting into blocks or shapes for use in the raw state for lining soaking pits or acid Bessemer converters.

Mica Schist—Mica schist is a highly laminated micaceous silica rock of composition similar to sandstone, but generally slightly lower in silica. Like sandstone it is readily cut and is also used in the raw state for hot metal ladle and mixer linings in addition to linings for soaking pits and converters.

Siliceous Fireclays—While the term siliceous fireclays might properly refer to clays having a rather wide range in silica content, reference here is to those clays with a minimum of 75 per cent SiO_2 , which are employed in the manufacture of semi-silica brick, and are characterized by a very low percentage of impurities such as alkalies, alkaline earths and iron oxides. Somewhat less siliceous and often more impure clays are also widely used for mortars and daubing purposes with siliceous linings.

B. FIRECLAY GROUP

Chemically, clays are all hydrous silicates of alumina and occur widely distributed. They are identified as being plastic when in a wet and finely divided state (as by pulverizing, wetting, and mixing), rigid when dried, and vitreous when heated to a sufficiently high temperature. Clays may be residual or sedimentary, and have been formed by the natural decomposition or weathering of feldspathic rock. Ordinary varieties contain high percentages of combined water and impurities that render them unfit for use as high-temperature refractories. Impurities include alkalies, titania, compounds of iron, calcium and magnesium, and organic matter from various sources. Even in small quantities the alkalies, and the compounds of iron, calcium and magnesium are important fluxes and have a pronounced effect on the

refractory properties of the clay. The species used as a refractory is known as fireclay. Fireclays are of sedimentary origin, are usually associated with coal measures, and contain limited percentages of fluxing impurities. There are several varieties, namely:

Plastic Fireclay—A fireclay of sufficient natural plasticity to bond nonplastic materials.

Flint Fireclay—A hard or flint-like fireclay occurring as an unstratified massive rock, practically devoid of natural plasticity and showing a conchoidal fracture.

Nodular Fireclay—Also called burley or burley flint clay, nodular fireclay occurs in the form of a rock containing aluminous or ferruginous nodules, or both, bonded by fireclay.

Kaolins—While not fireclays, certain kaolins are highly refractory and are sometimes employed in the manufacture of fire brick. Kaolins are both sedimentary and residual, and quite pure, generally closely approaching the theoretical clay composition represented by the formula $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$.

C. HIGH-ALUMINA GROUP

This group includes those materials capable of serving for the production of refractories of higher alumina content than the maximum that can be provided by fireclays, namely, more than 47.5 per cent Al_2O_3 . Occurring naturally are two classes of minerals, those which are combinations of Al_2O_3 and H_2O , typified by bauxite and diaspore, and those which are combinations of Al_2O_3 and SiO_2 , typified by sillimanite, andalusite and kyanite.

Bauxite and Diaspore—Bauxite is a high alumina rock composed of $\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$ and $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$, found in Arkansas and South America. It is less used for refractories than diaspore, $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$, although its use is now increasing as supplies of diaspore diminish. Diaspore is found chiefly in Missouri where it occurs in lenses or pockets chiefly as diaspore clay, a rock consisting essentially of diaspore bonded by fireclay. The best grades, upon calcination yield a product containing between 70 and 80 per cent Al_2O_3 . Impurities include iron oxides and titania.

Sillimanite, Andalusite, and Kyanite (Cyanite)—These minerals all have the formula $\text{Al}_2\text{O}_3 \cdot \text{SiO}_2$ and contain theoretically 62.9 per cent Al_2O_3 and 37.1 per cent SiO_2 . On heating, all form mullite ($3 \text{Al}_2\text{O}_3 \cdot 2 \text{SiO}_2$) and a siliceous glass, but differ in the ease with which this decomposition takes place, kyanite being easiest to convert (about 2415° F) and sillimanite the hardest (about 2785° F). Indian kyanite is most desirable for high-alumina refractories, although South African and domestic kyanites are now being used, sometimes blended with the Indian kyanite. Brick made from these materials, usually referred to as sillimanite or mullite brick, are generally superior in high-temperature volume stability and spalling resistance to other types of high-alumina brick.

Fused alumina, an electric-furnace product, is frequently used as an addition in high-alumina refractories.

D. MAGNESIUM-SILICATE GROUP

This group includes the olivines and serpentines. Olivines have the general formula $2 \text{RO} \cdot \text{SiO}_2$, in which the RO may be MgO, CaO, FeO, or MnO. The most common refractory olivine is a mixture of forsterite ($2 \text{MgO} \cdot \text{SiO}_2$) and fayalite ($2 \text{FeO} \cdot \text{SiO}_2$), and is found primarily in North Carolina. Serpentines are hydrous magnesium silicates of the general formula $3 \text{MgO} \cdot 2 \text{SiO}_2 \cdot 2 \text{H}_2\text{O}$. Forsterite is the most desirable magnesium silicate for refractories, and is generally the end product aimed for in employing these minerals.

E. MAGNESIA-LIME GROUP

In this group have been classed all natural and synthetic magnesites, brucite, and dolomite. These constitute the most important group of refractories for basic steelmaking processes, as all are sources of magnesia, MgO, the best basic refractory known.

Natural magnesite, MgCO_3 , is rarely found in a pure state, but contains varying percentages of silica, alumina, iron oxides and lime, so that on calcination the MgO content generally ranges from 80 to 85 per cent. Silica is the most objectionable impurity and varies from 4 to 10 per cent. Formerly, much magnesite was imported from Austria, but for several years, the steel industry has relied almost exclusively on sources in the state of Washington and in Canada.

Magnesia is derived synthetically from sea water brines, well brines, dolomite and brucite, or processes combining brines and dolomite. The production of magnesia or, more popularly, sea-water magnesite by these means has increased rapidly in recent years, and this material is now used extensively in the manufacture of basic brick and granular refractories.

Brucite is a hydrate of magnesia found chiefly in Nevada and is used largely in conjunction with dolomite in the manufacture of granular refractories.

Dolomite is a double carbonate ($\text{CaCO}_3 \cdot \text{MgCO}_3$), which calcines, evolving 47.8 per cent as CO_2 , leaving 30.4 per cent CaO and 21.7 per cent MgO. Principal impurities are silica and alumina, but large deposits are found in several localities such as Illinois and Ohio, which closely approach the theoretical composition. Thus, for best refractory service, a dolomite should contain more than 21 per cent MgO, less than 1 per cent SiO_2 , and less than 0.5 per cent Al_2O_3 , in a crude state. From the standpoint of consumption, dolomite exceeds all other refractories, as in either the raw, calcined, or dead-burned state*, it is the chief mainstay in open-hearth-bottom maintenance.

F. CHROMITE GROUP

Chrome ores in general consist of a highly refractory spinel ($\text{RO} \cdot \text{R}_2\text{O}_3$) composed of FeO, MgO, Al_2O_3 and Cr_2O_3 , in various proportions, and less refractory associated silicates. Chrome ores of widely different compositions are suitable for refractory purposes, the chief limitations being lime, iron oxide, and silica. Most of the suitable refractory grades are obtained from the Philippines, Cuba, India, Russia, South Africa, Greece, and Turkey.

Chrome ore is used either alone or in various combinations with magnesia in brick manufacture, and as a granular or plastic refractory for a number of purposes.

G. CARBON GROUP

This group includes natural and artificial graphites, and various types of coal, coke, and tar.

Graphite deposits occur widely distributed, both in this country and abroad. It is often mixed with calcareous or siliceous rock and requires expensive purification. Flake graphite, as found in Ceylon, is preferred for crucible and stopper-head manufacture, in which the graphite is bonded with high percentages of clay. The amorphous graphites found in Mexico are sometimes used to enhance the slag-resistant properties of plastic-clay refractories.

Baking of molded coke and tar or coke and pitch mixtures will produce artificial graphite or carbon shapes, although much higher temperatures are required for

* See Section 2B, Granular Refractories—Fired Products for explanation of terms "calcined" and "dead burned."

Table 8-1. Typical Chemical Compositions of Refractories

TYPE	PER CENT									
	SiO ₂	Al ₂ O ₃	TiO ₂	CaO	MgO	FeO	Fe ₂ O ₃	Cr ₂ O ₃	Alkalies	C
Silica Brick:										
Conventional	95 - 96.5	0.6 - 1.2	0.04 - 0.1	2.0 - 3.5	0.09 - 0.15	0.4 - 0.7	0.05 - 0.35
Super Duty	96.4 - 97.4	0.2 - 0.4	0.03 - 0.06	2.2 - 3.0	0.09 - 0.10	0.3 - 0.6	0.05 - 0.06
Ganister	97.0 - 98.20	0.4 - 0.9	0.05 - 0.11	0.05 - 0.15	0.07 - 0.10	0.3 - 0.6	0.05 - 0.35
Firestone	93.47	2.77	0.25	0.44	0.03	0.81	0.01	0.80
Mica Schist	81.98	11.12	1.08	0.24	0.25	3.10
Fireclay Brick:										
Super Duty	51 - 54	40 - 43	2.1 - 2.5	0.2 - 0.5	0.2 - 0.5	1.4 - 2.3	0.8 - 1.4
High Duty	51 - 59	35 - 40	2.0 - 3.0	0.3 - 0.5	0.5 - 0.6	1.6 - 2.5	1.5 - 2.6
Intermediate Duty	56 - 70	25 - 36	1.3 - 1.9	0.2 - 0.4	0.5 - 0.6	1.8 - 3.4	1.0 - 2.7
Low Duty	53 - 69	25 - 34	1.0 - 2.0	0.3 - 0.6	0.4 - 0.6	2.3 - 3.4	1.8 - 2.9
Ladle Brick	63 - 67	23 - 29	1.2 - 1.6	0.4 - 1.1	1.0 - 1.1	2.0 - 5.0	2.5 - 3.6
Semi-Silica Brick (New Jersey)	78.05	18.50	1.65	0.11	0.04	1.01	0.33
High Alumina Brick:										
50% Al ₂ O ₃	43.0 - 47.0	47.0 - 51.0	2.2 - 2.4	0.5 - 0.6	0.5 - 0.6	0.9 - 1.6	0.8 - 1.3
60% Al ₂ O ₃	32.0 - 37.0	58.6 - 60.4	2.8 - 3.3	0.2 - 0.5	0.3 - 0.5	1.7 - 2.4	1.4 - 1.9
60% Al ₂ O ₃ (Kyanite Base)	32.0 - 35.0	62.0 - 66.0	0.7 - 1.5	0.1 - 0.3	0.1 - 0.3	0.05 - 0.2	0.3 - 0.5
70% Al ₂ O ₃	21.0 - 26.0	68.0 - 71.0	2.9 - 3.7	0.1 - 0.2	0.1 - 0.3	1.8 - 3.0	1.2 - 1.6
Basic Brick:										
Forsterite	31.0 - 34.0	1.5 - 1.7	1.4 - 1.6	53.0 - 55.0	9.0 - 9.3	1.4 - 1.7
Chrome	3.0 - 7.0	17.0 - 30.0	0.2 - 0.4	16.0 - 21.0	13.0 - 15.0	34.0 - 42.0
Magnesite	3.7 - 6.0	0.4 - 2.0	1.8 - 4.5	86.0 - 92.0	0.4 - 5.5	0.7 - 1.0
Magnesite, 96% MgO (Burned)	1.45 - 1.65	0.15 - 1.15	0.90 - 1.80	93.5 - 96.0	0.30 - 0.60	0 - 0.65
Chrome-Magnesite:										
Fired	2.0 - 8.0	21.0 - 23.0	0.9 - 1.2	30.0 - 37.0	10.0 - 12.0	22.0 - 30.0
Chemically Bonded	5.0 - 5.5	17.0 - 22.0	0.3 - 1.0	29.0 - 32.0	10.0 - 13.0	23.0 - 30.0
Magnesite-Chrome:										
Fired
Chemically Bonded	4.9 - 5.5	6.5 - 23.0	0.7 - 2.7	33.0 - 69.0	8.9 - 9.4	6.0 - 23.0
Dead-Burned Magnesite	4.5 - 7.5	1.8 - 2.4	4.0 - 5.3	78 - 84	4.0 - 5.5
Dead-Burned Dolomite	0.78 - 3.00	0.45 - 0.94	49.35 - 51.80	34.74 - 38.01	5.50 - 9.10
Chrome Ore	1.2 - 9.1	12.4 - 28.4	0.1 - 1.0	10.6 - 18.7	11.9 - 16.4	32.4 - 49.9
Carbon Block	4.3 - 5.9	3.0 - 3.1	0.13 - 0.43	0.1 - 0.2	0.4 - 0.8	88.2 - 92.0

graphitization. Carbon is used extensively as a refractory, and may be made from foundry coke, petroleum coke, or anthracite coal, bonded with pitch or tar of low moisture content. Such refractories are being used increasingly for blast furnace refractories and other applications where reducing atmospheres prevail.

PREPARATION OF REFRACTORIES

Steel-plant refractories are used in a wide variety of forms, the preparation of which may vary from little

more than the operation of mining, to highly complex grinding, screening, molding, and firing procedures demanding precise control throughout. Hence their preparation will be discussed according to their use in the massive, granular, or finely-divided form, and according to whether these forms are in the raw, fired, or chemically-bonded state.

The chemical compositions of the various types of refractories discussed above are given in Table 8—I on Page 182.

SECTION 2

A. MASSIVE REFRACTORIES

Raw State—Sandstone and mica schist are invariably used in the form of large blocks sawed or split from the raw rock, and further shaped as needed on the job. To minimize spalling from heat, these blocks are laid flat, with laminated edges exposed to the heat. Chrome ore and olivine have been similarly used for open-hearth bridge walls, but their use in block form has practically ceased. A siliceous insulating brick, accurately cut to standard brick size from natural diatomaceous earth, is sometimes used to back up fireclay-brick walls. Such brick have superior insulating value as compared with brick made from the same material by conventional methods, although that property will be greater with heat flow normal to the laminations.

Burned Products—Fired brick and shapes of all types of refractories constitute the bulk of refractories used in the massive form. The processes by which these brick and shapes are produced are basically similar although, as might be expected, the nature of the raw materials and the wide variety of properties desired in the final products have caused the adoption of a number of variations in these processes.

The first step is the mining of raw materials. Mining methods depend upon the location, size, and uniformity of the deposit. For the most part, deposits are underground, and mining methods are similar to those employed in the various types of underground mining and quarrying. However, large uniform deposits, such as are characteristic of some plastic clays and dolomites, are worked more economically by the open-pit method, even though this may involve stripping of many feet of overburden. More exacting refractory requirements and diminution of established sources of some materials are constantly forcing changes in methods of mining. For example, the quarrying of flow ganister on mountain sides for use in silica-brick manufacture is gradually giving way to the mining of ledge rock in some localities in Pennsylvania.

Following the mining, some purifying operations may be necessary. At present, these may be merely visual selection, weathering to remove soluble salts, or only a simple washing operation as employed with some flint clays and increasingly so with some quartzites. However, the recent employment of a flotation process for removal of lime and dolomite impurities from natural domestic magnesites is indicative of the trend toward raw-material beneficiation.

The future will likely see the adoption of more ore-dressing practices by the refractory industry, although careful study will be required to determine which procedures are applicable and can be justified with each material.

The next step is one of crushing, grinding, screening, and mixing. These are important operations, as, in large measure, they control the density, porosity, strength,

spalling resistance, and thermal characteristics of the brick, although there are no general rules for their control. The fineness of the grinding must be varied for different raw materials and different mixes, and with a given material the grinding to obtain a certain effect depends upon equipment available for subsequent operations. Formerly, no screening was done, but by 1930 it was being used as a means of controlling the particle size which, in turn, is one of the means of controlling the significant properties of porosity and bulk density of brick, and led to the important developments of super-duty fireclay brick and vastly-improved basic brick. Mixing may consist of adding fines to coarse material of the same kind to control the grain or particle size, or of adding one material to another; for example, lime to quartzite and plastic clay to flint clay to act as binders; burned clay or refractory to a plastic mix to control shrinkage; or magnesite to olivine, magnesite to chrome ore, bauxite to clay to change the composition and character of the brick.

Forming operations may be carried out by hand molding, extrusion, extrusion and pressing, dry pressing, drop molding, and pneumatic ramming. In 1905, all brick were formed by hand. The moist mix (15 to 20 per cent moisture) was tamped into forms set on pallets, the forms were removed, the pallet set upon a "hot floor" to "temper" and the tempered brick were repressed in a hand-operated press, thoroughly dried and then fired in a rectangular or round down-draft periodic kiln. By 1913, however, a goodly percentage of the brick was being formed by machine.

In the stiff-mud process, the wet clay mix, after tempering with 12 to 15 per cent moisture in a pug mill, is extruded through a die by a power-driven auger or plunger, and the column is cut with taut wires into the sizes desired. This may be the final processing before drying and firing, or the shapes may be given a final shaping and branding in a repress machine. This process is used almost exclusively for fireclay products.

A more universal forming method is power pressing or dry pressing. In this process, the refractory is thoroughly mixed with from 3 to 8 per cent moisture, and this relatively dry mixture is fed to steam-, electric-, or hydraulic-powered presses, where measured quantities are pressed in shape under pressures varying from 1,000 to 10,000 pounds per square inch. Dry-press products are generally more true to shape and more uniform in size than stiff-mud products. The density of the dry-press products may be greater or less, depending on the degree of particle-sizing control exercised and the pressure employed, but the cold strength is generally less than that of stiff-mud brick. Obviously, dry pressing lends itself to a wider range of refractories than extrusion processes, and fireclay, silica, and all types of basic brick are frequently made in this manner.

A measure employed to improve the bulk density of brick with either of the foregoing processes is **deairing**. In the stiff-mud process, the auger chamber is evacuated. While the effectiveness varies with different clays, remarkable improvements in the properties of the clay column and the final product are often effected by this means. During pressing in the dry-press process, the mold box is evacuated of air through small holes in the pressing pads.

Drop molding is a process exclusive to the manufacture of silica brick, as the prepared mix has little or no plasticity. In this operation, the wet mixture of crushed ganister and about 2 per cent lime is elevated by bucket conveyor and dropped from a height of 15 to 20 feet into multiple molds below resting on pallets. The mold is struck off and removed, and the brick placed on dryer cars while still on pallets. Organic binders are often used to enhance the weak green strength of silica brick and thus minimize breakage.

Many large or special shapes of superior refractories which cannot be pressed in the usual way are produced by **pneumatic ramming** in steel molds or by special **vibrating devices**. The mix employed is very low in moisture and very carefully sized, as the aim is generally to obtain high density. The mix is fed in slowly and uniformly during ramming.

Refractories are burned or fired in either periodic kilns of the round down-draft or rectangular type, or in tunnel kilns. In **periodic firing**, the dried green brick are set in the kiln, the kiln is gradually brought to the required temperature, held for varying periods, cooled and drawn. This cycle will vary from about two to four weeks, with a production of about 25,000 to over 50,000 nine-inch equivalent, (cubical content of a standard $9 \times 4\frac{1}{2} \times 2\frac{1}{2}$ -inch brick), per cycle. In **tunnel-kiln firing**, the green brick loaded on small cars are passed slowly through a long tunnel-shaped brick structure in a direction counter to the flow of gases, whereby they pass progressively through the pre-heating, firing, and cooling zones, generally taking 3 to 5 days for the trip. Temperature of firing is important, because both the quality and properties of the brick may be affected. By maintaining an oxidizing or a reducing atmosphere, the subsequent behavior of most brick car. be changed, and by controlling the rate of heating and the firing temperature, changes in the crystalline structure can be effected, which, in turn, may affect the service performance of the brick. In general, the objects in firing are: (a) to drive off hygroscopic and combined water; (b) to bring about desired chemical changes—drive off CO_2 and oxidize iron and sulphur compounds, organic matter, etc.; (c) to effect transformations of the mineral constituents and convert them to most desirable stable forms; and (d) to effect necessary combinations and vitrification of bonding agents.

By other developments in technique of manufacture, the porosity of fireclay and silica brick can be greatly increased, making the brick good insulators and considerably lighter in weight. Hence, they are designated as **lightweight** or **insulating brick**. Processes for making these brick aim for high porosity, preferably with fine pore structure. This is accomplished by mixing a bulky combustible substance, like sawdust or ground cork, or a volatile solid, such as naphthalene, with the wet batch, by forcing air into the wet plastic mass, or by mixing into the batch reagents which will react chemically to form a gas and a product not injurious to the brick.

Carbon refractories, as sometimes used in blast-furnace linings, are truly massive refractories, commonly being used in blocks 22 inches by 30 inches in cross section, up to 15 feet in length, and exceeding 6000

pounds in weight. As previously mentioned, these blocks are made of calcined anthracite coal and/or low-ash coke with a tar bond, and are formed by extrusion, after which they are baked in coke at a temperature of approximately 1800° F for long periods, seven weeks sometimes being required to complete a cycle. Smaller shapes and brick of standard size are likewise made and are replacing the larger blocks in many blast-furnace wall installations.

Chemically Bonded Products (Basic Brick)—Since their introduction about 1923, the manufacture of chemically-bonded basic brick has steadily increased, until today unburned basic brick constitute approximately one-half of the total basic-brick production. In this process, a suitable chemical-bonding material is mixed with the raw materials and the brick are formed under pressures sometimes exceeding 10,000 pounds per square inch. Careful sizing of the raw materials and thorough mixing to insure uniform distribution of the bond are necessary. If the brick are to be metal encased, this process permits the pressing of the steel jacket onto the brick during the actual forming operation. After forming, all that is necessary is a thorough drying out at less than 500° F to develop maximum cold strength, which often exceeds that of burned basic brick.

Refractory Concrete Products—Refractory concrete is made of sized refractory aggregates, such as calcined flint clay, crushed fireclay brick bats, chrome ore, olivine, or magnesite, and a hydraulic bond, usually calcium aluminate cement. Aggregate-cement ratios may vary from 4 to 1 to 10 to 1. Calcium aluminate cements are employed because of their rapid setting and, more important, their ability to retain their water of hydration, and hence their bonding power, to considerably higher temperatures than Portland cement. Besides the use of such concretes for pouring refractory structures in place, they are often used to cast special shapes, such as burner blocks, on the job.

Electrocast Products—The electrocast process consists of mixing raw materials of high purity in proper proportions, heating them to complete fusion in an electric furnace, and casting in forms, very much as liquid iron or steel is cast. By 1935, many refractory bodies, consisting essentially of fused mullite, had been produced. Since that time, other products, consisting of various combinations of chrome, magnesite, alumina, and zirconia have also been produced. Advantages of the process are almost complete lack of voids, a narrow range of melting, and better control of chemical properties.

B. GRANULAR REFRACTORIES

Raw State—The chief refractories used in the raw state in granular form are clays, chrome ore, and dolomite. Clays coarse enough for this classification are seldom used without further reduction on the job, blast-furnace-taphole clays being the principal application. Granular chrome ores are used in open-hearth front-wall maintenance as ground, or for reheating-furnace hearths and open-hearth doors when plasticized with clay and bonded with sodium silicate. Crushed raw dolomite is used extensively for open-hearth maintenance, principally for building up banks, where its consumption may vary from 10 to as much as 100 pounds per ton of steel produced.

Fired Products—Fired granular products include dolomite, magnesite, and other special basic refractories. Two types of dolomite are recognized in this category. Calcined dolomite, often called single burned, is produced in this country by heating the natural rock to a temperature somewhat above the decomposition temperature of the dolomite or generally to about 2000° F. This

product consists of crushed porous granules which are very active chemically and tend to slake rapidly upon exposure to the atmosphere. Its use is decreasing. Dead-burned, or clinkered dolomite is a product made by calcining or dead-burning dolomite at temperatures of approximately 3000° F, usually in rotary kilns, using iron oxide as a dead-burning agent. The iron oxide forms calcium ferrites in the dead-burning operation. This has an important function in mineralizing and shrinking the refractory in the burning operation and in helping to coalesce the refractory into a monolithic mass in the open-hearth furnace. This is the most satisfactory form in which to use dolomite, but as it is also the most expensive, it is often used in combination with raw or single-burned dolomite.

Magnesite for making furnace bottoms, for furnace maintenance, and for brick making is always used in the dead-burned state, produced, like dolomite, by firing at high temperatures in rotary kilns. While it is the important refractory in making new open-hearth bottoms, its consumption per ton of steel, in contrast to dolomite, varies from about one to two pounds.

Special granular refractories are used principally for the basic ramming and patching materials which have been introduced since 1939. These are carefully-sized, chemically-bonded preparations composed of dead-burned magnesites or magnesia-dolomite mixtures. The latter generally contain a minimum of 60 per cent MgO, the balance being so proportioned between lime and silica as to form a predominance of dicalcium silicate when clinkered in a rotary kiln. To prevent the dusting of dicalcium silicate on cooling, it is necessary to prevent its sudden inversion from the beta to the gamma form. This is accomplished by stabilizing with small amounts of mineralizers such as Cr_2O_3 , B_2O_3 and phosphates, or combinations of these. These special ramming mixtures are now widely used for open-hearth-bottom making, as they are quicker to install, provide more accurate contours, and generally are more uniform in composition, than burned-in magnesite bottoms.

Another group of special granular refractories is the fireclay-base plastic refractories. These consist generally

of sized mixtures of raw and calcined clays, often containing sodium silicate or organic bonds, which are shipped wet or dry for applications where a rammed refractory may substitute for brickwork. Such plastic refractories may also be made with super-duty clays, kyanite, or other high-alumina materials as additions.

The refractory concrete products referred to above are often made from dry, prepared mixtures called **castables**. As these are sold ready for mixing with water, they must be included as granular refractories.

C. FINELY DIVIDED OR PULVERIZED PRODUCTS

Raw State—Clays for brick laying and various bonding purposes are about the only refractory used in steel mills in the raw, finely-divided state. However, large tonnages of such clays covering a considerable range in refractoriness are employed.

Processed Products—These include the various heat-setting and air-setting cements and mortars. They are produced for laying every type of brick, and, hence, may have as many different base materials, the principal ones for steel-plant use being silica, fireclay, chrome, and magnesite. Silica cement, or silica fireclay, is usually made of finely-ground quartzite or sand to which is added about 10 per cent plastic clay and possibly an organic binder, although in a few superior grades, the undesirable introduction of alumina from the clay is avoided by using dolomite or lime bonds with other additions to provide the necessary workability for brick laying. Fireclay cements differ from plastic fireclay refractories chiefly in respect to fineness, as they also consist of mixtures of raw and calcined clays. The heat-setting varieties rely on vitrification for bonding, while the air-setting types usually contain liquid or dry sodium silicate and, hence, may be shipped wet or dry. Chrome and magnesite cements may contain small clay additions for workability, and like fireclay cements, may also contain sodium silicate or organic binders. All prepared cements generally are ground to completely pass a 30-mesh screen with substantial percentages passing a 100-mesh screen.

SECTION 3

PHYSICAL AND CHEMICAL CHARACTERISTICS OF REFRATORIES AND THEIR APPLICATION TO MEET SERVICE CONDITIONS

The foregoing discussions have indicated that a wide variety of refractories, possessing nearly every extreme in characteristics normally used to differentiate between materials, have found a place among the requirements of steel-plant processes. Obviously, these requirements must be equally diversified. Analysis of service conditions in general shows that refractories are required to withstand:

- (1) All ranges of temperature up to 3200° F.
- (2) Sudden changes in temperature—"thermal shock."
- (3) Stresses—mainly compressive, at both high and low temperatures.
- (4) The action of slags, ranging from acid to basic in character.
- (5) The action of molten metals, always at high temperatures and capable of exerting great pressures and buoyant forces.
- (6) The action of gases, including SO_2 , CO, Cl, CH_4 , H_2O , and volatile oxides and salts of metals, even volatile metals. All are capable of penetrating and reacting with the brick. SO_2 may react with oxides

in the brick to form easily-fused salts. CO, penetrating fireclay brick, breaks down to C and CO_2 in the presence of iron carbides, the carbon being deposited in the brick and causing its disintegration. Zinc in the vapor phase has a similar effect. At relatively low temperatures, steam and water vapor exert a slaking effect on dolomite and magnesite refractories. And nearly all refractories are apt to be subjected to, and adversely affected by, volatile iron and alkalis.

- (7) The action of dust in gases, which may be fluxing or non-fluxing, and acid or basic.
- (8) Impact and abrasive forces at both high and low temperatures, as those in the stack of a blast furnace.

At the same time a refractory is being subjected to one or more of the above conditions, it may be required to function as a storehouse for heat, as in checkers; as a conductor of heat, as in the walls of coking chambers; or as an insulator.

With such an array of service requirements it would

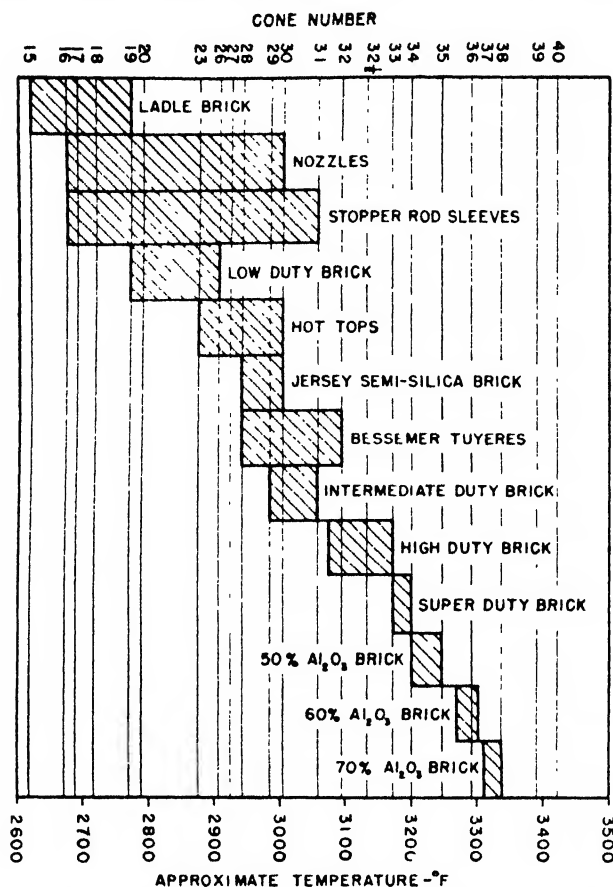


FIG. 8-1. Pyrometric cone equivalent (P.C.E.) of various refractories.

be extremely helpful to have ways of predetermining the suitability of a refractory for a given application. However, it is difficult, or in many cases, impossible, to duplicate these widely variable service conditions in the laboratory. Even were this not the case, time and cost are effective barriers to the development of practical tests which will provide information necessary for positive selection of the proper refractory. Therefore, the standard tests which have been devised, generally within these limitations, primarily provide gages of performance which, while invaluable for quality control, must be judiciously correlated with additional fundamental knowledge of refractory behavior and with service observations before they can be effectively used by consumers. In many instances, information is not available to satisfactorily bridge the gap in this manner, and special tests must be devised.

Thus, by comparison with other materials of construction, refractories appear to suffer in the degree of precision with which they can be applied to fill a certain need. To appreciate this situation it is necessary to know something of the high-temperature behavior of refractories and to recognize that this behavior is the end result of a complex and inextricable combination of physical and chemical factors. Some of these factors are fixed, as they constitute the inherent properties of the materials employed, or of the products of their reaction with other materials in service. Others, mostly physical, may be varied within limits to produce certain desired behavior. In their combined effect on the cold and hot

strength, resistance to spalling, resistance to deformation under hot load, permeability and other measured properties, these factors often oppose each other; hence a particular refractory may represent one or more compromises, with a sacrifice of excellence in one characteristic in order to achieve it in another more important for a given application. In the following discussion of refractory characteristics, these matters are emphasized rather than the details of the tests used in refractory evaluation.

Fusion or Softening Temperature—The fusion of most refractory materials is not clear cut, but a more or less gradual transition from solid to liquid. The amount of liquid that can be tolerated by a refractory and still leave it in a serviceable condition is largely governed by the viscosity of the liquid and the type of crystallization of the solid phases present. For example, fireclay refractories may develop liquid and actually start to soften as low as 1800° F, but due to the high viscosity of the liquid their limiting service temperature may be several hundred degrees higher. An arbitrary procedure has, therefore, been established for gaging the refractoriness of such materials, and is called the **Pyrometric Cone Equivalent**, or P.C.E. test, in which the softening behavior of small cones of the refractory are compared with that of standard pyrometric cones of known time-temperature softening behavior. The P.C.E. is reported as the number of that standard cone whose tip touches the supporting plaque simultaneously with a cone of the refractory being investigated when tested in accordance

with the Standard Method of Test for Pyrometric Cone Equivalent (A.S.T.M. Designation: C-24). The composition of the standard cones and the temperature range covered make the P.C.E. test most applicable to alumina-silica refractories. Figure 8—1 gives the P.C.E. of various refractories and the corresponding softening temperatures when heated under the test conditions. It must be recognized that the end points of pyrometric cones primarily reflect the influence of time and temperature, and hence are reproducible only under identical conditions. A cone used in a kiln fired for a week will soften completely at a temperature which would leave it unaffected in the short P.C.E. test. Fireclay refractories behave similarly, the influence of time being strikingly shown in the case of ladle brick. Normally, maximum service temperatures of fireclay brick are considerably below P.C.E. temperatures, but low-P.C.E. ladle brick are successfully used at temperatures 300° F above their P.C.E. temperatures because the duration of this exposure is seldom more than an hour at a time.

It is not particularly disturbing that the P.C.E. test is not equally satisfactory for silica and basic brick, as other tests provide a far more realistic appraisal of refractoriness. The melting point of pure silica (3140° F) is certainly not impressive when compared to that of other materials, and yet silica brick, containing 95 to 98 per cent SiO_2 , so far have successfully resisted replacement by more refractory materials in open-hearth roofs attaining temperatures frequently exceeding 3000° F. This is because silica brick approach the sharp melting behavior of a pure compound, and, aided by a strong crystalline structure, remain rigid to within a few degrees of the temperature of complete fusion.

On the other hand, the main constituents of basic brick (magnesite, chrome, chrome-magnesite and forsterite) all melt at temperatures far above any encountered in steel plant service. Periclase (MgO) melts at 5070° F, chrome spinel, $(\text{MgO}, \text{FeO}) \cdot (\text{Cr}_2\text{O}_3, \text{Al}_2\text{O}_3, \text{Fe}_2\text{O}_3)$, above 3600° F, and forsterite ($2 \text{MgO} \cdot \text{SiO}_2$) at 3470° F. However, the raw materials for basic brick contain other minerals considerably lower in refractoriness which can form fluid liquids at temperatures as low as 2400° F, and as these materials are the matrix of most basic brick, they largely govern the melting behavior of the brick as a whole, particularly as the basic minerals lack the strong, interlocking crystalline structure of the silica and alumina-silica minerals. Fortunately,

these low-refractory minerals are subject to control, so that their detrimental effects can be minimized or even eliminated by corrective additions, special firing treatments, and proper distribution.

Porosity and Permeability—Unless expensive measures are resorted to, all manufactured refractories will contain a certain amount of voids, the amount, size, and continuity of which have important influences on refractory behavior. The apparent porosity indicates the percent of the total volume which is open pore space, and hence is a measure of the area of surface available for reaction with slags and gases. The total porosity is the percentage of the total volume which is voids, whether open or closed, and thus, depending on the nature of the material, method of manufacture and degree of burn, may only slightly exceed the apparent porosity or be more than twice as great. The effect of porosity on various refractory properties will be discussed later, but in general, increasing the porosity adversely affects the cold strength, resistance to deformation under hot load, heat capacity, thermal conductivity and resistance to attack by gases and slags. Within limits, increasing porosity will improve spalling resistance, but as this can also be achieved in brick of low porosity, it is evident that as a general rule, far more can be gained in performance from refractories possessing the lowest possible porosity. The porosities typical of various types of refractories are given in Table 8—II.

Porosity bears but little relation to permeability, which is a measure of the rate of diffusion of liquids and gases through the refractory, and thus is governed by the size and number of connected pores or channels which are continuous from one side of the refractory to the other. Permeability to liquids will increase with increasing temperature as the viscosities of the liquids will decrease, while just the opposite is true of gases, whose viscosities increase at higher temperatures. While permeability is not commonly determined, and no standard tests exist in this country, it is undoubtedly important in such applications as blast-furnace linings where disintegration frequently occurs as a result of penetration by carbon monoxide and subsequent carbon deposition within the brick.

Bulk Density and True Specific Gravity—Like porosity, the bulk density of refractories is used as an indication of the voids/solids ratio, but is meaningless in this sense without reference to the true specific gravity of

FIG. 8—2. Typical curves of linear expansion of various types of refractories.

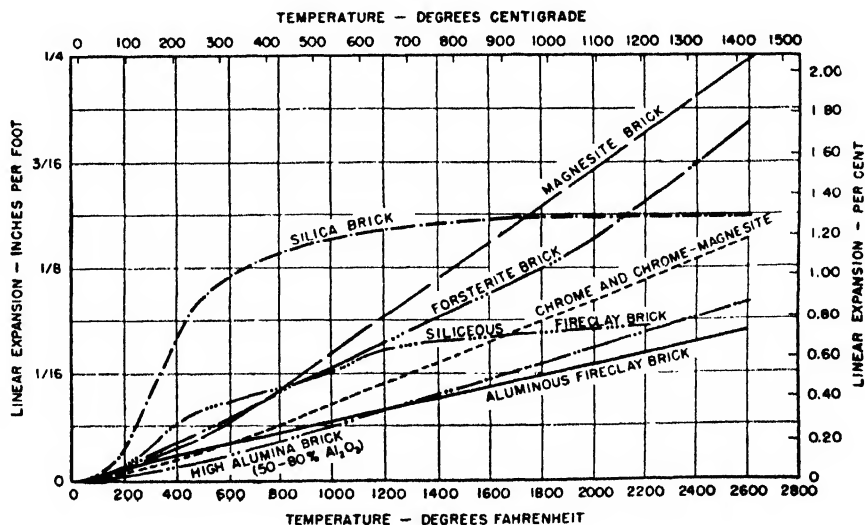


Table 8—II. Some Typical Physical and Thermal Properties of Refractories

Refractory	Apparent Porosity %	True Specific Gravity	Bulk Density		Cold Strength		Behavior Under 25 Psi Load			Spalling Resistance
			Lbs. per Cu. Ft.	Oz. per Cu. In.	Crushing on End Psi	Modulus of Rupture Psi	Percent Deformation	When Held 1½ Hrs. At	Temp. of Failure (° F)	
Silica Brick:										
Conventional	21-30	2.30-2.38	102-115	0.94-1.06	1000-3700	400-1500	2950-3020	Below 1100° F— Poor
Super Duty ..	22-26	2.33-2.35	105-115	0.97-1.06	1000-3000	350-800	3060-3090	Above 1100° F— Good
Fireclay Brick:										
Super Duty—Normal Burn ..	13-20	2.65-2.75	136-151	1.26-1.40	900-4000	600-1300	2.5-9.0	2640° F	Good to very good
Super Duty—Hard Burn	8-15	2.69-2.76	141-151	1.31-1.40	2500-4500	900-1300	0.7-4.0	2640° F	Poor to very good
High Duty—Normal Burn ...	12-25	2.60-2.70	128-142	1.18-1.32	1000-6000	400-2000	1.0-10.0	2460° F	Fair to good
High Duty—Hard Burn	10-17	2.70-2.78	134-146	1.24-1.35	3000-8000	1300-2500	2.0-5.0	2640° F	Poor to good
High Duty—Siliceous:										
New Jersey	26-30	2.60-2.70	116-121	1.08-1.12	1800-2300	450-600	0.5-1.5	2460° F	Very good
Texas	21-25	2.60-2.70	125-132	1.16-1.22	1500-2500	600-1100	0.5-1.5	2460° F	Good
Intermediate Duty	18-21	2.55-2.65	128-140	1.19-1.30	1500-4000	900-1600	2.0-6.0	2460° F	Fair
Low Duty	18-21	2.55-2.65	126-145	1.17-1.34	2500-4500	900-1500	2.0-10.0	2460° F	Fair
Ladle Brick	10-18	2.64-2.67	130-146	1.20-1.35	3000-12000	Fair
High Alumina Brick:										
50% Al ₂ O ₃	20-24	2.75-2.85	125-135	1.16-1.25	2500-4000	1000-1500	4.0-6.0	2640° F	Good
60% Al ₂ O ₃	17-30	2.90-3.05	134-150	1.24-1.39	2000-5000	800-1300	1.0-5.0	2640° F	Fair—very good
60% Al ₂ O ₃ (Kyanite Base) ..	20-25	140-150	1.29-1.39	4500-6000	1100-1400	0.5-1.8	2640° F	Very good
70% Al ₂ O ₃	24-36	3.15-3.25	122-149	1.13-1.38	4000-6000	1300-1600	2.0-4.0	2640° F	Fair
Basic Brick:										
Forsterite	18-27	3.30-3.40	154-164	1.42-1.52	1500-4000	450-800	2890-3050	Fair
Chrome	18-25	3.90-4.10	185-205	1.71-1.90	3000-6000	1200-2000	2350-2650	Poor
Magnesite	18-22	3.40-3.60	185-200	1.71-1.85	6000-10000	1200-2000	2600-2800	Poor
Magnesite—96% MgO Burned	19-24	3.54-3.56	175-180	1.62-1.66	8600	2300	3170	Poor
Chrome-Magnesite:										
Burned	18-23	3.84-4.10	180-195	1.66-1.80	2000-5600	900-1500	2950-3060	Fair
Chemically Bonded	3.90-4.10	190-205	1.76-1.90	2000-5000	750-1200	2700-2850	Good
Magnesite-Chrome:										
Chemically Bonded	3.60-3.80	185-205	1.71-1.90	2000-5000	750-1000	2700-2850	Good
Carbon Block	19-25	1.96-1.97	94-100	0.87-0.92	3500-10000	0	3100° F	Very good

the refractory. Thus, the bulk density of a chrome brick having a true specific gravity of 4, but a porosity of 25 per cent is still much greater than that of a dense fireclay brick whose porosity is 10 per cent, but whose true specific gravity is only 2.6. Bulk density may be expressed in a number of ways, those most common being grams per cubic centimeter (bulk specific gravity), pounds per cubic foot, or ounces per cubic inch. Being thus the mass of refractory material in a given volume, an increase in the bulk density of a given refractory increases the strength, volume stability, heat capacity, resistance to slag penetration, and generally, resistance to structural spalling, or spalling resulting from vitrification caused by fluxing with dusts and slags.

The true specific gravity of a fired refractory may differ substantially from that of the raw materials from which it is made, owing to conversion of the mineral constituents. For example, quartzite with a specific gravity of 2.65 converts on heating to cristobalite and tridymite with specific gravities, respectively, of 2.32 and 2.26, so that the resultant specific gravity of fired silica brick reflects the degree of conversion, and is a most reliable test for adequacy of firing treatment. The conversion of kyanite to mullite is another example. Table 8-II shows typical bulk densities and specific gravities of various refractories.

Thermal Expansion and Volume Changes—From the early steps in manufacture throughout their service life, refractories are subject to various types of volume changes. As these changes may affect product uniformity and service performance they are of concern to both manufacturer and user. Most familiar of these volume changes is the reversible thermal expansion, or dilation that practically all materials exhibit on heating and lose on cooling. Figure 8-2 shows typical curves of linear expansion for various types of refractories. Those which have a uniform expansion rate generally present the least difficulties when furnace temperatures fluctuate widely, and of these, those with the lowest total expansion, as a general rule, are less subject to thermal spalling. Such is the case with fireclay refractories. Silica brick, as previously mentioned, possess varying proportions of the silica minerals cristobalite, tridymite and unconverted quartz, depending on the heat treatment. As each of these silica forms has its own characteristic expansion with sudden changes accompanying inversions from low- to high-temperature forms, the curve for a silica brick will reflect its mineral composition to some extent. However, most of the expansion from all sources takes place below 1060° F, hence, if care is taken in heating and cooling below dull-red heat, silica brick behave admirably. In fact, one of the major problems in the application of basic brick to open-hearth roofs arises from the continuing high-temperature expansion of basic brick, which, while uniform, causes a "working" of such roofs with temperature changes in ranges that have no effect on silica roofs.

Permanent volume changes may be either expansion or contraction. The latter is first encountered in drying molded refractories, particularly clay brick, and must be accurately controlled to make a uniform product. Further shrinkage may take place on firing. Finally, if the refractory is used at temperatures above that at which it was fired, shrinkage may occur, or it may be caused by fluxing with dusts or slags and subsequent vitrification. Permanent expansion may also take place on firing refractories. Thus silica brick, because of the previously indicated decrease in specific gravities of the higher-temperature forms of silica, will expand about 13 per cent on firing. Some fireclay brick may also exhibit what has been termed **secondary expansion** on

firing or reheating. This has been attributed to an ex-foliation of the clay grains, and is to be distinguished from the bloating or vesicular structure development brought about by gas evolution and expansion in an overheated and hence pyro-plastic clay body. While generally indicative of misapplication of refractories, the latter condition is actually desired in ladle brick as it is conducive to the formation of a monolithic, steel-tight ladle lining. In service, refractories may also permanently expand. The action of alkalis on fireclay refractories to form alkali-alumina silicates with an accompanying expansion of about 12 per cent causes surface peeling of such refractories, as sometimes observed in open-hearth and blast-furnace-stove checkers. The expansion or "bursting" of basic brick on absorption of iron oxide at high temperatures is another example.

Cold Strength—The mechanical strength of refractories, as determined by modulus of rupture or cold crushing tests, is generally far more than that required to meet any dead loading encountered in construction. Nevertheless, we are interested in strength. Refractories must be handled a great deal before they are finally put in service, and it is naturally desired that they survive this handling with a minimum of breakage, abrasion and lost corners. Soft-burned clay brick, particularly if dry pressed, may be very friable and easily broken but still possess a crushing strength above 1000 pounds per square inch. As cold strength reflects the heat treatment a refractory receives in manufacture, and as this heat treatment affects porosity, bulk density, refractoriness under load, and abrasion resistance, considerable general information on any specific burned refractory can be obtained from strength tests with sufficient correlation. For this reason, cold-strength requirements are often used in the specification of refractories. The ranges in crushing strength and modulus of rupture for various refractories are shown in Table 8-II.

Strength and Behavior Under Load at High Temperatures—The cold strength of refractories is governed mainly by the amount of glassy bond present, the hot strength by the changes brought about in that glassy bond by heat. Generally, with increasing temperature, the glass gradually becomes a viscous liquid, although the initial softening temperature and the temperature range of viscous flow may vary widely between different types of refractories. Thus, to different degrees, all refractories at some temperature exhibit plastic flow, and if subjected to pressure, the subsequent behavior will depend on the amount of pressure and the duration and rate of its application, on the amount and viscosity of the liquid present, and on the crystalline structure of the solids. Loaded heavily and rapidly as in normal compression testing, all refractories will fail in shear until temperatures are reached at which they contain so much liquid of such low viscosity that almost instant deformation occurs. Under such conditions, the compressive strength or modulus of rupture may not change materially below temperatures of initial liquid formation, but will then rapidly decrease.

Of far more significance, however, is the behavior under light, sustained loads comparable to those encountered in service, and numerous tests have been devised to measure the amount or rate of deformation occurring under specified conditions of time, temperature and stress. While prolonged tests are especially informative to the refractory user, all standard tests are of relatively short duration, although increasing attention is being given to the longer tests, particularly those aimed at the determination of the almost imperceptible flow or "creep" of refractories at temperatures immediately

below and above those of the initial liquid formation.

Fireclay refractories have the longest softening range and are most susceptible to plastic deformation. Under standard test conditions, consisting of heating a 9-inch brick on end under a 25-pound per square inch load in $4\frac{1}{2}$ hours to test temperature (2460°F for high-duty brick; 2640°F for super-duty) and holding for $1\frac{1}{2}$ hours, the temperature of initial deformation and the total amount of deformation will depend on the flux content, the degree of burn, and the porosity. Impurities such as lime, magnesia, iron oxides and alkalis, will lower the temperature of initial liquid formation and the viscosity of that liquid, and hence increase deformation. Hard-burned brick will show much better resistance to load than a soft-burned brick made of the same clays, due in part to greater mullite development, but primarily due to the fact that harder firing has developed a more refractory glassy phase which forms a liquid of initially higher viscosity at test temperatures. Increasing porosity decreases the ability of a particular brick to resist deformation under load, as the voids provide opportunity for a brick to compress on itself and also decrease the amount of refractory available in a given volume to resist pressure. The temperature of initial deformation under load will vary widely for a given brick with deviations from standard test conditions. For example, a fireclay brick deforming initially at 2250°F in the standard test might not deform until 2350°F with a lighter load or a faster heating rate, and conversely, might deform as low as 2000°F with a much heavier load or with that temperature maintained for several hours.

The behavior of high-alumina brick under load is similar to that of fireclay brick, although the temperature range of plastic deformation will be higher the greater the alumina content.

Due to their strong, interlocking crystallization and purity, silica brick show little evidence of plastic deformation under load, remaining rigid under a load of 25 or even 50 pounds per square inch until they fail by shear at temperatures varying from 3000°F to 3100°F . The latter temperature, however, is attainable only by the super-duty types containing a maximum of 0.5 per cent of the impurities Al_2O_3 , TiO_2 , and alkalis. Because of their strong crystalline structure, silica brick may be more than 30 per cent molten at the temperature of failure.

Basic brick, like silica brick, generally fail under hot load by shearing, but lacking the intercrystallization characteristic of silica minerals, practically all resistance to failure resides in the liquid phases present. In chrome and magnesite brick, these liquids may be low in refractoriness and viscosity and failure will occur in the range of 2400°F to 2500°F . However, with the development of a refractory forsteritic matrix, as in burned forsterite and chrome-magnesite refractories, failure may not occur until 2950°F to 3050°F under a load of 25 pounds per square inch in a rising temperature test. Unburned basic brick usually fail 100°F to 200°F below the temperatures of failure of burned brick of the same composition.

A notable exception to the general rule that magnesite brick are weak in load-carrying ability, is the recently-developed high-purity sea-water-magnesite brick containing in excess of 96 per cent MgO . Such brick, even if unburned, normally will not shear under load, and have been found to deform less than 3 per cent at temperatures in excess of 3300°F .

Carbon refractories show little or no loss in strength and no deformation under load throughout the range of steel-plant temperatures.

Typical load test data for various refractories are shown in Table 8—II.

Heat Capacity and Thermal Conductivity—The heat capacity of a refractory at a given temperature is a function of its bulk density and its specific heat at that temperature. The thermal conductivity is a measure of the rate of heat transmission through the refractory. Both properties are increased with increasing bulk density and hence decreased with increasing porosity. As refractories may be used to confine, store, or transmit heat, the thermal conductivity desired will vary widely with the application. While a wide range in conductivity is available in commercial refractories, if the insulating types are included, it is not always possible to use that refractory whose conductivity is best suited to the job. Thus in the majority of cases, it is desired to confine heat, but it was previously noted that in a majority of cases high bulk density was also desirable from a service standpoint. With refractories used for heat storage and transmission, such as checker brick, no conflict in properties exists, as high density is desired both for stability and increased heat storage and conductivity.

Typical conductivity curves for general types of refractories are shown in Figure 8—3. It will be noted that

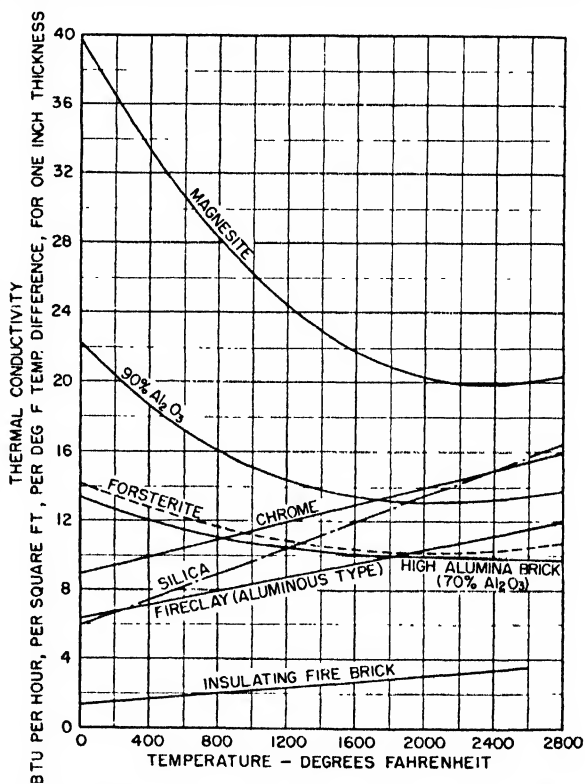


FIG. 8—3. Typical thermal conductivity curves for general types of refractories.

at the temperatures common to steel-plant operations, the differences in conductivity found in these refractories at low temperatures are greatly reduced. The accuracy of these higher-temperature determinations is often questionable, although standardization of test methods in recent years has greatly improved the reproducibility of results from different laboratories. However, actual heat flow through furnace walls in service still may differ considerably from calculations. Thus the conductivity will change with the alterations in structure and composition of the hot face; the rate at

which heat is carried away from the cold face affects the flow through the wall; and the gas pressure within the furnace has a great effect, not only on the heat flow but on refractory life as well, as positive pressures will drive heat in, increasing the depth of vitrification and fluxing.

Resistance to Thermal Shock (Spalling)—One of the major causes of refractory destruction is spalling, which is the breaking away of pieces of refractory from the hot face, thus exposing fresh surfaces. Spalling may result from too-rapid expansion or contraction of the hot face of the refractory with sudden temperature changes, called **thermal spalling**. Spalling also may result from changes in the hot face brought about by flux absorption or vitrification which set up zones in the brick which differ in expansion and sensitivity to thermal shock from the original brick, or shrinkage may occur, so that pieces separate; this is termed **structural spalling**. A third type, **mechanical spalling**, may result from mechanical abuse, as in removing slag accumulations from refractory surfaces, or may be caused by shifting loads and stresses, particularly in arches, so that refractories are pinched and crack off.

The properties of refractories which most influence their resistance to spalling are thermal expansion, elasticity (ability to yield to stress without rupture), and strength. Because of their low thermal expansion, fireclay refractories as a class have the best spalling resistance, but since the elasticity and strength are strongly affected by the degree of burn, particle size and porosity, wide variations exist. Thus a soft-burned,

coarse-grained, porous brick has higher elasticity and higher thermal-spalling resistance at low temperatures than a hard-burned dense brick, but may be expected to fail from structural spalling more readily at high temperature, due to greater shrinkage and flux absorption.

Basic refractories have high expansion coefficients and vary from very poor to fair in spalling resistance, magnesite and chrome being the poorest, and chromemagnesite being the best, particularly if chemically bonded.

As indicated earlier, the spalling resistance of silica brick varies from very poor to excellent, depending on whether they are subjected to temperature fluctuations below or above 1100 °F, respectively.

Abrasion Resistance—The resistance of refractories to abrasion and erosion is primarily a function of the strength. As strength is governed by degree of burn, some correlation also exists between abrasion loss and porosity and bulk density. While no standard hot-abrasion test has been adopted, there are some indications that the strength-abrasion loss relationship holds at higher temperatures, in which case resistance to abrasion might not be expected to change appreciably below 2000 °F, but to decrease at higher temperatures. Variations of the standard rattler test for paving brick are sometimes used in refractory testing, and more recently, an accelerated test employing a sand blast has been devised. For fireclay refractories, this test indicated a rapid decrease in abrasion loss with small increases in modulus of rupture to 1700 pounds per square inch, with further increases in strength being far less effective.

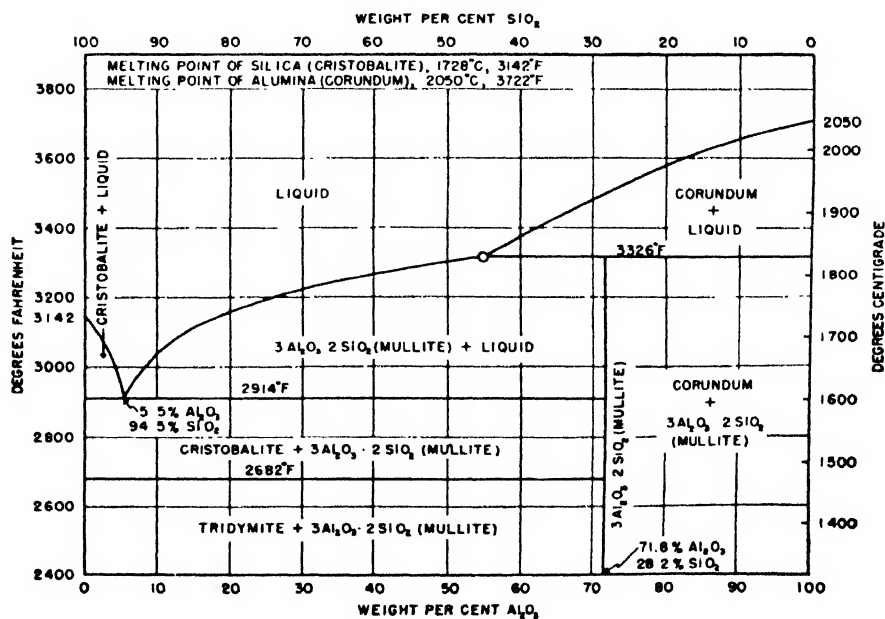
SECTION 4

REACTIONS AT ELEVATED TEMPERATURES

Discussions of the high-temperature behavior of refractories thus far have tended to emphasize the influence of various physical factors, and have given but little indication that behavior also reflects the working of immutable laws governing the high-temperature reactions occurring not only within the refractories themselves but between refractories and oxides encountered

in service. Until comparatively recent years, little was known of these reactions, and improvements in refractory products were slow, as they derived primarily from trial and error, and were generally physical in nature. Today there are available an ever-increasing number of phase-equilibrium diagrams which have proved invaluable guides to the production of superior refractories

FIG. 8-4. Phase diagram of the Al_2O_3 - SiO_2 system.



and to the correction of service difficulties. An indication of the importance of such data is found in the studies on the steel-plant refractory-oxide systems being carried out at the Pennsylvania State University under the sponsorship of the American Iron and Steel Institute. However, it should be recognized that these diagrams are not without limitations in their use to predict or explain refractory behavior. For example, the various systems have been explored with pure oxides and represent equilibrium conditions, while refractories are rarely pure and seldom in equilibrium, either as manufactured or in service. Because of this complex nature of refractories, information is often needed on reactions involving so many oxides that the usefulness of systems involving no more than three oxides is minimized. Finally, and most important, the diagrams give no information on such significant matters as viscosity of the liquids formed or the rate at which reactions proceed. Since it will be shown that under certain conditions many of the commonly-used refractories apparently can be liquefied to the point of destruction at temperatures encountered in steel-plant service, we are often vitally concerned in the rate at which reactions progress toward equilibrium and failure. As excellent compilations of diagrams have been published (see references at end of chapter), only a few are reproduced here.

Figure 8-4 shows the $\text{Al}_2\text{O}_3\text{-SiO}_2$ system which applies to silica, fireclay, and high alumina refractories. It will be noted that the lowest temperature at which any liquid is developed in the system is 2914°F , while for those compositions more aluminous than mullite ($3\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$) or above 71.8 per cent Al_2O_3 , no liquid is developed below 3326°F . A eutectic occurs at the composition 94.5 per cent SiO_2 and 5.5 per cent Al_2O_3 . It is evident that if bonding and other manufacturing difficulties could be overcome, quite good refractories could be made of the pure oxides, even in the high-silica range. In fact, the semi-silica type brick containing about 80 per cent SiO_2 represent a practical approach to this condition. Such brick, because of their low impurity content, are superior in load-carrying ability to many far more aluminous brick having a higher PCE. The pronounced effect of the impurities present in most commercial refractories in this system can be appreciated by comparing the temperatures at which they deform initially under load with the initial liquid temperatures of 2914°F or 3326°F indicated in the diagram. The oxide impurities largely responsible for lowering the refractoriness of fireclay refractories are CaO , MgO , FeO , Na_2O , and K_2O , and these are also encountered in service.

Figure 8-5 shows the $\text{FeO-Al}_2\text{O}_3\text{-SiO}_2$ system. Here

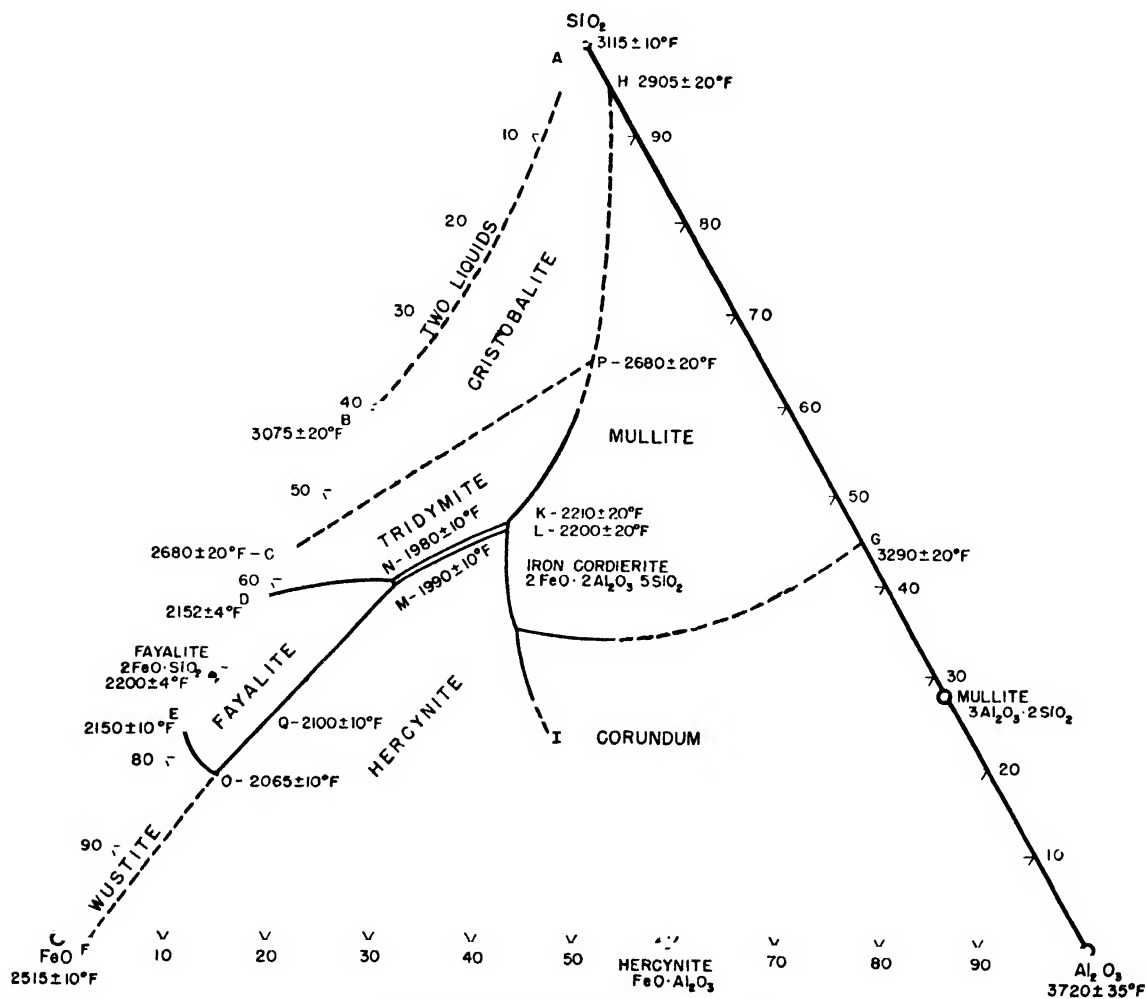
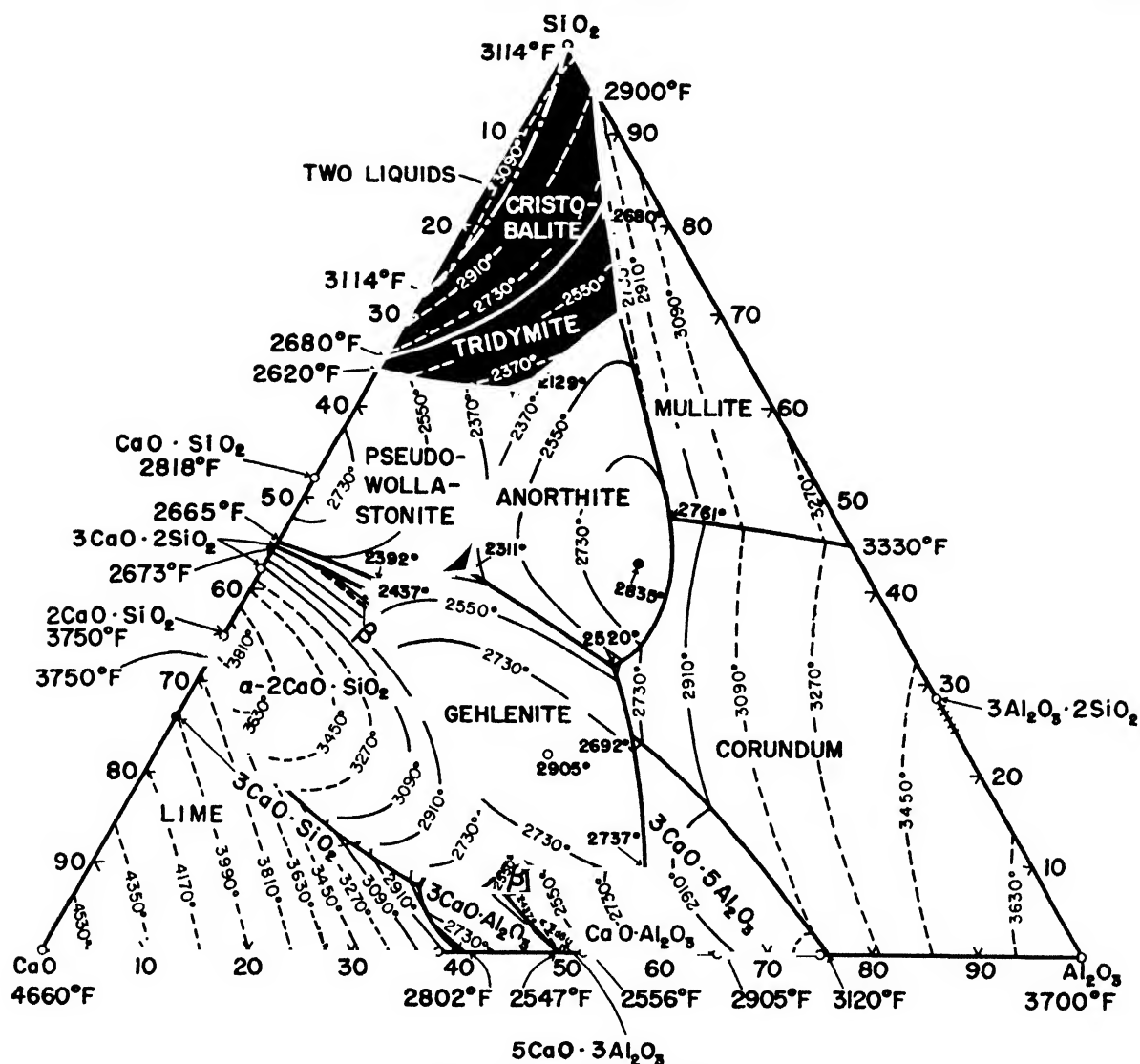


FIG. 8-5. The $\text{FeO-Al}_2\text{O}_3\text{-SiO}_2$ system.

FIG. 8-6. The $\text{CaO}-\text{Al}_2\text{O}_3-\text{SiO}_2$ system.

we see that with unlimited amounts of iron oxide available for reaction with any but the purest end members of the $\text{Al}_2\text{O}_3-\text{SiO}_2$ series of refractories, the formation of some liquid can be expected even below 2000°F , and that very damaging amounts will be formed at the higher temperatures common to steel processes. This is particularly true as iron-oxide-bearing liquids are characteristically very fluid.

Figure 8-6 is the diagram of the $\text{CaO}-\text{Al}_2\text{O}_3-\text{SiO}_2$ system, which is most applicable to reactions of fireclay refractories with blast-furnace slags, but capable of supplying some information on other reactions involving refractories, as those occurring with the more complex open-hearth slags, or in the bond formed in magnesite brick. This system has also been useful in predicting behavior of silica brick, which will be discussed later.

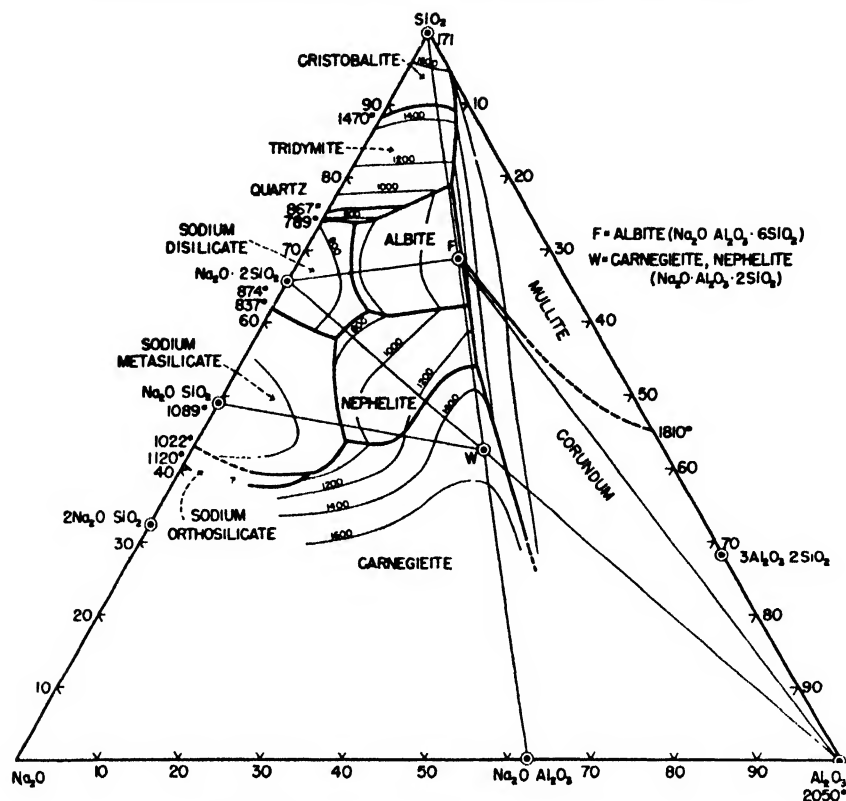
The reactions of alkalis with alumina-silica refractories have recently been clarified by the publication of both the $\text{Na}_2\text{O}-\text{Al}_2\text{O}_3-\text{SiO}_2$ and the $\text{K}_2\text{O}-\text{Al}_2\text{O}_3-\text{SiO}_2$ systems by the Geophysical Laboratory. These systems are reproduced in Figures 8-7 and 8-8. It is evident that the refractoriness of alumina-silica refractories will be

seriously affected by very small amounts of Na_2O , less than 1 per cent being sufficient to lower the temperature of initial liquid formation to less than 2000°F , while approximately 10 per cent is sufficient to completely liquefy the more siliceous alumina-silica compositions at 2200°F . K_2O has a similar effect in amounts up to 10 per cent, but whereas Na_2O continues to lower the melting point with greater additions, the reverse is true of K_2O additions in the range of 10 to 20 per cent.

Small percentages of impurities are far more critical in silica brick than fireclay brick, and alkalis are the worst offender, although fortunately they seldom occur in amounts greater than 0.3 per cent. The percentages of Na_2O , K_2O and Al_2O_3 required to lower the melting point of pure silica from 3140°F to 3050°F are, respectively, 1.4, 1.9 and 3.1 per cent.

Another very serious fluxing agent for fireclay refractories is MnO , as can be seen in the diagram of the $\text{MnO}-\text{Al}_2\text{O}_3-\text{SiO}_2$ system in Figure 8-9. It will be noted that this system is quite similar to the $\text{FeO}-\text{Al}_2\text{O}_3-\text{SiO}_2$ system.

Because the raw materials for silica brick lack both a

FIG. 8-7. The $\text{Na}_2\text{O}-\text{Al}_2\text{O}_3-\text{SiO}_2$ system.

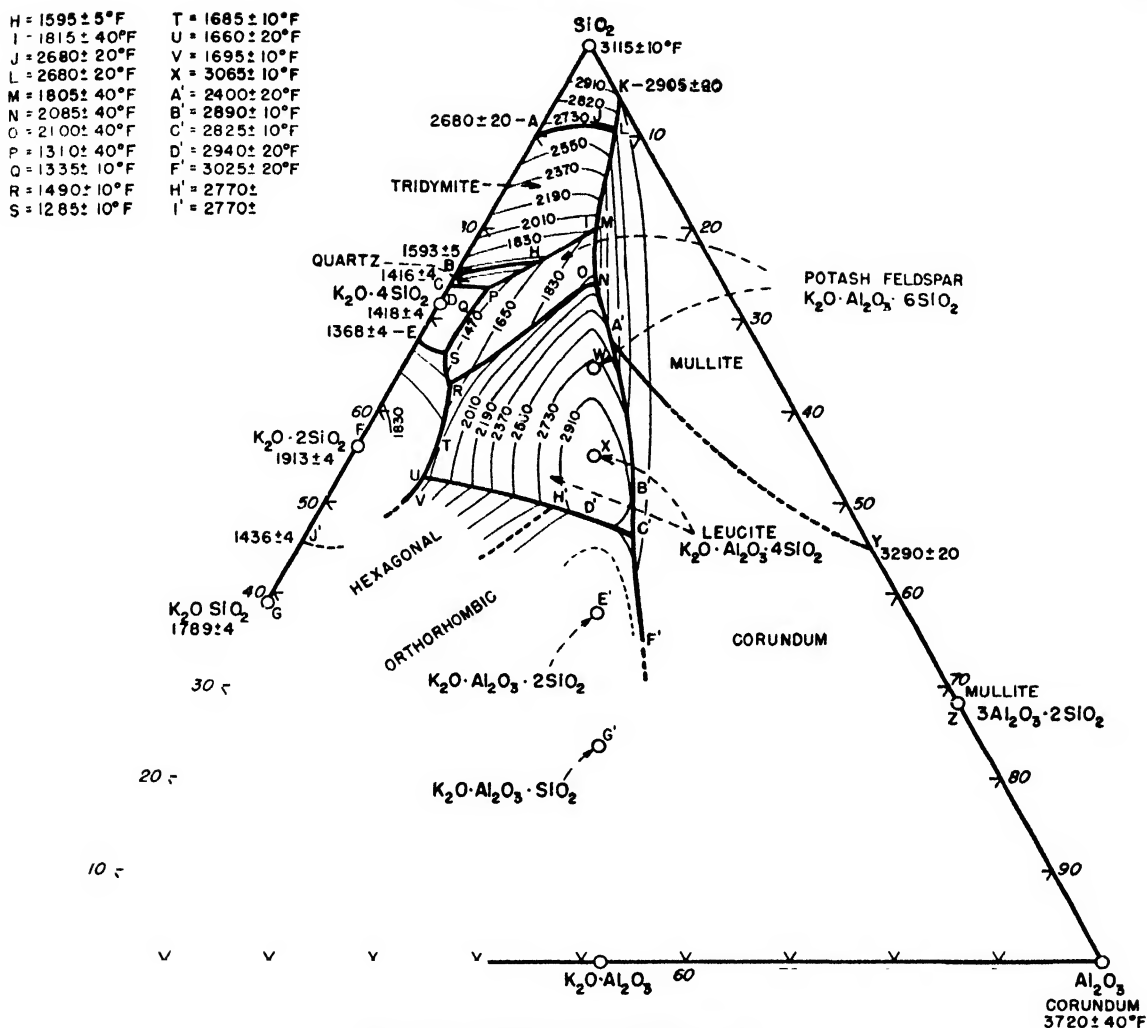
natural bond and a high melting point, care must be taken that the required bonding addition has the minimum effect on refractoriness. Figure 8-10 shows the $\text{CaO}-\text{SiO}_2$ system and explains why lime is universally used for this purpose. With additions of CaO to SiO_2 , the melting temperature remains unchanged between 1 and 27.5 per cent CaO , due to the formation of two immiscible liquids. We have already seen that no such phenomenon occurs in the $\text{Al}_2\text{O}_3-\text{SiO}_2$ system, and by referring again to the $\text{CaO}-\text{Al}_2\text{O}_3-\text{SiO}_2$ system, we find that only a small amount of Al_2O_3 is required to destroy the $\text{CaO}-\text{SiO}_2$ immiscibility. In fact, the effect of minor increments of Al_2O_3 on the liquid development of silica brick is such that the temperature of failure under a load of 25 pounds per square inch will decrease approximately 10°F for each 0.1 per cent increase in Al_2O_3 in the 0.3 to 1.2 per cent range of Al_2O_3 between super duty and conventional silica brick. Figure 8-11 of the $\text{FeO}-\text{SiO}_2$ system shows that FeO , like CaO , also forms two immiscible liquids when added to SiO_2 , thus greatly increasing the tolerance of silica brick for FeO . Furthermore, as with CaO , a small amount of Al_2O_3 can eliminate this immiscibility.

As atmospheric conditions in steel-plant furnaces may range from highly reducing to highly oxidizing, the form of iron oxides present may vary from FeO to Fe_3O_4 . Accordingly, Figure 8-12, showing the system $\text{FeO}-\text{Fe}_3\text{O}_4-\text{SiO}_2$, as recently determined by Muan in the AISI-sponsored studies at Pennsylvania State University, is of considerable importance. Thus, it is seen that the lowest-melting liquids occur from reaction of FeO and SiO_2 , and that at temperatures in the range of 2650° to 3030°F less liquid, and a less siliceous liquid, will be produced with either Fe_3O_4 or Fe_2O_3 than with FeO ,

due to the greater extent of the two-liquid region under oxidizing conditions. Hence, under oxidizing conditions, an open-hearth silica roof will have more tolerance for iron oxide, and therefore will be more durable.

The simple binary system $\text{CaO}-\text{MgO}$ shown in Figure 8-13 contains a eutectic approximating the composition of dead-burned dolomite, but it is evident that the whole series, within itself, is very refractory. However, as no binary compound is formed, dead-burned dolomite exists as a mixture of the two oxides, each free to act more or less independently. From a refractory standpoint, MgO behaves admirably, but the far more chemically active CaO is a decided trouble-maker in two respects. First, CaO is very reactive with iron oxides, forming low-melting calcium ferrites, and hence is readily fluxed by high-iron-oxide slags. Second, it imposes a major limitation on dolomitic refractories by its tendency to hydrate, regardless of the degree of previous heat treatment. Hence, if dolomite is to be used for other than a granular hearth-maintenance material, the lime must first be stabilized by combination with other oxides, usually silica, which forms dicalcium silicate.

It is evident that the refractory value of dolomite resides largely in the oxide MgO , and this is likewise true of the magnesium silicate olivine, and of course, the various types of magnesites. The chief characteristic of MgO accounting for its popularity as a refractory for basic steel processes is its great tolerance for iron oxide. Figure 8-14 shows that MgO and FeO form a continuous series of solid solutions which are amply refractory even when their composition is more than half FeO . This situation prevails under reducing conditions. Under oxidizing conditions MgO reacts to form magnesioferrite ($\text{MgO} \cdot \text{Fe}_2\text{O}_3$) which contains 80 per cent Fe_2O_3 .

Fig. 8-8. The $K_2O-Al_2O_3-SiO_2$ system.

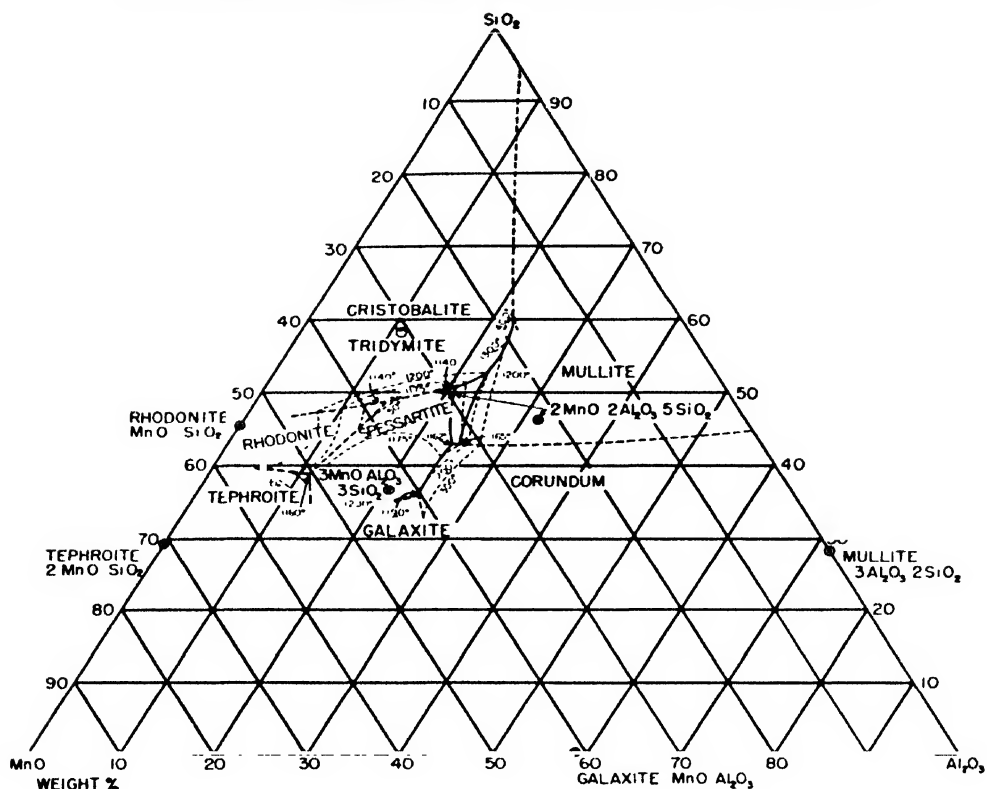
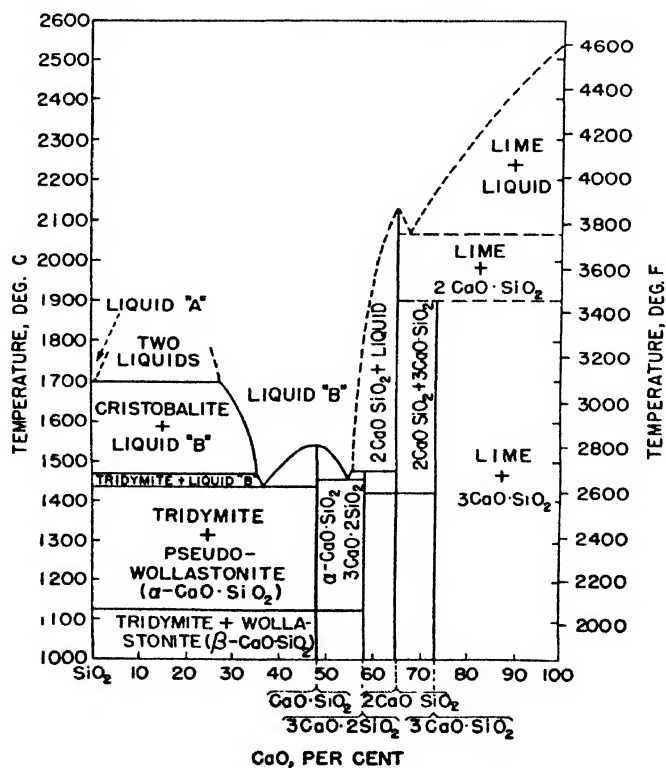
but melts only above 3200° F, and is in solid solution with MgO with lower percentages of Fe_2O_3 .

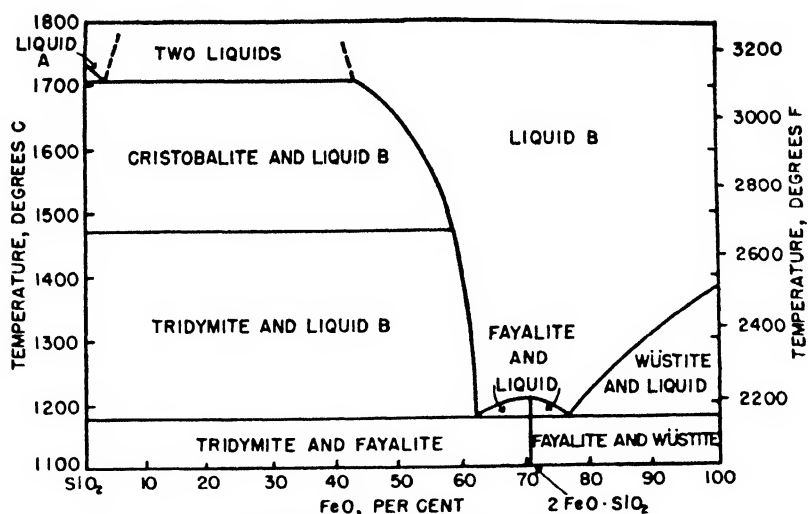
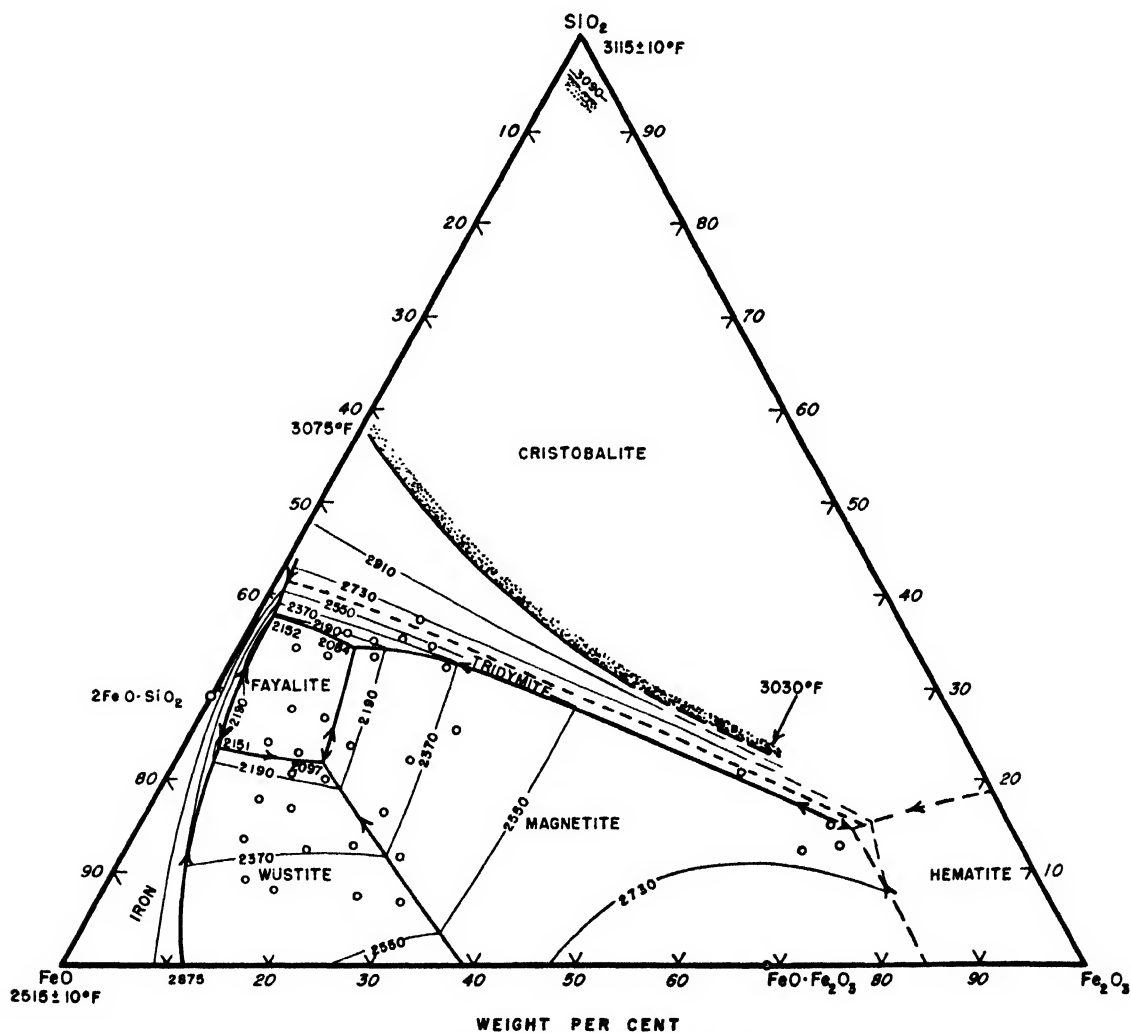
However, magnesia-bearing refractories contain other oxides, such as CaO and SiO_2 , which are also encountered in service; hence reactions occurring in the CaO-MgO- SiO_2 system shown in Figure 8-15 are important in the manufacture and use of most basic refractories. Many compounds, differing widely in refractoriness and stability are found in this system, their occurrence in refractories being governed largely by the lime-silica ratio; their desirability as a refractory constituent being governed by the intended service. Thus, aside from periclase (MgO) three of the most important compounds are forsterite ($2 MgO \cdot SiO_2$), monticellite ($CaO \cdot MgO \cdot SiO_2$) and dicalcium silicate ($2 CaO \cdot SiO_2$), with melting points of 3470° F, 2734° F (incongruent), and 3865° F, respectively. Generally, forsterite is the most desired bonding constituent in basic brick as it is stable and provides ample refractoriness and load-carrying ability. In high-magnesia refractories, forsterite is formed when lime is absent or the molecular ratio of lime to silica is less than 1. If, under these conditions silica is increased at the expense of MgO, low-melting liquids form; hence silica brick will react in contact with magnesite at about 2800° F, and magnesite must be added to some magnesia-deficient olivines used in forsterite-brick manufacture

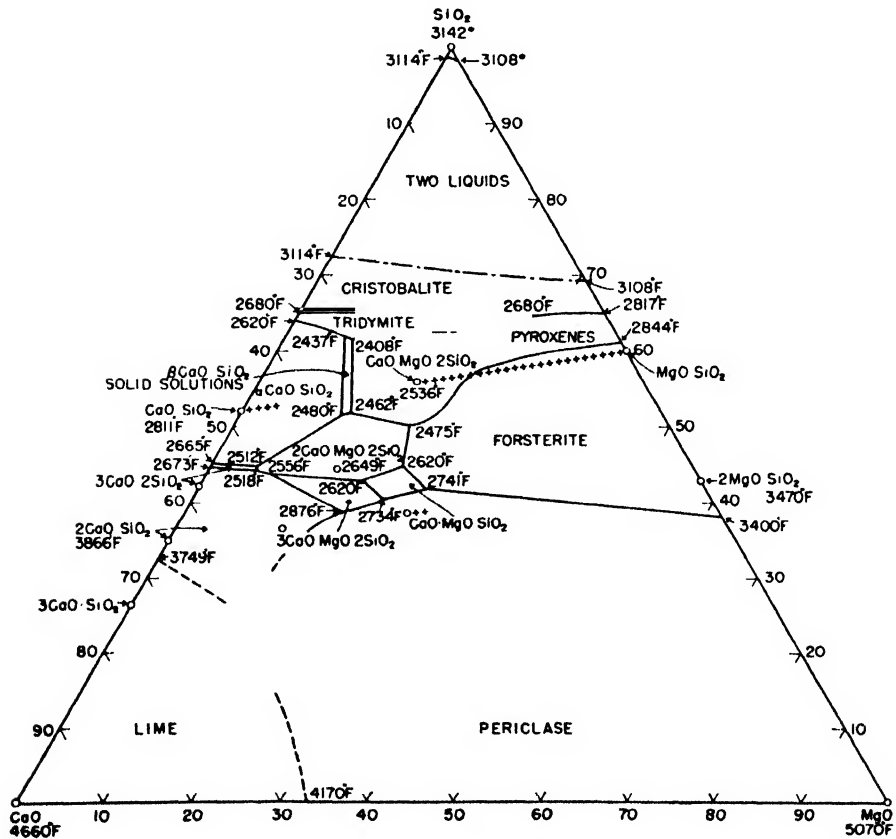
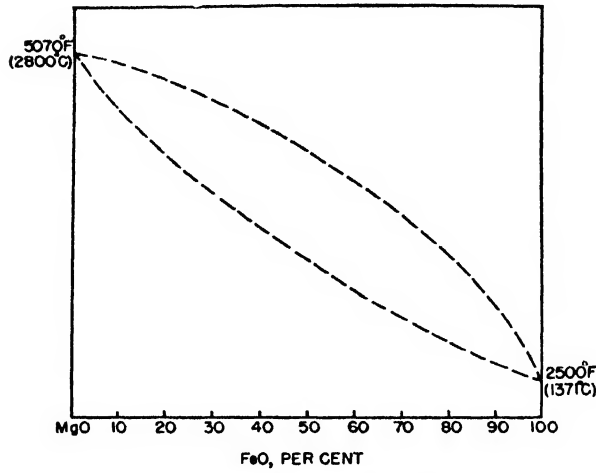
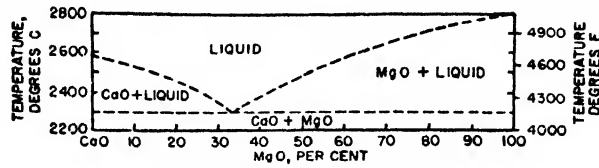
to assure conversion of the silicates to forsterite on firing. With an increase in the lime-silica ratio to 1, the less-refractory monticellite is formed; thus lime is a flux to forsterite. However, monticellite, within limits, is desirable in promoting a bond or set in granular open-hearth-bottom refractories. With a further increase in the lime-silica ratio to 2, the refractory but unstable dicalcium silicate will form. When stabilized against dusting with small amounts of Cr_2O_3 , B_2O_3 , or phosphates, the intentional formation of this compound in some granular refractories is a convenient means of correcting either for excess lime which would hydrate, or excess silica which would produce an undesired amount of monticellite.

In general, the susceptibility to fluxing by iron oxides in this system increases as MgO is replaced by CaO and as Fe_2O_3 is replaced by FeO. However, those portions of the CaO-MgO- SiO_2 -FeO, or CaO-MgO- SiO_2 -Fe $_2$ O $_3$ systems which involve refractories are receiving further study.

The reactions involved in the manufacture and use of chrome-bearing refractories are numerous and complex, and are the subject of many continuing investigations. The major improvement in such refractories occurred about 1930 with the recognition of the fact that combinations of chrome ore and magnesite in any proportion

FIG. 8-9. The MnO-Al₂O₃-SiO₂ system.FIG. 8-10. The CaO-SiO₂ system.

FIG. 8-11. The FeO-SiO₂ system.FIG. 8-12. The FeO-Fe₂O₃-SiO₂ system.



produced refractories with greater thermal shock resistance, volume stability, and refractoriness under load than did either material alone. The added magnesia converts the low refractory silicates in chrome ore to forsterite, and also replaces FeO in the chrome spinel, thus maintaining the $RO-R_2O_3$ balance when FeO is oxidized to Fe_2O_3 , which readily occurs. Of primary interest, however, are those reactions occurring in service, particularly the growth or disruptive swelling occurring

on absorption of iron oxide by the chrome spinel under oxidizing conditions. It is not clear that this phenomenon is the major cause of the usual process of failure, which is characterized by a progressive peeling or flaking away of the hot face, as it is known that substantial growth can be induced in basic brick by suitable heat treatment without additions of iron oxide.

Further studies are being made in refractory systems involving Cr_2O_3 , MgO, Al_2O_3 , SiO_2 , CaO and iron oxides.

SECTION 5

SELECTION AND TESTING OF STEEL-PLANT REFRACTORIES

The ultimate goal in refractory selection is to achieve minimum overall operating costs per ton of steel. However, this is the universal endeavor of all phases of steel production, in some of which costs may be lowered only at the expense of refractories, as by harder driving of furnaces to increase the rate of production, or by a change to cheaper fuels which may increase refractory wear. Careful costs analyses may be required to determine which of the conflicting factors can best be sacrificed for maximum economy. Therefore, the specific aim of the refractories engineer is to obtain minimum refractory costs per ton of steel, but only those obtainable under operating conditions dictated by the above considerations. The successful selection of refractories requires a knowledge of refractory characteristics, a knowledge of service conditions and, based on experience, the ability to correlate the two in terms of performance. The importance of this work is attested by the fact that most steel companies now employ ceramic engineers and maintain laboratories devoted to quality control testing and research on refractories.

Information on refractory characteristics as revealed by standard A.S.T.M. tests is readily available from refractory suppliers or independent laboratories but, as indicated earlier, such data are often inadequate and

even misleading if not used intelligently. However, as it is always desired to establish correlations between performance in service and the simplest possible laboratory tests, the standard tests are an excellent starting point, and, because the range of variation in a given property is needed for proper correlation and quality control, the use of these tests in steel-plant laboratories is often the only means of determining such variations.

Obtaining information on service conditions is most important to the proper application of refractories. Systematic studies of used linings by chemical analysis, petrographic microscope, and various laboratory tests have clarified many of the mysteries as to the precise mechanism of failure of blast-furnace and open-hearth refractories, and have been responsible for some of the most significant advances in refractory practice and, also very important, in furnace design.

With information on refractory behavior and service conditions much of the guesswork can be eliminated from service trials, which of course, provide the final answer on refractory suitability. When failures are encountered in such trials they are generally indicative of the need for more fundamental refractory research, or, as often occurs, of changes in operating conditions from those on which the refractory selection was based.

SECTION 6

SPECIFIC USES OF REFRACTORIES IN STEEL PLANTS

Blast Furnaces and Stoves—For many years, blast furnaces were lined throughout with high-duty fireclay brick and, despite much recent experimentation with special grades of fireclay brick, the high-duty type still predominates. However, the present high-duty brick, reflecting the demands for improved performance sought with the use of premium-priced, high-fired, super-duty brick, are substantially improved in strength, resistance to abrasion and to disintegration by carbon monoxide, and in size uniformity, over those produced prior to 1950. The special, high-fired types of both high-duty and super-duty brick are now generally confined to the zone immediately below the wearing plates and in the bottom, where optimum advantage is taken of their superior density and strength, and where their rigidity and sensitivity to thermal shock, believed to be a factor in their somewhat disappointing behavior in stacks and hearth walls, is of small concern.

After many false starts and much experimentation, carbon brick or block prepared from petroleum coke or anthracite are now widely used in hearth walls from below the tuyeres to the bottom of the cooling staves. Carbon layers across the hearth bottom have been less successful, and where now installed in a few furnaces, these layers are generally keyed in and built in greater

thicknesses, up to 85 inches, to prevent flotation. Used above the mantle, carbon has proved vulnerable to oxidation, carbon disintegration or alkali attack, and it is questionable whether its use in this region will ever prove economical. In the bosh, thin carbon linings, externally water-cooled, are operating successfully, and this practice will probably increase, as fireclay linings endure only through excessive use of expensive water-cooled plates.

Refractory requirements in hot-blast stoves vary with the temperature of operation, design, gas cleanliness, and location in the stove. Combustion-chamber walls must resist deformation under load, spalling and alkali attack; and, with the exception of spalling, this is true of the top 10 or 20 feet of the checkers. Mechanical strength, density, and size uniformity are the chief requirements of the lower checkers. Although long service has been obtained with entire high-duty fireclay-brick linings, particularly in stoves with proper temperature control, many installations employ super-duty, semi-silica, or combinations of these brick in combustion chambers, domes, and the top checker courses.

Hot-Metal Mixers—Hot-metal mixers present some refractory problems, despite their comparatively low operating temperatures, which seldom exceed 2500° F.

The chief difficulties derive from excessive joint erosion resulting from a mechanical washing by the molten iron and corrosion by slag, and from designs which prevent streamlined construction at the spouts. These difficulties are accentuated if the mixer is fired to maintain temperature since, as the temperature of the slag is raised, it becomes more reactive on the lining. While linings of high-duty blast-furnace brick, mica schist, and occasionally sandstone may still be used, there is a definite trend to the more volume-stable, denser, hard-fired super-duty brick which have greatly prolonged lining life by minimizing joint erosion. Occasionally, in efforts to bring the lining life into balance, the critical areas around the spout may be lined with various types of high-alumina brick. Numerous trials of various types of basic brick have generally been disappointing.

Basic Open-Hearth Furnaces—The modern basic open-hearth shop probably employs a wider range of refractories than is found in any other high-temperature process. In the furnace itself, the types of brick that may be found include silica, semi-silica, all grades of fireclay, some grades of high-alumina, and nearly all kinds of basic brick. In granular or plastic form are found raw, calcined or clinkered dolomite, magnesite, special magnesite or dolomitic ramming mixtures, chrome ore, plastic chrome ore, and fireclay, silica, and basic mortars. With few exceptions, however, none of these refractories can be said to be securely established in a given application, as not only do requirements vary widely with the different practices in different shops, but changes are constantly being brought about by improvements in refractories and changes in fuels with accompanying changes in design. Currently, the overall trend in open-hearth refractories is toward increased use of basic refractories. Other trends and developments are revealed by a brief examination of refractories used in the various furnace parts, and in accessory equipment in the shop.

Hearth—Insulating brick or concrete may be used on the steel bottom followed by fireclay brick or basic brick (usually burned chrome) or both. Above this point the bottom may be (1) burned in entirely with dead-burned magnesite and slag, (2) rammed in completely with special sized and bonded magnesite or dolomitic-magnesia mixtures or (3) partially rammed with plastic chrome or the above ramming materials and covered by a burned-in layer of magnesite or clinkered dolomite. Until 1939, practice (1) only was employed, but (2) is currently more popular.

Front, Back and Endwalls—Front and backwalls were the first applications above the hearth to be converted from silica to basic refractories and now these are almost universally basic, of either chrome-magnesite or magnesite-chrome, burned or unburned. Basic endwalls followed later, and while now predominant, silica endwalls are still employed. Steel-encased magnesite-chrome unburned brick are most popular, and these walls are generally constructed with steel tie plates every 2 to 4 courses which are welded to the buckstays to prevent buckling.

Port Sidewalls—Port sidewalls at present are generally silica, except in a relatively small number of furnaces with basic ends, in which they may be constructed like endwalls, with suspended panels immediately above floor level.

Doors—An important development in open-hearth refractories has been the introduction of stud-type doors with plastic base refractory linings, usually chrome ore. Performance has generally more than justified the extra cost over brick-lined doors.

Roof—The supremacy of silica brick for open-hearth

roofs has not seriously been threatened by basic brick, although a few experimental all-basic roofs have shown considerable promise of eventual justification. With the trend to silica brick of ever lower impurity content to obtain maximum refractoriness, particularly as exemplified in the super-duty silica brick, some minor relief of the temperature limitations imposed on the open hearth by silica is obtained, but scarcely enough to offset the attractiveness of the far greater potentialities provided by basic brick. Basic roof experience thus far indicates the need for brick of improved volume stability in service and further improvements in roof design.

Uptakes—Uptakes, like port sidewalls are generally silica except in furnaces with basic ends, in which suspended basic construction is universally used. Occasionally super-duty fireclay brick or higher-alumina brick are employed in the lower zones.

Slag Pockets, Checker Chambers, and Flues—In basic-end furnaces, basic brick may extend down into the slag-pocket walls, but these are generally silica and sometimes fireclay brick. Checkers and checker chambers are generally built of high-duty brick, although super-duty or 60 per cent high-alumina brick may be used in the hottest sections, or in the fantail, and semi-silica brick may be used in the checker-chamber roof. Intermediate- and low-duty brick may be used in the flues and stack.

Basic Electric Steelmaking Furnaces—While electric-furnace refractories do not have to contend with the high-velocity gases common to the open hearth, they are often subjected to higher temperatures and in the latter part of the process, to reducing atmospheres. For this reason, more consideration has been given to more expensive refractories of higher melting points in the basic process. Sillimanite or mullite brick made of kyanite have been justified in many roofs in recent years, although silica is still generally relied on for the larger furnaces. Basic brick, either chrome-magnesite or magnesite-chrome, are generally used in the walls, and some experimentations have also been carried out with basic brick in the roof. The developments in hearth refractories have paralleled those of the open hearth.

Pouring-Pit Refractories—These refractories—sleeves, nozzles, stopper heads, and ladle brick—must withstand severe thermal shock and corrosion by molten steel and slag; and, with the exception of stopper heads, they generally do this best when made of the lower grades of refractory clays. There are two reasons for this apparent paradox, both based on the time factor. First, sleeves and nozzles are single-use refractories and seldom subjected to these conditions for much more than an hour; and ladle brick are necessarily cooled between each similar treatment. Second, each of these refractories, especially nozzles and ladle brick, during this short period must become pyroplastic to function to best advantage—nozzles to form a tight seal for the stopper, and ladle brick to seal and prevent accelerated wear of joints. Stopper heads obviously must remain hard and corrosion-resistant, and hence are made of refractory clay-graphite mixtures. However, to minimize the rate of wear of such low-refractory ladle brick and nozzles, these refractories must be dense. Because of the peculiarities of these requirements, other types of refractories have seldom proved satisfactory, and improvements have been gained more from a recognition of the foregoing facts and efforts to improve density, texture, and workmanship than from any substitution of other refractory types.

Soaking Pits—Soaking-pit linings must possess good load-carrying ability, high resistance to abrasion, and resistance to iron-oxide attack, particularly at the slag

line. Cover linings must be resistant to spalling as must the top of the pit walls, although mechanical abuse in this zone is the chief cause of deterioration. Several types of refractories for pit walls have been and are being employed, with a trend to the more siliceous types, such as sandstone, silica and semi-silica brick, and mica schist, as these generally meet the above requirements better, and require less maintenance, than do fireclay brick. Slag-line maintenance remains a problem, and although basic brick are usually employed, comparable service has been obtained with 70 per cent alumina brick with resultant savings from lower first cost. Super duty brick, semi-silica brick, castables and plastics have also come into use, particularly for cover linings, where a definite trend toward the monolithic refractories has developed. Wide variations in operating conditions are chiefly responsible for the variety of refractories in current use, as each plant presents individual refractory problems.

Heating Furnaces—Heating furnaces include a variety of types and service conditions, but refractory requirements are generally met by fireclay brick. Spalling resistance is needed in most furnaces however, and recognition of this has led to greater use of super-duty and semi-silica brick in recent years, particularly in suspended and sprung arch roofs, and in the walls of heating zones of continuous slab-heating furnaces. More recently, rammed or cast monolithic walls and roofs have been used with considerable success. The problem of refractories for solid hearths in such furnaces has been eased but not eliminated by basic brick, rammed

basic plastics, or high-alumina brick. Burner-block difficulties have been greatly improved through the use of super-duty, high-alumina, or sillimanite block, either prefired or rammed. In many low-temperature heat-treating furnaces advantage can be taken of the low heat capacity and insulating value of light-weight fireclay brick.

Chapter 9

ADDITION AGENTS USED IN STEELMAKING

Definitions—Steelmaking involves the deliberate addition of various chemical elements to the molten metal to effect several desirable ends. These ends may include deoxidation of the molten metal to the desired degree, control of grain size, improvement of the mechanical and physical properties and corrosion resistance of the steel, increase of the response of the steel to subsequent heat treatment, or attainment of other specific effects that are discussed elsewhere in this book. Originally, the chemical element it was desired to incorporate into the steel was added to the bath in the form of an alloy that consisted principally of iron but was rich in the desired element. Such alloys, because of their high iron content, became known as *ferroalloys*, and most of the available types were produced in the iron blast furnace. Eventually, the production of alloys for steelmaking purposes began to be carried out in electric-reduction and other types of furnaces as well, and a number of alloys now produced contain relatively little iron. For this reason, the term *addition agent* is preferred to describe any of the materials added to molten steel for altering its composition or properties; under this definition, the ferroalloys form a special class of addition agents.

The more common addition agents definitely in the ferroalloy class include alloys of iron with aluminum, boron, calcium, chromium, columbium, manganese, molybdenum, nitrogen, phosphorus, selenium, silicon, tantalum, titanium, tungsten, vanadium, and zirconium. Some of these chemical elements and others are available in addition agents that are not ferroalloys, as well as in almost pure form; these include relatively pure metals such as aluminum, calcium, cobalt, copper, manganese and nickel; oxides of molybdenum, nickel and tungsten; carbon, nitrogen and sulphur in various forms; and alloys consisting principally of combinations of two or more of the foregoing elements. The more important of the addition agents (including ferroalloys) will be discussed individually later in this chapter. Although work on the use of rare-earth alloys is in progress, the results reported thus far are still under investigation and the use of such alloys in steelmaking will not be discussed here.

Use of Addition Agents—Addition agents may be added with the charge in the steelmaking furnace, or in the molten bath near the end of the finishing period, or in the ladle or in the molds. Timing of the alloy additions is dependent on the effect of the addition on the temperature of the molten metal, ease with which specific addition agents go into solution, susceptibility of a particular addition agent to oxidation, and formation and elimination of reaction products.

Economy in manufacture of alloy steels requires consideration of the relative affinity of the alloying elements for oxygen as compared with the affinity of iron for oxygen. For example, copper, molybdenum, or nickel may be added with the charge or during the working of the heat and are wholly recovered. Chromium and manganese, because they are easily oxidized, should be added late in the heat and all or part of these two may be added in the ladle. In open-hearth practice, easily

oxidized materials such as aluminum, boron, titanium, vanadium, and zirconium are normally added in the ladle in order to minimize oxidation losses.

It is often necessary to preheat the ferroalloy to avoid undue chilling of the bath. When large additions are made entirely to the bath, time must be allowed for the molten steel to be reheated to the desired temperature before tapping. The ferroalloy additions may be split between the furnace and the ladle, and in cases where excessive chilling of the metal in the ladle is to be avoided, the lower alloy recovery in the furnace must be accepted.

The agents to be added to the bath should be lump size (say 5 inches) in order to penetrate the slag easily. For ladle additions, the alloy should have a maximum size of approximately 2 inches in order to assure rapid solution.

Storage Facilities for Addition Agents—From the standpoint of material handling and of conservation and identification, it is advisable to store addition agents in properly designed bins in which they are protected from the weather. The location and design of bins should make the contents quickly available and with a relatively low handling cost. All bins should provide identification of the contents since confusion may be costly due to failure to meet specified chemical composition of the finished steel. Certain addition agents are more easily broken on handling than others, and caution should always be exercised to avoid production of fines.

COMPOSITIONS OF COMMON ADDITION AGENTS

Ferromanganese is the most important of the ferroalloys used in steelmaking. **Standard ferromanganese** contains 74 to 82 per cent manganese, and contains not more than 1.25 per cent silicon, 0.35 per cent phosphorus, 7.50 per cent carbon, and 0.05 per cent sulphur. **Low-phosphorus ferromanganese** suitable for addition to acid open-hearth steel should not contain over 0.10 per cent phosphorus. **Low-carbon ferromanganese** is used when it is important to limit the amount of carbon entering the steel from the ferromanganese addition. This low-carbon product is available in several grades containing increasing amounts of carbon, e.g., 0.07 per cent, 0.10 per cent, 0.15 per cent, 0.30 per cent, 0.50 per cent, and 0.75 per cent (all maxima), and the lower the carbon content the higher the price per pound. All of the foregoing grades of low-carbon ferromanganese contain 80 to 85 per cent manganese. **Medium-carbon ferromanganese** contains a maximum of 1.5 per cent carbon and 80 to 85 per cent manganese. If low-carbon and medium-carbon ferromanganese are used as bath additions (to the furnace) such additions should be made after the bath has been deoxidized.

Silicomanganese is used by some open-hearth furnace operators as a furnace addition to **block the heat** (retard the oxidizing reactions taking place in the furnace toward the end of the finishing period) because of the shorter holding time required from time of addition to tap, as compared with the use of ferrosilicon followed by

ferromanganese. Silicomanganese contains 65 to 68 per cent manganese, 18 to 20 per cent silicon, and a usual maximum of 1.5 per cent carbon.

Spiegeleisen is available in grades containing 16 to 28 per cent manganese, with the carbon content not over 6.5 per cent. The silicon content of spiegeleisen is from 1.0 to 4.5 per cent. A grade known as *silicospiegel* contains 25 to 30 per cent manganese, 2 to 3 per cent carbon, and 7 to 8 per cent silicon. Spiegeleisen (usually called *spiegel*) is used as an addition to the bath and in the refining or last stage of the open-hearth process.

Ferrosilicon—The low-silicon grades of ferrosilicon, which usually start at 10 per cent silicon and ordinarily do not exceed 17 per cent silicon, generally are blast-furnace products, and contain 1.50 per cent maximum carbon. **Electric-furnace ferrosilicon**, made by an electric-furnace process, is graded according to silicon content. The principal grades contain, respectively, 25 per cent, 50 per cent, 75 per cent, 85 per cent, and 90 to 95 per cent silicon. The grades containing the most silicon can be obtained with special low aluminum content. The 50 per cent silicon grade is by far the most widely used, and is employed chiefly as a ladle addition. The 10 to 15 per cent and 25 per cent silicon grades are those usually used in the open-hearth furnace for blocking, although other grades are also used, and may also be used as deoxidizers added prior to addition of other more expensive alloys.

Ferrochromium, containing 65 to 72 per cent chromium and a maximum of 2 per cent silicon, is classified into grades by carbon content. The respective grades contain 0.06 per cent, 0.10 per cent, 0.15 per cent, 0.20 per cent, 0.50 per cent, 1.00 per cent, and 2.00 per cent carbon, along with the most commonly used and cheapest grade containing 65 to 69 per cent chromium and 4.50 to 8.50 per cent carbon. Ferrochromium containing the higher amounts of carbon is used as a furnace addition. As in the case of ferromanganese, the alloys of lower carbon content are the more expensive. Briquettes of ferrochromium in which are incorporated chemical reagents to provide exothermic reactions permit chromium to be added to the ladle in this form without an undue chilling effect.

Ferrovandium, containing 35 to 55 per cent vanadium, usually is added to killed steel in the ladle.

Ferromolybdenum, containing 55 to 75 per cent molybdenum, is used where the higher molybdenum contents are desired. For lower molybdenum contents, molybdenum oxide or calcium molybdate may be used. All are furnace additions.

Ferrotitanium available in the following grades generally is used as a ladle addition: regular grade containing 40 per cent titanium and a maximum of 0.10 per cent carbon; low-carbon grade containing 25 per cent titanium and a maximum of 0.10 per cent carbon; and a high-carbon grade containing 15 to 19 per cent titanium and 6 to 8 per cent carbon.

Zirconium is obtained from one alloy containing 12 to 15 per cent zirconium, 39 to 43 per cent silicon, and a maximum of 0.20 per cent carbon; or another containing 35 to 40 per cent zirconium, 37 to 52 per cent silicon, and a maximum of 0.50 per cent carbon. These zirconium alloys generally are added in the ladle.

Ferrophosphorus in two grades containing, respectively, 17 to 19 per cent and 23 to 26 per cent phosphorus, usually is added in the ladle.

Nickel, obtained in the forms of sheared electrolytic cathodes or as ingots produced from remelted cathodes, both containing a minimum of about 99 per cent nickel, is used as a furnace addition. Little or no nickel is lost through oxidation when it is added to the bath, so nickel

steels are made by charging nickel-steel scrap and adding metallic nickel after the charge has been melted completely. Nickel oxide is sometimes used in conjunction with reducing slags in the electric furnace as a source of metallic nickel.

Copper usually is added to the bath in the form of virgin copper pigs or as scrap copper. It also can be recovered, with little or no loss, from copper-bearing steel scrap in the initial furnace charge.

Aluminum usually is added in the form of secondary aluminum having an aluminum content of from 85 per cent upward. It generally is used in the form of shot for addition to the ingot mold, or as shot or bars for addition to the ladle. In special cases, as when used for the deoxidation of small ingots, the metallic aluminum content of the addition may be as high as 97.5 per cent. Limitations are placed on the copper content of aluminum used in steelmaking operations, and generally it is not allowed to exceed 4.5 per cent.

Cobalt in metallic form (97 to 99 per cent pure) is usually added to the furnace in the form of shot or rondelles in the manufacture of high-speed steels, permanent-magnet steels and other special steels in the electric furnace.

Ferrocolumbium, containing 50 to 60 per cent of columbium and up to 8 per cent silicon with a maximum of 0.40 per cent carbon, is usually added through the reducing slag in electric furnaces in the production of austenitic stainless steels of the chromium-nickel type and alloys for use at high temperatures.

Ferroselenium, used for the addition of selenium to stainless steels to improve their machinability, is added to the ladle during the tapping of electric-furnace heats. It contains 50 to 60 per cent of selenium.

Tantalum for steelmaking purposes is available as ferrotantalum-columbium, which has a content of approximately 20 per cent tantalum, 40 per cent columbium and a maximum of 0.30 per cent carbon. It can be used as a replacement for ferrocolumbium in some cases.

Tungsten, used principally in the manufacture of high-speed tool steels in electric furnaces, is available from ferrotungsten, the standard grade of which contains 70 to 80 per cent of tungsten and a maximum of 0.60 per cent of carbon. Tungsten is also obtained from oxide forms such as scheelite and calcium tungstate.

Sulphur as required by specification is added to the ladle in the form of such addition agents as flowers of sulphur, stick sulphur, iron sulphide, or less often as manganese sulphide, sodium sulphide, etc.

Carbon—For raising the carbon content of the steel during tapping, additions may be made to the ladle in the form of coke, anthracite coal, graphite, or petroleum derivatives. It is desirable that such additions be low in sulphur and high in carbon.

"Hardenability Intensifiers"—A number of ferroalloys are designed to increase the hardenability of steel when added in relatively small amounts. They usually contain boron together with one or more of the following elements: silicon, titanium, vanadium, zirconium, aluminum, manganese, calcium, or other deoxidizers or denitrifiers.

The foregoing brief descriptions of addition agents summarize only the principal sources of the various elements for the manufacture of steel, with some reference to the manner in which most of them are used. Specific reasons and the manner for the use of individual addition agents, or combinations of them, are discussed in other chapters describing the several steelmaking processes and the properties and heat treatment of AISI alloy steels, stainless and heat-resistant steels, and tool steels.

Chapter 10

SCRAP FOR STEELMAKING

Scrap consists of the by-products of steel fabrication, and worn out, broken or discarded items containing iron or steel. It is one of the two principal sources of metal in steel-making; the other principal source is iron from the blast furnaces, either molten as it comes from the blast furnace ("hot metal") or in solid pig form. Scrap is of great practical value. Every ton of scrap consumed in steelmaking is estimated to displace and conserve for future use $3\frac{1}{2}$ to 4 tons of other natural resources including iron ore, coal and limestone. On the average, the steel industry consumes about equal quantities of scrap and blast-furnace iron. According to recently published figures of the American Iron and Steel Institute, the steel industry consumes an average of about 54,000,000 tons of iron and steel scrap in producing 100,000,000 net tons of ingots and steel for castings. The metallic charge for an open-hearth steelmaking furnace consists, on the average, of about one-half steel scrap and one-half molten iron (hot metal). Bessemer converters and other pneumatic processes employ molten iron (hot metal) almost exclusively; while the electric furnace usually is charged almost entirely with cold scrap. Table 10—I summarizes the consumption of scrap and blast-furnace iron by the several steelmaking processes and the blast furnace during 1953.

Table 10—I. Consumption of Scrap and Hot Metal and Pig Iron by Steelmaking Processes and Blast Furnaces During 1953*

Process	Consumption (Net Tons)	
	Scrap	Hot Metal and Pig Iron
Bessemer	218,998	4,249,149
Open-Hearth	48,698,391	61,105,870
Electric	7,327,627	134,144
Blast Furnace	5,005,873	21,456

* American Iron & Steel Institute

Types and Sources of Scrap—Scrap iron and steel may be classified as originating from two sources: home scrap produced as unsalable products unavoidably resulting from steelmaking and finishing operations, and purchased scrap.

Home scrap (also called "revert scrap") includes such items as pit scrap; ingots too short to roll; rejected ingots; ingot crops; crop ends from blooms and billets; shear cuttings from trimming flat-rolled products to accurate size; products irrecoverably damaged in handling or finishing; ends cut from bars, pipe or tubing to bring them to standard or exact ordered length; turnings from machining operations, and so on. Bloom crops constitute the largest single item of home scrap. Advances in steelmaking techniques have tended to reduce the

proportion of the ingot that must be cropped. While this has reduced the quantity of home scrap, it has resulted in a higher proportion of steel from the ingot being shipped to consumers in the various forms of finished steel. This has partially offset the loss in home scrap by increasing the proportion of prompt industrial scrap, discussed below.

In general, according to the source referred to above, about 31,500,000 net tons of home scrap results from the manufacture of 100,000,000 net tons of steel ingots (and steel for castings—a small percentage of the whole) and the processing of this steel into finished products.

Purchased scrap must be used to the extent of 22,500,000 net tons to supplement the 31,500,000 tons of home scrap to provide the 54,000,000 tons of iron and steel scrap needed to produce the hypothetical 100 000,000 net tons of steel used here as a basis for discussion. Purchased scrap is divided into two general classifications: (1) dormant scrap and (2) prompt industrial scrap.

Dormant scrap comprises obsolete, worn out or broken products of consuming industries. Typical examples of dormant scrap are: discarded steel furniture, washing machines, stoves and other outdated consumer goods; beams, angles, channels, girders, railings, grilles, pipe, etc., arising from the demolition of buildings; useless farm machinery; obsolete, broken or damaged industrial machinery; old ships; railroad rails and rolling stock that have outlived their usefulness; wrecked automobiles, and so on. This type of scrap, because of its miscellaneous nature, requires careful sorting and classification in segregated lots to prevent the contamination of steel in the furnace with unwanted chemical elements that may be present in some of the scrap. It should also be of such physical size as to facilitate handling and loading into charging boxes. The need for proper classification and preparation of dormant scrap is emphasized by the existence of 75 different specifications covering various grades of scrap for use in blast furnaces, acid and basic open-hearth furnaces, electric furnaces, bessemer converters, gray-iron foundries and elsewhere. In addition, the Association of American Railroads has forty-five specifications applying to scrap of railroad origin. These all have been prepared to facilitate proper classification of scrap for different uses.

Prompt industrial scrap is generated by steel consumers in making their products. It may consist of the unwanted portions of plate or sheet that has been cut or sheared to the desired final size and shape, trimmings resulting from stamping and pressing operations, machine turnings, rejected products scrapped during manufacture, short ends, flash from forgings, and other types of scrap. Prompt industrial scrap can usually be identified easily as to source and composition, provided that proper plans for segregation are in effect in the consumer's plant, the scrap dealer's yard, and in the steel plant.

Of the 33,000,000 tons of purchased scrap actually con-

sumed in 1953, about 9,500,000 tons originated as prompt industrial scrap in metal-working factories in the processing of finished steel and of iron castings into capital and consumer goods. About 5,000,000 tons came from auto-wrecking operations, 4,000,000 tons from the railroads, 2,000,000 tons from farms, and the remainder from shipbreaking, public utilities, refineries, collectors, and demolition—with demolition the major general source. In a typical month during that year, open-hearth furnaces used 56 per cent of all purchased scrap, 72 per cent of the home scrap, and 83 per cent of the pig iron. Electric furnaces accounted for 6 per cent of scrap and pig iron combined. Blast furnaces used 8 per cent of all purchased scrap.

Physical Preparation of Scrap—Scrap is classified according to its physical size and chemical composition. Pieces too large to be accommodated by charging-machine boxes must be cut into satisfactorily smaller sizes; shears, flame-cutting, impact devices and other means may be used, depending upon the type of scrap being handled. Sometimes, very large pieces of scrap that cannot pass through the furnace doors may be charged into an open-hearth furnace by overhead crane when the furnace roof is off during rebuilding. Sheet shearings, punchings and similar types of relatively thin and usually small pieces of scrap may be compressed into block-like bundles in specially-designed hydraulic baling presses; since about half of the steel rolled in the United States at present is in the form of relatively thin flat-rolled products, large quantities of scrap require baling.

Assuming that scrap has been properly sorted with respect to its chemical composition, as discussed later, the primary purpose achieved by the proper preparation of scrap is to increase the amount of scrap that can be loaded at one time into a charging box for placing the scrap in the steel-producing furnace. The denser the load in the charging boxes, the fewer the number of boxes that need to be loaded, transported and their contents dumped into the furnace, and the less the time consumed in charging the furnace. Delays in charging can result in considerable loss of steel production.

The prime grade of purchased scrap for production of open-hearth steel must be at least $\frac{1}{8}$ -inch thick, no more than 18 inches wide, nor more than 5 feet long. Electric furnaces require purchased scrap of smaller dimensions ranging in size from 3 feet down to punchings; further details relating to scrap for electric-furnace melting are given in Chapter 16. Short lengths of turnings are the preferred form of scrap for blast-furnace use.

Chemical Composition of Scrap—Certain chemical elements are desirable constituents of scrap for steel-

making, especially when used in electric furnaces, as discussed in Chapter 16. In general, however, scrap for all of the steelmaking processes should be free from unknown and unwanted elements referred to as "tramp alloys." The increasing use of alloy steels of many compositions has aggravated the tramp alloy problem since more and more purchased scrap may be expected to include unidentified alloy steels.

The segregation of home scrap according to its chemical composition is relatively simple. Purchased scrap, especially dormant scrap, presents some problems since a large percentage of it is of unknown origin and composition. While it would be impossible to chemically analyze each individual piece of the huge amounts of dormant scrap consumed every year, the chemical analysis of selected samples of individual lots sometimes is employed by steel plants in the classification of scrap. Spectrographic analysis sometimes is employed instead of chemical analysis because it is more rapid; however, both are relatively time-consuming and expensive and both require careful selection and preparation of samples. Some less costly but less accurate tests are commonly used; these include magnetic tests, spark tests, spot tests and pellet tests as described briefly in Chapter 16.

When the chemical composition of scrap is known, the scrap can prove to be a valuable source of alloying elements needed in the production of alloy steels. Full advantage is taken of this source in the production of alloy steels in the electric furnace. In the open-hearth furnace, however, the preponderance of production consists of carbon and low-alloy steels and, in general, alloying elements in scrap are a source of trouble.

Tin, copper, nickel and other elements present in scrap will alloy readily with steel and, in many instances, render it unfit for its intended use. Relatively small amounts of these metals can contaminate an entire heat of steel. Tin and copper in certain ranges of composition cause brittleness and bad surface conditions in steel. Nickel and tin not only contaminate heats into which they may be unintentionally introduced, but may deposit a residue in the furnace that is absorbed by successive heats with resultant contamination. Zinc volatilizes in steelmaking furnaces and, by reacting with the hot refractories, causes rapid deterioration of brick work. Lead is extremely harmful to furnace bottoms and refractories and, if present in sufficient quantities, may cause furnaces to break out by penetrating joints or cracks in the bottom to form channels that may be followed by molten steel. The foregoing examples represent some of the difficulties caused by only a few of the chemical elements that may enter steel from poorly prepared or carelessly classified scrap.

Chapter 11

THE MANUFACTURE OF SPONGE IRON AND WROUGHT IRON

SECTION 1 SPONGE IRON

Sponge iron provided the main source of iron and steel for many centuries before the blast furnace was developed around 1300 A.D. It was produced in relatively shallow hearths or in shaft furnaces, both of which used charcoal in the double role of a fuel to supply heat and a chemical reducing agent to reduce the iron from the ore by combining carbon from the fuel with the oxygen from the ore. The product of all of these smelting processes was a spongy mass of coalesced granules of nearly pure iron intermixed with considerable slag. Usable articles of wrought iron were produced by hammering the spongy mass while still hot from the smelting operation, to expel most of the slag and compact the mass. All of the methods whereby wrought iron can be produced directly from the ore are called **direct processes**. After development of the blast furnace, which made large quantities of iron having a high carbon content available, wrought iron was produced by refining this high-carbon material; because two or more steps were involved in the processes employed, they came to be known as **indirect processes**.

Direct methods are still in use among certain peoples, and, indeed, have never been wholly abandoned even by the most advanced nations. The ease with which iron ores are reduced makes the direct process appear enticingly simple and logical. The reduction takes place at low temperatures and absorbs little heat, some of the reactions actually being exothermic.

For the past hundred years, sponge iron itself has found increasing use in various industrial processes, other than in the manufacture of wrought iron. Since the iron produced in sponge form is of high purity, it is used extensively in the chemical industry as a strong reducing agent and is chemically much more active than steel or iron in the form of millings, borings, turnings, or wire, because of its porous or spongy nature. Sponge iron may be produced as a granular material or as a sintered mass, depending upon the methods of manufacture. In the granular form, in which it is commonly known as **powdered iron**, it is used in the manufacture of many useful articles by the techniques of **powder metallurgy**, in which the powders are first compacted by pressure alone into the approximate shape of the finished article; the compact is then "sintered" at a temperature ranging roughly between 1800° and 2000° F in furnaces provided with a protective atmosphere to prevent oxidation. The sintered articles are then pressed or otherwise processed to give them their final accurate shape.

It may be mentioned that iron powders are produced,

not only by direct reduction of iron ores or oxides using solid carbonaceous reducing agents (coal, coke or charcoal), but also by electrolytic processes, by thermal decomposition of iron carbonyl which has the chemical formula $\text{Fe}_2(\text{CO})_9$ and by the reduction of iron ore or pure oxides with various gaseous reducing agents such as carbon monoxide or hydrogen.

In connection with the direct reduction processes mentioned, it is interesting to note that dry iron oxides (hematites or magnetites), in the presence of a small excess of reducing substance, form dark-gray porous masses having the same size and shape as the original lumps or particles when reduced at temperatures below 1650° F (900° C), and that a temperature of 1740° to 1830° F (950° to 1000° C) is necessary to effect complete reduction in a reasonable length of time, unless the ore is of a selected particle size. At 1830° F (1000° C) the product begins to sinter, and forms a pasty, porous mass at 2190° F (1200° C). At 2370° F (1300° C) the mass absorbs carbon rapidly, if the latter is present, and begins to fuse, though the upper melting (completely liquid) point of pure iron is 2785° F (1530° C).

In addition to the commercial production of sponge iron by direct reduction for the foregoing purposes, much effort and expense has been incurred for a long period of time in an effort to discover some practicable direct-reduction process that would produce iron economically for use in steelmaking operations, thus eliminating, at least partially, the need for blast-furnace pig iron. Although iron ore can be reduced directly with relative ease on a small scale, no commercially useful process has been developed to date that shows any promise of supplanting the blast furnace as the chief source of iron units for steelmaking.

The attempts to develop a direct process for manufacturing iron and steel that could compete with the indirect processes now in use have embraced practically every known type of apparatus suitable for the purpose, including pot furnaces, reverberatory furnaces, regenerative furnaces, shaft furnaces, rotary and stationary kilns, retort furnaces, electric furnaces, and various combination furnaces. Many different kinds of reducing agents have also been tried, such as coal, coke, graphite, distillation residues, fuel oil, tar, producer gas, coal gas, water gas and hydrogen. Practically all of these processes have been abandoned, and it is unnecessary to give details of them here, especially since adequate descriptions are to be found in other works on this subject.

While the process of producing low-carbon iron direct from ore is theoretically very attractive and appears

more logical than indirect processes (in which high-carbon pig iron is first produced and then purified to make iron and steel), direct processes have failed in competition with indirect methods. Apparently, the chief reasons why the direct process resists all efforts to establish it are these: (1) The ore must be a very rich one. (2) The ore must be finely divided and intimately mixed with or carefully placed over the reducing agent. (3) So far, no practical plan has been evolved whereby the ore and reducing agent may be mixed in proper proportions to leave no excess of either. If an excess of

the former is permitted, the process is wasteful of ore, while in the presence of an excess of the latter, the iron is obtained at a low temperature in sponge or pasty form which is hard to handle. If produced at a higher temperature, phosphorus in the ore will be reduced and the carbon absorbed from the fuel, giving a very impure metal, little better than pig iron. Sulphur, except when an excess of calcium oxide is present, is readily absorbed by the metal and cannot be removed except under highly reducing conditions such as prevail in the blast furnace or an electric furnace.

SECTION 2

DIRECT PROCESSES FOR MAKING WROUGHT IRON

Historical Background of Direct Processes—From about 1350 B.C. to 1300 A.D. all of the iron wrought into tools and weapons was produced directly from ore. Reduction of iron ore was carried out in a relatively simple manner using charcoal as fuel. In the earliest times the operation was carried out in crude hearths which eventually were superseded by furnaces of various designs having a strong family resemblance.

In the first smelting hearths or furnaces, natural draft alone supplied the air for combustion. Later, it seems to have been the practice to construct the hearth on a hillside or at the base of a cliff facing in the direction of a prevailing wind (Figure 11-1). A wind of suitable di-

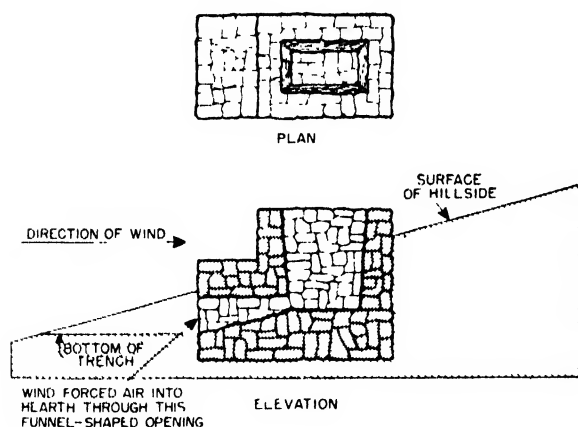


FIG. 11-1. Schematic representation of an early smelting furnace, built on a hillside to take advantage of the direction of prevailing winds to supply a gentle blast. Actual discovered remains of similar furnaces indicate that the hearth of such furnaces may have been relatively shallower and greater in area in proportion to their height.

rection and velocity could be led into the burning fuel bed through an opening in the hearth or furnace wall to provide the air blast required to produce strong combustion of the charcoal fuel. Still later, devices for blowing air into the fuel bed were developed to make the process independent of wind and weather. These devices ranged all the way from mouth-blown hollow reeds, through foot-operated bladders of animal skins, foot-operated bellows, hand-operated bellows, and air-blowing devices operated by treadmills and water wheels. Another device somewhat widely used was the *trompe*, which made use of the aspirating effect of a falling column of water inside a tube, to draw air in

through holes in the sides of the tube near its top, and expel the air into a closed chamber at the bottom; the air was piped from this chamber to the furnace (see Figure 11-2).

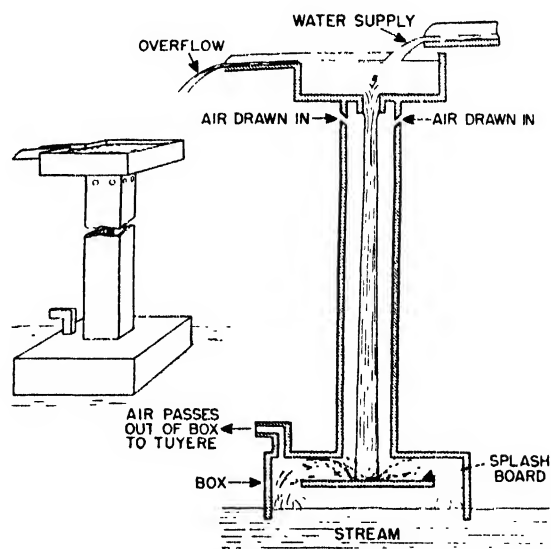


FIG. 11-2. (Right) Schematic representation of the operation of the *trompe* for utilizing the principle of aspiration to provide air blast for smelting furnaces. (Left) Sketch of external appearance of a *trompe*. The cross-section of the vertical column was more commonly round than square.

In view of the many centuries in which the direct process was used, it is to be expected that many different methods and types of apparatus would have been developed. Little is known of the furnaces in use prior to the eighteenth century, but the majority were of the hearth type, while the remainder were of the shaft type and may be compared to small blast furnaces, as will be discussed later. While these furnaces might, and did, differ widely as to form, size, and materials of construction, the fundamental metallurgical principles were the same in all. The ore, in lumps or in more or less finely divided form, was mixed with charcoal and charged into the furnace. Charcoal served the triple purpose of (1) a fuel to supply heat, (2) a reducing agent, and (3) a protector to shield the hot reduced metal from the oxidizing influences of the air.

The Catalan Process—The Catalan hearth, as the furnace used in this process was called, was anywhere from 20 inches square and 16 inches deep to around 30

inches by 40 inches and something over two feet deep. The nozzle or tuyere, through which the blast was blown into the furnace, was placed about 9 inches from the bottom in the smaller hearths and about 15 inches from the bottom in the larger hearths. The hearth was filled to the level of the tuyere with charcoal, on which was piled lump ore together with charcoal. These materials were placed so as to form two separate columns, the charcoal against the tuyere side of the hearth, and the ore against the other side (Figure 11—3). A gentle

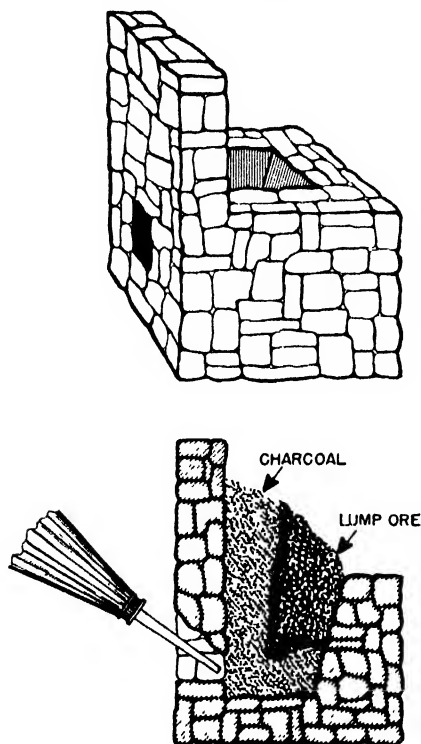


FIG. 11—3. (Above) Representation of a Catalan hearth or forge used for smelting iron ore up until relatively recent times. (Below) Cross section showing method of charging fuel and ore in the Catalan hearth, and approximate position of the nozzle supplied with air by a bellows.

blast of air was applied at first and the carbon monoxide, formed by combustion of the charcoal, passed preferentially through the open pile of lump ore. The ore was reduced to metallic iron when the oxygen in the iron oxide of the ore combined with the carbon monoxide to form carbon dioxide. The waste gases escaped at the top of the charge. Charcoal (along with fine ore) was added at regular intervals to replace that consumed in combustion. After about two hours, the lump-ore column was gradually pushed downwards and the temperature of the hearth was raised by increasing the blast. As successive portions of the ore became reduced, they were pushed nearer the tuyere where the hearth was hottest. By the time the ore had reached the hotter regions, it was largely reduced to the metallic state.

The unreduced portion of the lump ore, along with part of the fine ore added periodically with charcoal, formed a siliceous slag of high iron content with the gangue (waste material). The metallic iron resulting from reduction of the ore became pasty at the temperatures existing near the tuyere, to form a coherent loup

or bloom. After as much as possible of the ore had been reduced, the bloom was pried out of the hearth and hammered into bar form.

The American Bloomery—Among the variations of the process just described was the **American Bloomery Process** which was very similar to the Catalan process, differing from it chiefly in the fact that ore in a fine state, instead of in lumps, was mixed with charcoal to form the charge. The American bloomery represented the highest development in the simple hearth type of furnace for producing wrought iron. The bellows supplying the blast was operated by a water wheel or steam engine. The hearth was provided with a water-cooled metal bottom-plate, and cast-iron plates lined the sides. These hearths, rectangular in shape, were about 2 feet deep and 3 feet wide, and were surmounted by a tall chimney in the form of a truncated pyramid for carrying off the hot waste gases. The blast was heated (to save fuel) by passing the air through cast iron pipes around which the hot waste gases passed on their way from the furnace to the opening of the stack. Usually, bloomeries were open in front like an open fireplace, with the tuyere placed either at one side or at the back, about 20 inches above the bottom. Charcoal was first put into the hearth, the blast turned on, and when the fire was burning well, some ore was spread on the charcoal. Thereafter, charcoal and ore were added alternately until a sufficient amount of metal had collected upon the bottom. Then the iron, in a pasty mass and mixed with much slag, was removed from beneath the fuel bed with bars and tongs and hammered into a bloom. The last wrought iron to be produced by the bloomeries in this country was made in 1901.

The Stuckofen (High Bloomery)—The stuckofen or old high bloomery, variations of which appear to have been called salamander furnace, wolf furnace, wolf oven, wulf's oven and luppenofen or loup furnace, evolved from the Catalan type of hearth furnace. The earliest recorded types of these furnaces were in territories included in pre-World War II Germany (Nassau, Siegen, Saxony) and in parts of Austria, Belgium and the Netherlands. It was a furnace, 10 to 16 feet high, having a round, elliptical or rectangular cross section (greatest cross-sectional dimension, about 3 to 4 feet). One or two tuyeres supplied the blast, located somewhat over a foot above the hearth. Fuel and ore were charged into the top of the furnace, being replenished from time to time as smelting proceeded. A drawing hole was provided in the wall at the bottom of the shaft for extracting the blooms. This hole was closed by brick or stone work that was torn out each time a bloom was removed, after which the hole was again closed. Charcoal was the only fuel used.

The furnace called the wolf oven has been described as lower than the stuckofen, perhaps 6 to 7 feet in average height. Intermediate in size, between the wolf oven and the stuckofen, are the *blasofen* and *bauernofen*. The *bauernofen* corresponds to the *osmund furnace* (about 8 feet high) used in Sweden (a similar type was used in India). A type of furnace which originally resembled and was operated like the stuckofen, and was later adapted to produce either blooms of low-carbon wrought iron or molten, high-carbon iron, resembled a crude blast furnace and was termed *blauofen*, *blau furnace* or *blue furnace*.

The stuckofen may be considered as the forerunner of the modern blast furnace which produces only liquid, high-carbon iron. Liquid high-carbon iron often was produced in the stuckofen, intentionally or otherwise. This occurred when the reduced iron was in contact with hot fuel away from the blast long enough to absorb

sufficient carbon to reduce its melting point to where it would become liquid. The height of the furnace made this possible, especially if the operating temperature was high enough. A *flussofen* was strictly a primitive blast furnace intended only to produce molten, high-carbon iron. It may be said in general that the furnaces for producing molten high-carbon iron developed gradually from the early hearths in which only wrought iron was produced. The development consisted in gradually increasing the height of the furnace and introducing the charge at intervals through the top. These higher furnaces, distinguished as a class from the Catalan type of hearth or bloomery, were termed shaft furnaces. The

modern blast furnace is a shaft furnace, gradually evolved from the *stuckofen* and *flussofen*. In its early days it was called a **high furnace**, from its German name, *hochofen* (French: *haut fourneaux*). It is designed solely to produce molten iron and operates continuously, in that the solid raw materials (ore, coke and limestone) are charged at the top at regular short intervals, and the molten iron and slag which collect in the hearth are tapped out at longer intervals. Modern blast furnaces are commonly 80 to over 100 feet high, the larger furnaces producing well over 1000 tons of molten iron per day. The development of the modern blast furnace will be described in Chapter 12.

SECTION 3

INDIRECT PROCESSES FOR MAKING WROUGHT IRON

After furnaces which produced molten high-carbon iron became commonly employed in Europe, part of their product was used to produce iron castings by pouring the liquid metal into molds of the desired shape. Such cast iron had limited usefulness, since it was inherently hard and brittle due to its high-carbon content and the presence of other elements that entered the iron during reduction of the iron ore. It was not malleable, that is, it could not be shaped at any temperature below its melting point by either hammering or rolling.

In order to utilize the high-carbon product of these furnaces for making forged or wrought articles, it was necessary to develop purifying processes that would remove the excess carbon, manganese, silicon, etc., from the impure iron to produce relatively soft, malleable wrought iron that would have the same general composition and characteristics as the iron formerly produced directly from the ore in the Catalan and similar processes. As might be expected, a very great number of methods were developed in different localities. Two types of processes eventually became prominent: the charcoal-hearth processes and the puddling process.

Since the production of wrought iron from ore by any of these processes involved two separate steps: (1) reducing the ore to make pig iron and (2) remelting and purifying the pig iron to make wrought iron, they were referred to as **indirect processes**.

Some of the most widely used charcoal-hearth processes for purifying pig iron are described below (the Walloon, South Wales and Lancashire processes).

Walloon Process—Just how, when, where, and by whom wrought iron was first produced from pig iron is unknown, though it is probable the process originated in Belgium. The first attempts were, no doubt, made in the forge or on a hearth such as those already described for the production of iron directly from the ore. Here the action of the air from the blast (by that time in general use) would, if the iron were handled properly during melting, result in the oxidation of the silicon and the greater portion of the manganese and carbon, giving a ductile and workable product. The first reference to the process in written records appeared about 1620, but by that time the process had reached a stage of considerable development. Previous to that date, the

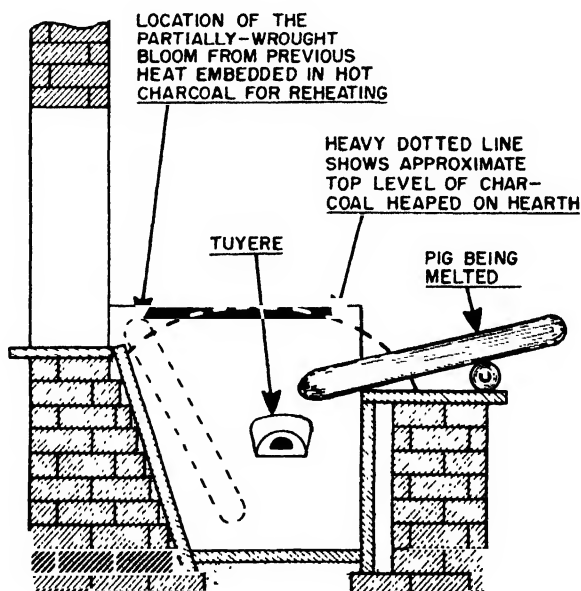


FIG. 11-4. General arrangement of a Walloon hearth used for purifying pig iron to make wrought iron.

Walloons of Flanders had gone to Sweden, where they had introduced the process, since known as the Walloon process. In this process a rather deep hearth with one or two tuyeres was used (Figure 11—4). With the hearth filled with charcoal and heated to a high temperature, the pig iron, in the form of long pigs, was fed into the fire so that the lower end of the pig would be gradually melted, and the molten metal would trickle to the bottom directly in front of the blast. The metal, desiliconized and decarburized by the oxygen in the air blast, would collect as a pasty mass upon the bottom, being worked vigorously as it collected. The ball of pasty metal was then separated into lumps that were raised above tuyere level and remelted. The new ball formed on the bottom was then removed from the hearth and hammered into a bloom. The second melting freed the metal from much of its entrapped slag. The pig used in Sweden, since it was reduced from the famous Dannemora ore in charcoal furnaces, was exceptionally low in silicon, sulphur and phosphorus, hence was especially adapted to this process.

South Wales Process—Few districts outside of Dannemora are favored with ore so free from phosphorus, or were able to continue using charcoal for fuel for such a period of many years. The use of coke in the blast furnace leaves no alternative but the production of high-silicon iron, if the sulphur content is to be kept suitably within limits. At the lower temperatures necessary to produce low-silicon iron, more of the sulphur content of the coke will be picked up by the iron. Such iron, high in sulphur and silicon, could not be purified in a single operation, as in the Walloon process, where the purification was carried on in the combustion chamber with the metal and slag in contact with the fuel. It was found, however, that this iron could be purified and converted into wrought iron very readily in two stages. The South Wales process (sometimes confused with the Lancashire process) was a typical two-stage process. For the first stage, a small, rectangular, water-cooled hearth, surmounted by a stack and provided with a number of tuyeres, was used. In some cases, this hearth was a separate structure; in other cases this hearth for melting the pig iron formed part of a two-hearth furnace in which the melting hearth was slightly above the second hearth where final refining took place. When a separate melting hearth was employed, coke was used as fuel to melt the pig iron; for refining the melted iron, the second hearth was fired with charcoal. When the two hearths were incorporated into one furnace structure, charcoal was used as fuel in both. Sometimes two charcoal hearths were served by one melting hearth that tapped directly into them. The hearths were known by various names. The melting hearth, when separate, was called the **refinery**, or **refinery fire**, if the metal tapped was allowed to partially or completely solidify before being transferred to the second hearth. If the metal was allowed to flow directly from the refinery into the second hearth or hearths, as in the two-hearth furnace, the melting hearth was known as the **melting finery** or **running-out fire**. In both cases, the second hearth was known as the **finery**, **charcoal finery** or, more often, **knobbling fire**.

With a good fire burning upon the melting hearth, alternate charges of coke and pig iron were made upon it. As the metal melted, it would collect upon the bottom of the melting hearth where the blast from the tuyeres impinged upon it, oxidizing the silicon and some of the phosphorus along with a part of the iron. Assuming that the melting hearth in this case was a separate unit, when a sufficient quantity of partially purified and partially solidified metal had collected, it was transferred to the

second hearth from the melting hearth, being piled in front of the tuyere and completely remelted while exposed to the blast. During the remelting, the metal was worked constantly and repeatedly raised slightly off the bottom, which treatment promoted the oxidation of the carbon. As the carbon was being removed, the metal gradually assumed a pasty condition, when it was worked into a ball, taken from the furnace, and hammered.

Lancashire Process—The Lancashire process differed essentially from the South Wales process, to which its name was sometimes loosely given, in that the pig iron was both melted and refined in a single hearth using charcoal as fuel. With some of the slag left from the previous operation to cover the bottom, the hearth was piled with charcoal up to above the tuyeres. The pig iron in lumps was placed on top of the charcoal pile, covered with more charcoal, and the blast turned on. The pig iron melted in drops which became partially decarburized in passing through the tuyere area and collected on the bottom. When all of the pig was melted, it was worked with bars to mix it with the slag and become thoroughly purified. As purification proceeded the metal became stiff and pasty, and when purification was completed the pasty mass was raised above the tuyeres and melted down again to free it from the intermingled slag. The pasty lump resulting from the remelting process was then taken from the furnace and hammered into a bloom.

These three are only a few of the many types of charcoal-hearth indirect processes developed for purification of pig iron to produce wrought iron.

Hand Puddling Processes—About 1613, Rovenson invented the reverberatory furnace, which he described as a bloomery, finery or chaffery "in which the material to be melted or wrought may be kept divided from the touch of the fuel," but it was not employed for purifying pig iron until 1766, when the Crane brothers received a British patent on a process which later came to be known as **puddling**. With careful manipulation of a reverberatory furnace, they were able to convert "white iron," or pig iron from which most of the silicon and phosphorus had been removed in a refinery, as described under the "South Wales Process," into a good malleable form of iron by the use of raw coal alone for fuel. In 1784, Henry Cort hollowed out the bottom of the furnace so as to contain the metal in the molten state, then by agitating this "puddle" or bath of metal with an iron bar or paddle he was able to convert white or partially-refined pig iron into a malleable form (wrought iron), the carbon being burned out by the oxidizing gases of the furnace atmosphere.

As the furnace bottom was made up of sand, it was rapidly fluxed away by the iron oxide formed. Besides, the process consumed much time and was wasteful of iron, the yield being less than 70 per cent of the metal charged. These objectionable features were largely overcome by Joseph Hall, who, in 1830, substituted old bottom material for the sand, thus introducing the iron oxide bottom, which adapted the process to any grade of iron, shortened the time of the heats, and increased the yield to about 90 per cent. On account of the boiling action of the bath caused by the rapid oxidation of the carbon by the oxides on the bottom, Hall's process came to be known as the **pig boiling process**. Later, this process became the leading method for the production of wrought iron.

The original method was designated as **dry puddling** because of the small quantity of slag formed, the slag-forming impurities having been removed in the refinery (see "South Wales Process," above). Hall, or his asso-

ciates, also introduced the use of air-cooled iron plates for supporting the bottom and sides, which materially increased the life of the furnace. During the next 30 years, few changes were made in the process, for the new process was so far superior to previous ones that there was left little incentive for improvement. This attitude was changed, however, with the introduction of the pneumatic, or Bessemer, process in 1856. Then, in order to overcome competition of the Bessemer steel, and incidentally lessen the labor of puddling, which, like all its predecessors, was very arduous, hundreds of attempts were made to improve and cheapen the process (see Section 8, "Mechanical Puddling"). Few of these attempts were successful, and even the most promising

of the successful ones, for various reasons, failed of universal adoption.

Between the years 1920 and 1930, however, hand puddling was almost entirely abandoned to be supplanted by the Ely mechanical puddler and the Aston process, the former duplicating conditions of hand puddling as closely as possible and the latter employing radically different principles and methods to obtain a similar but more uniform product. These three methods are briefly described in the following sections not only because they will help to define wrought iron and illustrate different methods of producing it, but also because they supply some fundamental knowledge as an introduction to the study of steelmaking processes.

SECTION 4

CONSTRUCTION OF THE HAND PUDDLING FURNACE

Although various modifications were introduced in the construction of puddling furnaces, affecting both size and design, the tendency was to adhere to the smaller and simpler types, such as the one shown in Figure 11-5. This type was known as a single furnace, had a

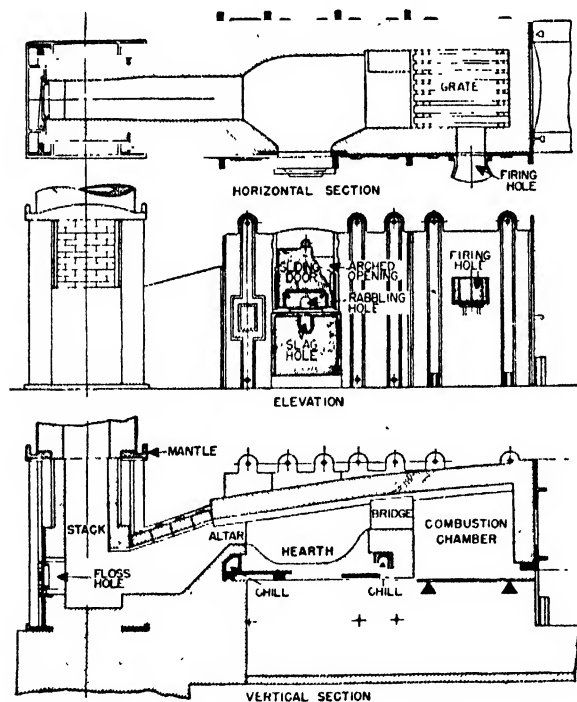


FIG. 11-5. Diagrammatic sections and elevation (center) of a typical hand puddling furnace of the simple design known as a single furnace.

capacity rating of 500 pounds per heat, and was coal-fired. The furnace was made up of three parts: the grate, or fireplace, located at one end of the furnace; the neck, or flue, at the opposite end, leading to the stack; and the hearth, or puddling basin, centrally located between the grate and the neck. The furnace was constructed entirely of brick, but was encased on the sides by a shell of iron plates held in place by tie rods. As the furnace was of the reverberatory type, all these parts were covered by an arched roof which sloped

from the fireplace to the uptake flue at an angle of 8 to 10 degrees. The roof over the fireplace was built of firebrick, but usually silica brick were used over the hearth and neck. The fireplace, which measured about $3\frac{1}{2}$ feet in length, $2\frac{3}{4}$ feet in width, and $3\frac{1}{4}$ feet in height at the rear, was enclosed on each side by 12-inch firebrick walls and at the rear by a 9-inch wall of the same material. To support the fire bed the space over the ash pit was bridged with iron bars. About 16 inches above the bars a 10-inch square hole in the firebox on the front side of the furnace was provided for firing. The neck, at the other end of the furnace, was an inclined firebrick flue, frequently lined with a course of best-quality silica brick. The neck terminated in a short uptake, or vertical flue, that led to the stack, which was independently supported upon a mantle. At the base of the uptake, directly opposite the neck, was an opening or door, called the floss hole, which was provided primarily for the removal of the cinder that was carried, or overflowed, from the puddling basin.

The hearth, or puddling basin, was the most vital part of the furnace. Externally, the bottom of this basin consisted of three iron plates, 1 inch thick, which were supported upon four heavy bearer bars laid transversely across the space between the side walls of the furnace. This construction provided all the benefits of air cooling. A low brick wall, laid across the furnace and known as the bridge, separated the hearth from the fireplace and also served as a backing for one end of the basin. At the opposite end of the hearth, a somewhat lighter and lower wall, known as the breast wall or altar, separated the basin from the neck. Imbedded in each of these walls next to the lining, was a hollow iron casting, called a chill, through which air or water was circulated to keep these parts cool. The other two sides of the basin were supported by the walls of the furnace itself, and were similarly air-cooled. The back wall was built up solid to the roof, but the front wall contained the arched opening to the hearth. The sides of this opening were made of specially formed silica brick, known as the jambs, while its bottom was made of a heavy iron plate called the fore plate. This opening was closed by a brick-lined sliding door, in the bottom of which was a small U-shaped opening, the rabbling hole, through which tools for working the heat could be inserted without raising the door.

Before the newly-built furnace was put into productive operation, a smooth one-piece working bottom or basin was built up in the hearth. The details of this operation were varied considerably with equally good results. In general, the bottom was composed of a

refractory fettling composed mainly of the ferrosiferrous oxide of iron (Fe_3O_4). Certain grades of ore or of heating-furnace cinder were frequently used, but more

often the bottom was made by applying, oxidizing, and fritting in successive layers of fine iron cuttings (such as thread cuttings from a pipe mill) known as swarf.

SECTION 5

OPERATION OF THE HAND PUDDLING FURNACE

With the hearth properly built up or repaired and the furnace in good working order and at a proper temperature, about 500 pounds of pig iron were charged by hand through the door. Following this operation, occupying 2 to 3 minutes, the purification of the pig iron and the process of puddling advanced by stages, known as melting, clearing, boiling, balling and drawing. To achieve quick melting, the door and other openings were closed, the furnace was fired vigorously, and the pigs turned once or twice by the puddler or his helper. In this way the charge was usually melted within 20 to 25 minutes after charging. The molten metal, covered with a thin layer of slag, was then stirred or rabbled by the puddler to hasten the oxidation of the silicon, the manganese, and a part of the phosphorus, an operation known as clearing and requiring 8 to 10 minutes. As soon as the metal had cleared, as revealed by a change in its appearance, the puddler endeavored to bring on the boil by raising the temperature of the furnace, charging some dry roll scale (iron-oxide scale detached from bars in rolling), and stirring the bath vigorously. After some 8 to 10 minutes of strenuous effort, the oxidation of the silicon and manganese was brought to a point where the carbon could also be oxidized.

As the product of this reaction was the gas, CO, and since the slag was somewhat viscous, the action caused the latter to foam or boil and rise in the furnace. At this point the slag was permitted to flow from the furnace freely unless it was desired to hold the phosphorus high in the iron, when a little coal was added from time

to time and as much of the slag as possible was held in the furnace. As the elimination of carbon became more rapid, the gas would escape in larger bubbles and burst into flame at the surface of the slag to form small flames called puddler's candles. With the disappearance of the candles, the puddler increased the stirring of the bath during the lowering of the heat until the metal in terms of the puddler, would come to nature. In this phenomenon, most characteristic of puddling, the metal appeared in small globules, like butter in churned milk, each globule representing a portion of the iron that had become decarburized. As this reaction neared completion, the bath became pasty and very hard to work. This change occurred because the high-carbon pig iron, which was molten at that temperature, was converted to low-carbon iron, which is solid (though pasty) at the same temperature. The change progressed rapidly, lasting only 6 to 8 minutes, so that in some 30 to 35 minutes after clearing the metal was ready for balling.

The globules agglomerated by the rabbling tended to collect in masses on the bottom, which had to be raised constantly and exposed to the heat to prevent them from freezing to the bottom. So the temperature of the furnace was raised as high as possible, and the metal was worked into a mass which was next separated into three parts or balls of about 150 pounds each, a size convenient for handling with tongs. This operation required about 15 minutes. Each ball in turn was then grasped by the tongs supported from the trolley (Figure 11-6) and drawn through the door. After the last ball was re-



FIG. 11-6. Puddler removing a ball from a puddling furnace for transfer to the squeezer. This photograph, taken in 1949, shows one of the last puddling furnaces still in operation in the United States. (Courtesy, Lockhart Iron & Steel Company.)

moved, the furnace was permitted to cool to some extent, and the bridge and breast were covered with a special ore mix. Any necessary patching of the bottom

was done, and another charge of metal was placed on the hearth for the next heat. These operations required about 2½ hours from heat to heat.

SECTION 6

ROLLING OF HAND-PUDDLED WROUGHT IRON

Shingling or Squeezing the Ball—At one time the balls were worked into the form of a rough bloom with a hammer, an operation called *shingling*. Later, hammering was superseded by the use of a device known as a *squeezer*, of which there were different types. One, known as the Burden squeezer, was most used (Figure 11-7). It was of the rotary type and consisted essentially

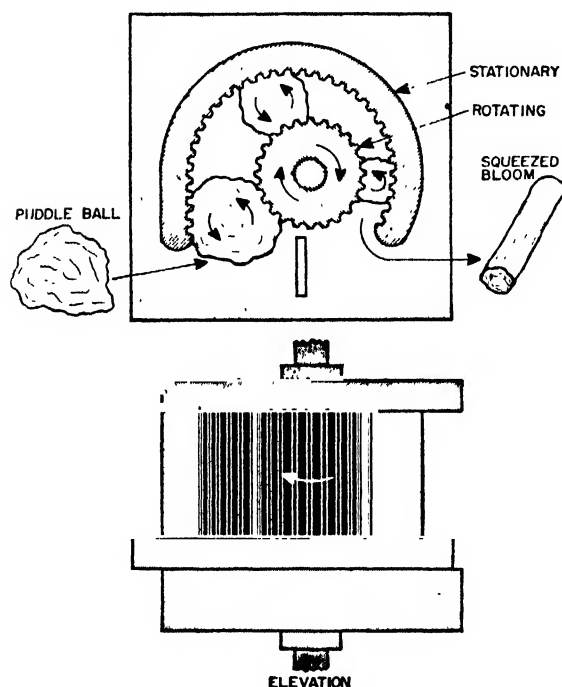


FIG. 11-7. (Above) Schematic representation of the path of a puddle ball passing through a Burden squeezer to reduce it to a crude bloom for subsequent rolling. (Below) Elevation of the Burden squeezer.

of a toothed cylinder mounted upon a vertical shaft so as to revolve within a section of a somewhat larger stationary cylinder set eccentrically to the revolving cylinder. Since the larger cylinder bore teeth or cogs on its inner surface and described only about three-fourths of a circumference, a ball placed in the larger opening between the cylinders was carried around the circumference by the smaller revolving cylinder to be compressed and discharged through the smaller opening in the form of a short round bloom. This action squeezed most of the excess slag out of the ball and compressed it into a form more suitable for rolling. As soon as the ball was delivered by the squeezer, it was grasped with tongs and at once delivered to the first pass of the rolling mill.

Rolling the Squeezed Ball—The muck bar mill for rolling the squeezed ball into muck bar was usually a 16- or 20-inch mill and consisted of a single stand or set of rolls, or of two stands in train; that is, end to end and

coupled together. The first pass was of the open-box type * with large fillets at the corners in order to take the cylinder from the squeezer, while the second pass, generally an edging pass, was of similar design, thus working the metal into the form of a round-cornered square and squeezing out more of the slag at each pass. The remaining passes were all of the closed-box or tongue-and-groove type for rolling the bloom into flats called *muck bars*, with some open-box edging passes for use in producing the narrow widths. The size of the muck bars, of course, was regulated by the product to be made from them and the manner or system used in forming the product. For ordinary bar iron, which was the chief product, the muck bar was usually about ¾ inch thick and 2½ to 8 inches wide. Bars of these sizes required from 5 to 9 passes. On account of the rapid cooling of the bar in the rolls, it was not practicable to attempt to roll sizes smaller than these, as the slag was no longer fluid enough at this stage to be worked out of the bar. As slag was squeezed out of the bar at all passes in the mill, the muck bar had a very rough surface with some torn edges and was otherwise unfit to do service as a finished bar. Having been rolled to the size required, the muck bar was allowed to cool before being subjected to further treatment.

Variables in the Muck Bar—Owing to irregularities in the pig iron used, differences in manipulation by different puddlers, and in different plants, the small quantity of metal refined with each heat, and the fact that the metal solidified before purification was complete, muck bar was an exceedingly variable product. Since the retention of the characteristics of wrought iron did not permit melting, this variation had to be overcome through heat and mechanical treatments. To effect the necessary refinement, two methods were used, known as busheling and piling, followed by rolling.

Busheling—Obviously, the surest way to obtain a thorough mixing of the iron, was to shear the muck bars into small pieces—the smaller the better. These small pieces from the different muck bars were allowed to collect in a pile, or piles, from which portions weighing 180 to 600 pounds were removed with a scoop or fork and charged into a reheating furnace, called a *balling furnace*, where they were heated "white hot," or to a self-welding temperature. With a paddle, these pieces were then collected into a ball, similar to a puddle ball, which was squeezed or shingled, then rolled or hammered into a bloom, which was then reheated and worked into the form desired. This process, known as *busheling*, was used for working up muck bar only when iron of the highest quality was desired. The process was also used in working up small scrap. In this case there was no necessity for shearing, and the cost of the scrap was usually considerably less than the cost of muck bar, but unless the scrap was very carefully selected it was liable to contain much steel, in which case the iron produced was considered to be of inferior grade.

Piling—The more common practice, therefore, was to

* See Part 1 of Chapter 23 on "Rolling Mill Rolls and Their Parts" for descriptions of the roll passes mentioned here.

shear the muck bar into lengths of from 2 to 3 feet, then arrange these pieces in piles of from 5 to 7 or more each and bind the pieces together with wire or bands. The piles were carefully charged into a furnace and heated white hot. The high temperature caused the different bars to weld together, so that they could then be removed and rolled into bars. The first 2 or 3 passes squeezed out more and more of the liquid slag, but in the last passes the bar had cooled to a point where the slag was merely plastic and would not flow. Thus, a fairly smooth and uniform bar was produced, which would be sold as **merchant bar**, **single-rolled iron**, **single-refined iron**, or **No. 2 iron**. In order to attain the highest degree of uniformity, particularly with respect to distribution of slag fiber, the once-piled or so-called single-refined bars were in turn cut up into short lengths and repiled and rerolled.

Double Refining—To further improve uniformity, then, merchant bar was cut up into short lengths, fagoted, reheated, and rerolled to produce the products known as **double-rolled iron**, **double-refined iron**, **best bar**, or **No. 3 bar**. The manner or fagoting (binding together into a bundle) or piling these bars varied in numerous ways, and depended only in part upon the use to which the iron was to be applied. Therefore, each manufacturer generally had his own methods of fagoting, which imparted to his iron an individuality detected by etching. Some of the more common methods of fagoting are illustrated by the accompanying sketches (Figure 11—8). When these fagots were heated and rolled into bars, more slag was expelled, the bar was made more uniform in composition, and the fibers were much elongated and reduced in size. As a result, the bars showed an improvement in mechanical properties, including both strength and ductility. There was a limit, in addition to the factor of cost, however, to the number of times the iron could be worked to advantage. After five or six workings its physical properties began to be lowered, and the bars decreased in strength and were less ductile. The cause, or causes, of this change was

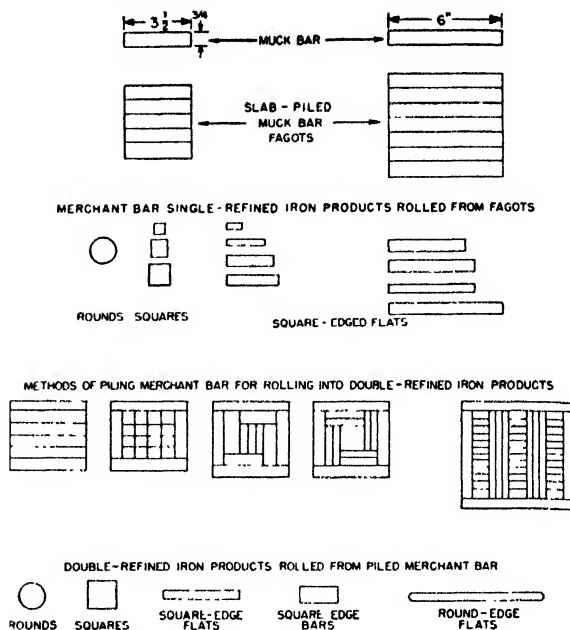


Fig. 11—8. Some of the more common methods of fagoting or binding together piles of single-refined wrought iron bars prior to heating for re-rolling to produce double-refined wrought iron bars.

questionable; probably, it was due to the elimination of much of the silicate slag or possibly too much reheating and rerolling caused the ferrous silicate fibers to become oxidized to the ferric condition, thus destroying some of the characteristics of wrought iron. Wrought iron in this condition was often referred to as **dry iron**.

SECTION 7

REACTIONS AND PROCESS LOSSES IN HAND PUDDLING

The changes in composition of the pig iron during puddling involved the elimination, or oxidation, of silicon, manganese, phosphorus, and carbon about in the order mentioned. In these reactions, the oxidizing agents were FeO and Fe_3O_4 . That Fe_2O_3 played little, if any, part in the reactions was evident from the fact that Fe_2O_3 decomposed at temperatures above 1010°F (1100°C) to form Fe_3O_4 . Also, it had been found that if hematite ore (Fe_2O_3) was used as the oxidizing medium, the boil came on very slowly; but if roll scale (Fe_3O_4 or $\text{FeO}\cdot\text{Fe}_2\text{O}_3$) were used, the reactions proceeded with much greater speed. Nearly all of the silicon and manganese and a part of the phosphorus were oxidized before the boil began, and at some period after melting and after the elimination of some of the silicon, the oxidation of all four elements, including carbon, might have proceeded simultaneously.

A study of the probable reactions that occurred in puddling indicates that a gain in weight of the puddled iron over the pig iron used could be expected, because iron was formed by reduction from the slag in the elimination of practically all impurities. By careful manipulation, furnaces could be operated to show a slight gain or a very slight loss. Nevertheless, in ordinary

working, there was a loss of from 3 to 6 per cent, which was sometimes a little more, at other times a little less, than the total of the impurities present. If the heat was properly handled, the loss was largely due to oxidation of iron after solidification had begun in the after-part of the boil and during the balling stage. If the heat was not skillfully handled, a variable part of the loss may have been due to the escape of the metallic granules with the "boilings" before the "heat was lowered." In reheating and rolling the muck bar, there was a variable loss of from 10 to 20 per cent for each time the iron was worked depending upon the number of times it was worked, the manner of the working, and other factors incidental to the operations of heating and rolling. In general, these losses were due to surface oxidation of the metal in heating and rolling, expulsion of the slag, and cropping. Slag expulsion was the smallest item of loss, except in the case of muck bar, and depended mainly upon the number of times the iron was worked, but was also affected by the temperature at which the iron was worked, and the nature of the incorporated slag itself. The loss was a little greater on iron with a high phosphorus content than on iron low in phosphorus.

SECTION 8

MECHANICAL PUDDLING

The never-ending competition in the iron and steel industry has been a constant spur to improve methods or lower costs of production. Just as the puddling process virtually eliminated the more primitive direct-reduction methods for the production of iron, so the Bessemer and open-hearth processes for the production of steel threatened the life of the wrought-iron industry. Even before the invention of these steelmaking processes, much attention was given to improving the puddling process, because the process was costly and the labor arduous, and the furnace was wasteful of heat. With a reverberatory puddling furnace of the type described earlier, from 2000 to 2400 pounds of coal were required to produce one ton of muck bar. This consumption of fuel was reduced somewhat by the use of double furnaces with enlarged hearths, but, since the application of regenerative and recuperative furnaces appears to have been impracticable, efforts along this line failed to achieve much in the way of lowering costs. The installation of waste-heat boilers in the stacks effected marked economies, and their use became general. To overcome the high labor cost, many attempts were made to carry out the puddling operations mechanically. Such was the situation up to about 1880 when the Danks puddling furnace appeared. From this time up to 1925, wrought iron lost ground to steel in spite of several efforts to revive it. In the meantime, however, it came to be recognized as a product with characteristic properties unlike those of steel, and in 1925 attempts to revive the industry were noted. These endeavors advanced along two lines, the one mechanical and the other metallurgical, the former aiming to duplicate the process of hand-puddling as closely as possible and the latter aiming to produce a material having all the characteristics of wrought iron through the use of the same metallurgical principles but applied in a manner entirely different from that of hand puddling. These two lines of effort are described under the headings of mechanical puddling and the Aston process, the latter representing a successful and most revolutionary method of producing wrought iron by A. M. Byers Company.

Principles of Mechanical Puddling—At first these mechanical puddlers took the form of stirring or rabbling appliances that could be attached to the top of the ordinary furnace. Because of the great variety of motions necessary in the different operations of charging, raising and stirring the heat, balling the iron, and drawing the balls, none of these were successful. The more successful attempts at mechanical puddling have involved a complete change in the design of the furnace, and some changes also in the process. These attempts have been too numerous to warrant description here. The furnaces themselves may be classified as follows:

1. The rectangular furnace that oscillated about a horizontal axis of rotation.
2. The circular flat-bottom furnace that revolved about an axis slightly inclined to the vertical.
3. The circular furnace with flat or troughlike bottom that oscillated about a horizontal axis of rotation.
4. The cylindrical furnace that rotated about a horizontal axis coincident with the center.
5. The cylindrical furnace that oscillated about a horizontal axis coincident with the center, or both oscillated and rotated about such an axis.

Furnaces built on any of these plans were made to puddle iron successfully, but those of the fourth and fifth types were most successful, partly on account of the facilities they afford for controlling the agitation of the metal, and partly because of the simplicity of their construction. The **Danks furnace**, somewhat widely used in this country from 1868 to 1885, was of the fourth type, while the **Roe furnace**, built and successfully operated in 1905, is of the third type. **H. D. Hibbard's furnace**, first operated on a commercial scale in 1921, was somewhat similar to the Danks furnace. The **Ely furnace**, patented by W. C. Ely, was first designed for busheling scrap but was later (about 1920) applied to the puddling of iron. It represents the fifth type, and, along with the Roe furnace, survived until recent years as two successful mechanical methods for making wrought iron.

SECTION 9

THE ASTON PROCESS *

From the descriptions given in the preceding sections, it is apparent that the basis of wrought-iron manufacture consists in refining the base metal to a close approach to pure iron, and incorporating therein an iron-silicate slag of desirable chemical composition in proper amount and distribution. Obviously, as has been brought out previously, several correlated steps are involved, quite distinct in nature and capable of separation, but carried out in the usual methods for hand or mechanical puddling as one interconnected operation. Departing radically from these former methods is the Aston process developed and put into large scale operation by the

A. M. Byers Company of Pittsburgh. In this process, metal refining, slag melting, and processing to form the slag-impregnated sponge ball are carried out as separate stages, each stage in a separate furnace or kind of equipment. The last stage is the crux of the process, and is based upon a positive and effective physico-chemical influence; namely, the change in gas solubility from a very high amount in molten iron to an amount practically negligible on solidification. This stage of the Aston process is carried out by pouring the refined metal in a continuous stream into a large volume of molten slag. The slag acts as a heat absorbing agent which effects solidification of the metal, with accompanying liberation of its dissolved gases, at a steady rate and with a force sufficient to disintegrate the plastic metal into a spongy mass, conforming in all particulars to the characteristics of high-quality wrought iron.

* Special acknowledgment is made of the assistance rendered by A. M. Byers Company, in the preparation of this section.

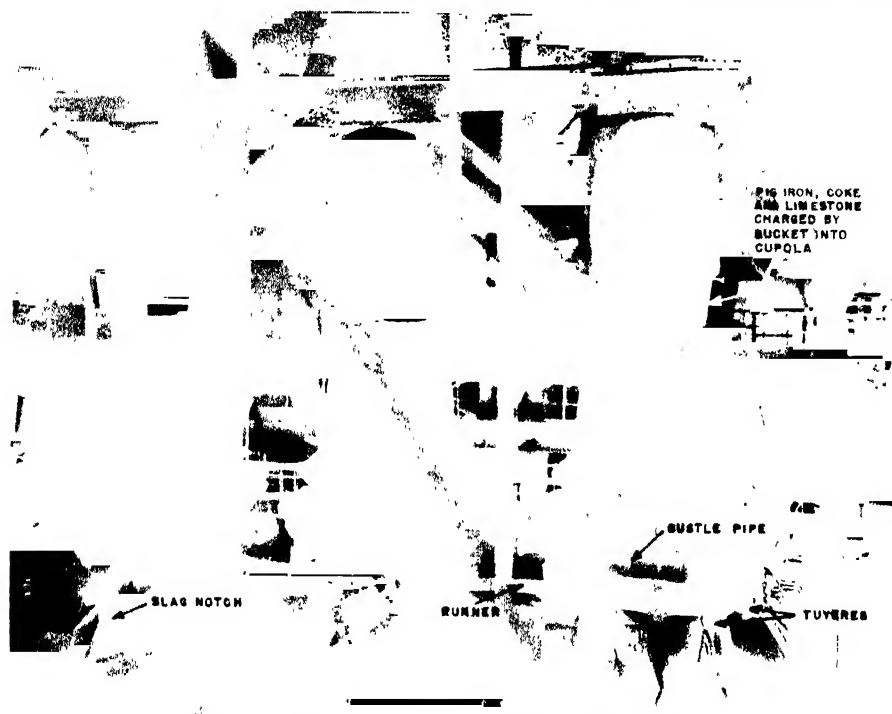


FIG. 11-9. Three cupolas engaged in the continuous melting of Bessemer-grade pig iron to produce molten iron for refining in the converter shown in Figure 11-11. (Courtesy, A. M. Byers Company.)

A large plant, with a daily capacity in excess of 1000 tons, is in operation in the Pittsburgh district. The essential features of the operation are illustrated in the accompanying Figures 11-9 to 11-14.

In the absence of hot metal from a blast furnace, standard Bessemer pig iron is melted in three cupolas operating continuously to produce molten iron for subsequent refining (Figure 11-9).

The iron-silicate processing slag is melted to exacting chemical requirements in special furnaces, which are described later (see also Figure 11-10), and then transferred to the processing cups.

Molten iron from the cupolas is blown to the low metalloid content of "full-blown" Bessemer steel in an acid-lined converter (Figure 11-11), and the highly refined metal is poured at an automatically controlled rate into a cup or thimble holding molten slag (Figure 11-12). After the metal from the converter has been

poured, the surplus slag is poured from the thimble, leaving the white hot sponge ball of wrought iron (Figure 11-13). This sponge iron is first pressed into a bloom of rectangular section (Figure 11-14) and is then rolled into slabs or billets.

Equipment for melting iron-silicate slag for the processing operation (sponge-making), and for incorporating in the matrix of iron, consists of four rotary furnaces. These furnaces, which are fired with oil or powdered coal, have no refractory lining and are operated in such manner that the lining material is composed of the same silicate slag that is being melted. This eliminates all contamination which would occur with customary refractory linings and makes it possible to produce the iron-silicate slag for processing to a very exacting chemical composition.

The key operation is effected by pouring the converter metal in a steady stream into a "processing cup" or

FIG. 11-10. Rotary furnace fired with pulverized or powdered coal for melting iron-silicate processing slags used in the Aston process for making wrought iron. (Courtesy, A. M. Byers Company.)





FIG. 11-11. Acid Bessemer converter in operation purifying molten pig iron to produce full-blown metal for the Aston process for making wrought iron. (Courtesy, A. M. Byers Company.)

vessel containing molten slag as illustrated by Figure 11-12. The metal ladle is automatically oscillated and moved forward and backward, insuring uniform distribution of metal into the slag. A sponge ball results as explained heretofore. The general size of bloom in present practice is three to four tons, and the pouring rate is one ton per minute.

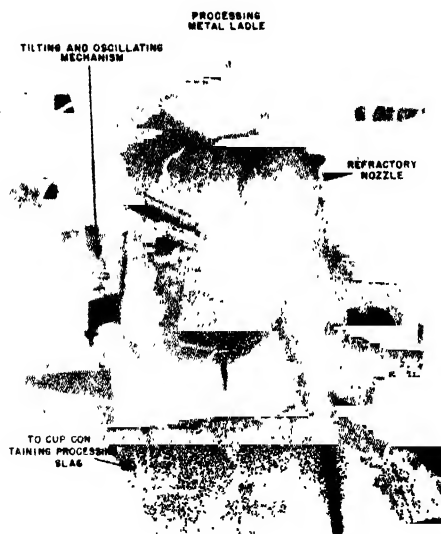


FIG. 11-12. Forming the slag-impregnated sponge ball characteristic of the Aston process for making wrought iron by pouring refined blown metal into molten slag. (Courtesy, A. M. Byers Company.)



FIG. 11-13. Decanting surplus slag from the processing cup, leaving the slag-impregnated, white-hot sponge ball of wrought iron. (Courtesy, A. M. Byers Company.)

These processing cups are on cars, so that the decanting of surplus slag and the dumping of the sponge ball (Figures 11-13 and 11-14) can be effected at a station remote from the pouring platform.

Pressing and rolling follow, using the original heat of the ball, to form intermediate products such as billets or slabs conforming to standard mill practice. Furnaces, mills, and auxiliary equipment, as well as man power, follow closely the standards of the modern rolling mill. In view of the large mass of the pressed bloom, most of the product—skelp, plate, etc.—is rolled from reheated solid sections, in marked contrast to older wrought-iron practice of building muck bar piles.



FIG. 11-14. White-hot sponge ball of wrought iron on the table of the machine which squeezes it into a rectangular bloom prior to rolling. (Courtesy, A. M. Byers Company.)

SECTION 10

COMPOSITION, STRUCTURE AND PROPERTIES OF WROUGHT IRON

Chemical Composition of Wrought Iron—Chemical composition has a place in the determination of wrought-iron quality comparable with its importance in the steel industry. One must bear in mind, however, that the customary metalloids may be, in greater or lesser degree, alloyed with the base metal or associated as oxidized constituents with the intermingled slag. The commonly reported composition of wrought iron lists the carbon, silicon, sulphur, phosphorus, and manganese of the composite mass. The ideal analysis should separate the metal from the slag, and reveal the metalloid distribution between the two. On the basis of analyses as commonly made, the following statements apply to wrought iron of high quality: Carbon content is generally 0.02 to 0.08 per cent, the lower percentages being more typically representative. Silicon content is 0.10 to 0.20 per cent, normally almost negligible in alloyed association with the metal, and existing almost entirely as silicates in the slag. Sulphur is always undesirable, and in well-made wrought iron, it should be under 0.03 per cent. A sulphur content of 0.025 per cent or under is quite common in quality wrought iron. Phosphorus is almost invariably higher in wrought iron than in steel. It must be borne in mind that phosphorus is in part dissolved in the base metal, and in part associated with the slag. Good wrought iron may have a phosphorus content of from 0.10 per cent or less to 0.25 per cent or more, according to manufacturer's preference, nature of raw materials, or adaptability to service conditions. The lower order is advisable for materials subjected to shock, high temperature, or requiring higher ductility. Manganese is usually under 0.05 per cent, due to refining conditions and to the fact that there are no after additions. This characteristic of real wrought iron has caused a manganese content of 0.10 per cent or more to be taken as indicative of steel scrap adulteration. Low manganese in wrought iron has usually been an earmark of quality, although there is no logical ground for condemning an otherwise well-made product because of a relatively high manganese content.

Macroscopic Structure of Wrought Iron—In view of the composite nature of wrought iron, its quality is ob-

viously affected by the nature of the association of base metal and slag. Methods of disclosing this internal structure have an importance greater even than the prominent place which the metallurgist assigns to them in the study of steel. Wrought iron exhibits a well-recognized fibrous fracture. The fracture test is a good over-all means for determining the general characteristics of wrought iron, but it should not be relied upon solely.

Macroscopic etching will reveal gross structure, reflecting such features of manufacture as methods of piling, and general slag distribution. Deep etching has a useful place but, like the fracture test, gives only a limited amount of information pertaining to the finer points of quality.

Microscopic Structure of Wrought Iron—Wrought iron consists essentially of a ferrite matrix, through which the slag is uniformly disseminated in the form of several hundred thousand filaments per square inch. Important disclosures of the microscope are:

- Grain Size**—Coarse grain, distortion, or lack of uniformity have a bearing upon quality in relation to mill history and use of product.
- Pearlitic areas** indicate the quantity and nature of distribution of the carbon, generally practically negligible or quite small in amount in real wrought iron.
- Slag—Type and Distribution**—Coarse, pocketed slag is undesirable. Finer textures result from progressive rolling reductions, and promote better mechanical properties, especially ductility.
- Chemical Composition**—The microscope is of no value in detection of alloying elements in solid solution in the base metal; for example, manganese, nickel, silicon, copper, etc.

Two photographs (Figures 11—15 and 11—16) illustrate typical structures and features related to the quality characteristics of wrought iron. These micrographs are of wrought iron produced by the A. M. Byers Company (Aston process), and show a typical longitudinal structure and a typical transverse structure.

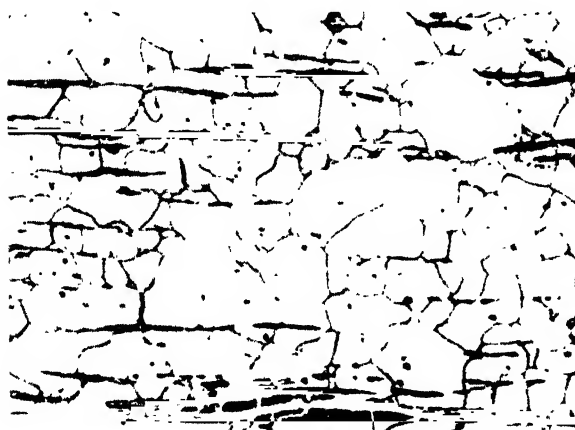


FIG. 11—15. Photomicrograph at 100X showing typical structure of wrought iron parallel to the direction of rolling. White areas are the highly refined iron matrix. Dark gray elongated lines are iron-silicate slag filaments. (Courtesy, A. M. Byers Company.)

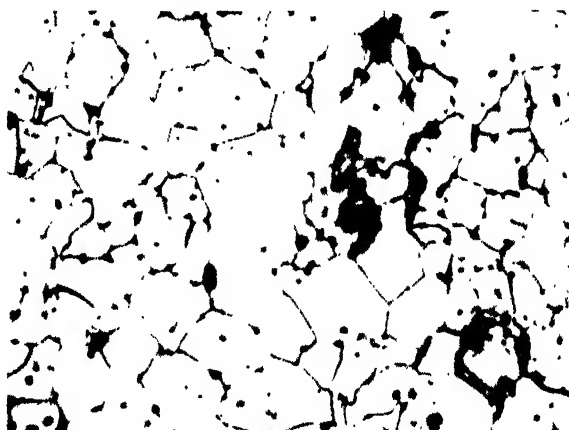


FIG. 11—16. Photomicrograph at 100X showing typical structure of wrought iron perpendicular to the direction of rolling. White areas are the highly-refined iron matrix. Dark areas are cross-sections of iron-silicate slag filaments. (Courtesy, A. M. Byers Company.)

Mechanical Properties of Wrought Iron—The mechanical properties of wrought iron are essentially those of pure iron, modified only slightly in general practice by metalloid content of the base metal and profoundly by the quantity and distribution of the incorporated slag. Up to certain limits, the ductility is increased by greater work in forge or mill, which causes a finer and more threadlike distribution of the slag. This is reflected in the common practice of the puddle mill of once or twice piling in products designated as "single" or "double-refined" iron. Obviously, a similar result will be achieved by rolling relatively large initial blooms into small final sections.

In comparison with steel or ingot iron, the longitudinal ductility of wrought iron is somewhat lowered, due to slag incorporation, while the transverse strength and ductility are markedly reduced. However, rolling history is an important factor with respect to quantity and direction of reduction.

The values below are representative of tensile properties for various wrought-iron products, compiled from standards of the American Society for Testing Materials. Because of the physical size of the products listed, only the longitudinal properties are reported, except for plate for which both longitudinal and transverse properties are given.

BAR IRON—SINGLE-REFINED

Under 1¼ Sq. In. Section

Tensile strength, lb. per sq. in. 48,000 (minimum)
Yield point, tensile strength factor ... 0.6
Elongation in 8 in., per cent. 25 (minimum)
Reduction of area, per cent. 45 (minimum)

BAR IRON—DOUBLE-REFINED

Under 1¼ Sq. In. Section

Tensile strength, lb. per sq. in. 48,000—52,000
Yield point, tensile strength factor ... 0.6
Elongation in 8 in., per cent. 28 (minimum)
Reduction of area, per cent. 48 (minimum)

The higher ductility accompanying greater work is reflected in the figures for double-refined material. For heavy sections, such as large diameter bars and forgings, strength and ductility requirements are somewhat lowered.

WELDED PIPE

Tensile strength, lb. per sq. in. 40,000 (minimum)
Yield point, lb. per sq. in. 24,000 (minimum)
Elongation in 8 in., per cent. 12 (minimum)

Herein are reflected the effects of high temperatures in welding, and the lessened stretch in testing tubular sections. However, where special precautions are taken in the making of pipe for bending purposes, the ductility figures are bettered in practice by several per cent in the elongation obtained.

PLATE

Under normal rolling practice, plate exhibits the maximum of difference in longitudinal and transverse properties. The A.S.T.M. specifications for usual rolling require plate meeting the following tensile properties:

Tensile strength, lb. per sq. in.
Longitudinal 48,000 (minimum)
Yield point, lb. per sq. in.
Longitudinal 27,000 (minimum)
Elongation in 8 in., per cent:
Longitudinal 14 (minimum)
Transverse ... 2 (minimum)

By proper attention to rolling practice, it is feasible to equalize the properties, so that a specification requirement of a tensile strength of 39,000 lb. per sq. in. (minimum) and an elongation in 8 in. of 8 per cent (minimum) in either direction may be obtained. This feature is of great value in producing plate for flanging or other forming purposes.

Chapter 12

THE MANUFACTURE OF PIG IRON

SECTION 1

HISTORY, PRODUCTION AND KINDS OF PIG AND CAST IRON

No one knows when man first used iron, for that date belongs to prehistoric times. Meteorites probably supplied early man with the first specimens of the metal; knowledge of how to obtain iron from its ores came much later. Archeological research can establish only that iron has been in use through a period of about four thousand years. There is thought to be some evidence of its use by the Egyptians in building the pyramids, about 3000 B.C., but this use is questionable. As to its use by the ancient Hebrews, by the Assyrians about 1400 B.C., and by the Greeks, there is less doubt. The arts of the Greeks, which involved little iron, were followed by those of the Romans, who became somewhat proficient in its metallurgy. The Romans, through their numerous and extensive conquests, the success of which they no doubt owed to the use of metals in making their instruments of war, spread the art of extracting and fashioning iron throughout Europe. Some knowledge

of the metal, however, had preceded them, for Caesar, crossing the English Channel, found it in use among the native Britons. During the Roman occupation of England, the industry grew to one of importance. At that time, and for centuries thereafter, iron was obtained by heating ore and charcoal in a furnace or forge, until there had collected a small body of pasty metal which then was drawn and worked by hammering to make wrought iron. Such methods were described in Chapter 11.

It is also described in Chapter 11 how, around 1350 A.D., the ironmakers of Central Europe developed furnaces that would melt iron and permit of casting; this was iron with a high carbon content—cast iron. This result they accomplished in a new type of furnace, built of masonry, which enclosed a shaft or vertical opening in the form of two truncated cones placed base to base—in a crude way, the lines of the modern blast

THE BLAST FURNACE

THIS IS NOT AN ENGINEERING DRAWING. THE VARIOUS PIECES OF EQUIPMENT ARE NOT DRAWN TO THE SAME SCALE. THE SKETCH IS INTENDED ONLY TO SHOW THE MAJOR EXTERNAL FEATURES OF A BLAST FURNACE AND ITS AUXILIARIES, THE SOURCES OF RAW MATERIALS, AND DISPOSITION OF THE PRODUCTS.

INCOM

COKE PLANT

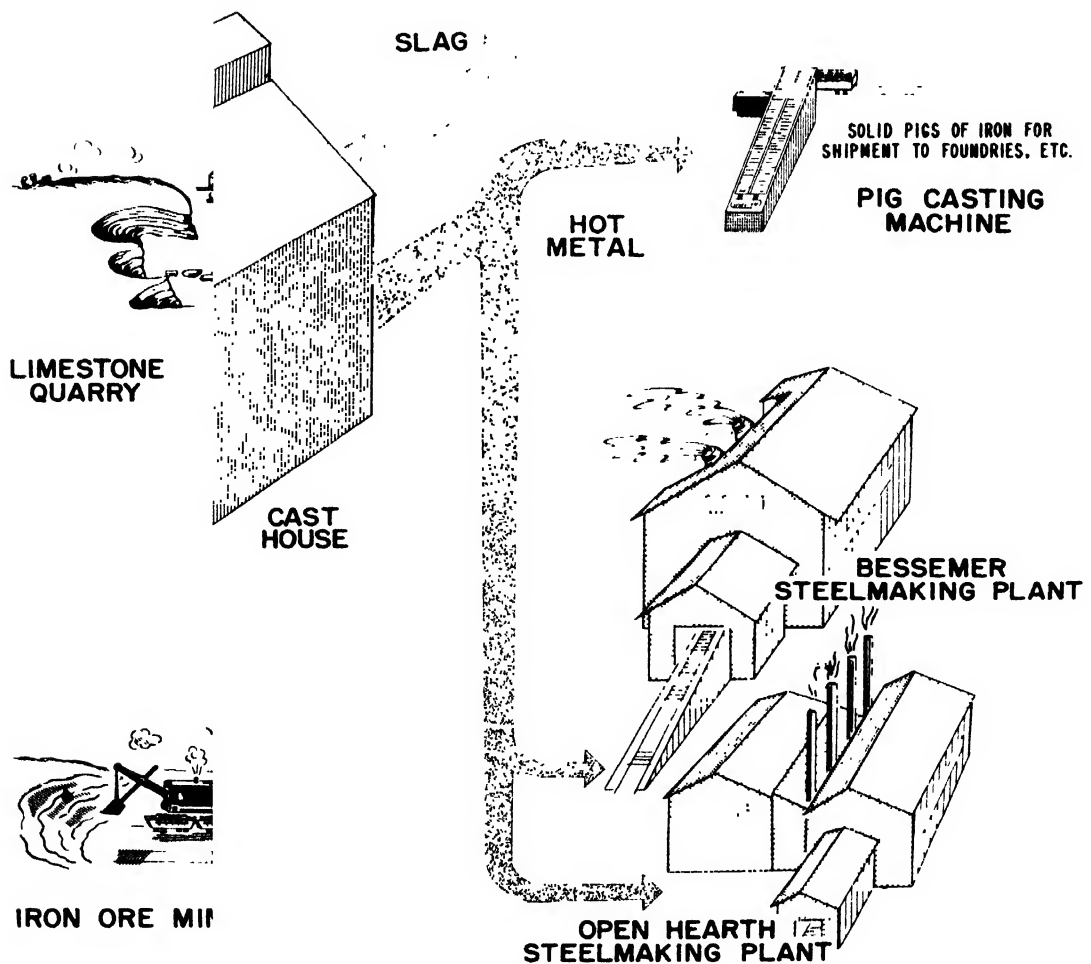
SLAG DUMP



CAST
HOUSE
TOP

SLAG PITS

WHERE SLAG IS PREPARED FOR CEMENT
MANUFACTURE OR VARIOUS COMMERCIAL USES



furnace. The lower frustum came to be known as the bosh and the upper frustum as the shaft. In this furnace, ore, flux, and charcoal were charged in at the top of the shaft, while air, under relatively very low pressure, was blown in near the bottom of the bosh. This method was introduced into England about the year 1500 where, in 1619, coke was first used as fuel, to be followed about 200 years later by the introduction of hot blast. In America, an iron works was established in Virginia on the James River, about 1619; this was destroyed in an Indian raid in 1622 and never rebuilt. The Hammersmith (now Saugus), Massachusetts iron works (recently restored) was begun in 1645 and was the first successful iron works in what is now the United States, not being abandoned until 1675.

The American furnaces of a period as recent as one hundred years ago now would be called very crude affairs. Portions of some of them still are standing. They were usually in the form of a truncated cone or pyramid, twenty to thirty feet high, and constructed of stonework which enclosed a shaft, about four feet across at the top and eight feet at the bosh. The hearth was either round or square in cross section. The capacity ranged from one to six tons a day. About the middle of the Nineteenth Century, the size of furnaces began to be increased, and by the year 1880, the output gradually had been raised to nearly 100 tons per day, with a daily coke consumption of nearly 300 tons. With all the basic

principles in use for so long a time, it may seem surprising that progress toward greater efficiency was made so slowly. About 1880, in the vanguard of the great industrial expansion, very rapid advancement began to be made, so that now there are furnaces, the daily pig-iron output of which exceeds 1,600 tons with a fuel consumption of less than 1,500 pounds of coke per ton of iron produced. A diagrammatic sketch of a modern blast furnace and its principal auxiliaries is shown in Figures 12-1 and 12-2.

Attention is called to these facts because it is well to remember, in beginning the study of the modern blast furnace, that the present method for the extraction of iron from its ores represents a very old pyrochemical process which only recently attained its present high state of engineering development.

The Importance of Pig Iron—Pig iron, besides being used directly in the form of castings, is the intermediate form of iron through which all commercial ferrous products, except sponge iron, must pass. Its importance is emphasized by Table 12-I, which summarizes yearly production for the United States during the twenty-five year period from 1930 to 1954, inclusive.

Kinds and Grades of Pig Iron—Pig iron is defined as the direct metallic product, either solid or molten, of a blast furnace smelting iron ore, though it is possible to make a similar material in other ways, for example, by melting steel scrap with an excess of carbon in an

Table 12-II. Chief Metallic Products of the Blast Furnace

Product	Maximum Range in Composition					Remarks
	Silicon (%)	Sulphur (%)	Phosphorus (%)	Manganese (%)	Total Carbon (%)	
Foundry Pig Irons.....	1.00 to 4.00	0.04 to 0.10	0.10 to 2.00	0.20 to 1.50	3.00 to 4.50	
Graded in steps of....	0.50	0.01	0.20	0.20		
Tolerance, variation from per cent ordered	±0.25	Only max. to be specified	±0.15	±0.20	Only the min. to be specified	
Most used grades:						
Northern Foundry .	1.75 to 2.50	Under 0.05	0.150 to 0.70	0.5 to 1.00	4.0 to 4.3	C not specified
Southern Foundry .	1.75 to 2.25	Under 0.05	0.7 to 0.9	0.40 to 0.75	3.75 to 4.1	C not specified
Malleable Pig, Standard	1.25 to 2.25	Under 0.05	0.1 to 0.19	0.4 to 1.00	3.75 to 4.30	
Graded in steps of	0.25	0.05	0.20	Not to be specified	
Gray Forge	1.20 to 1.75	Under 0.05	0.1 to 0.35	0.5 to 1.00	4.15 to 4.40	C not specified
Puddling Iron	0.75 to 2.50	Under 0.05	0.1 to 0.50	0.5 to 1.00	4.10 to 4.40	Cr < 0.01
Acid Pig, Bessemer	1.00 to 2.25	Under 0.045	0.04 to 0.1	0.5 to 1.00	4.15 to 4.40	C not specified
" Open Hearth.	0.70 to 1.50	Under 0.045	Under 0.05	0.5 to 2.50	4.15 to 4.40	C not specified
Basic Pig, Open Hearth	Under 1.50	Under 0.05	0.11 to 0.90	0.4 to 2.0	4.10 to 4.40	C not specified
" Bessemer or Thomas	0.3 to 1.00	Under 0.20	1.9 to 2.5	0.7 to 2.5	3.50 to 4.0	C not specified
Duplex Iron	1.2 to 1.75	Under 0.060	0.7 to 1.5	0.4 to 0.90	4.00 to 4.20	
Special Low Phosphorus Iron	Under 2.00	Under 0.035	Under 0.035	Under 2.00	C not specified
Spiegel (3 grades)	Under 1.25	0.03 to 0.05	0.14 to 0.25	16 to 30	5.0 to 6.5	C not specified
Silicospiegel	8.0 to 15.00	Under 0.02	Under 0.15	15 to 20	
Ferromanganese	0.5 to 1.00	Under 0.05	0.2 to 0.35	78 to 82	6.5 to 7.5	C not specified
Blast Furnace Ferrosilicon, Silvery Pig	6.00 to 17.00	Under 0.05	0.10 to 0.40	0.30 to 2.00	0.75 to 1.0	C not specified
Ferrophosphorus	1.5 to 1.75	Under 0.05	15 to 24	0.07 to 0.50	1.10 to 2.0	C not specified

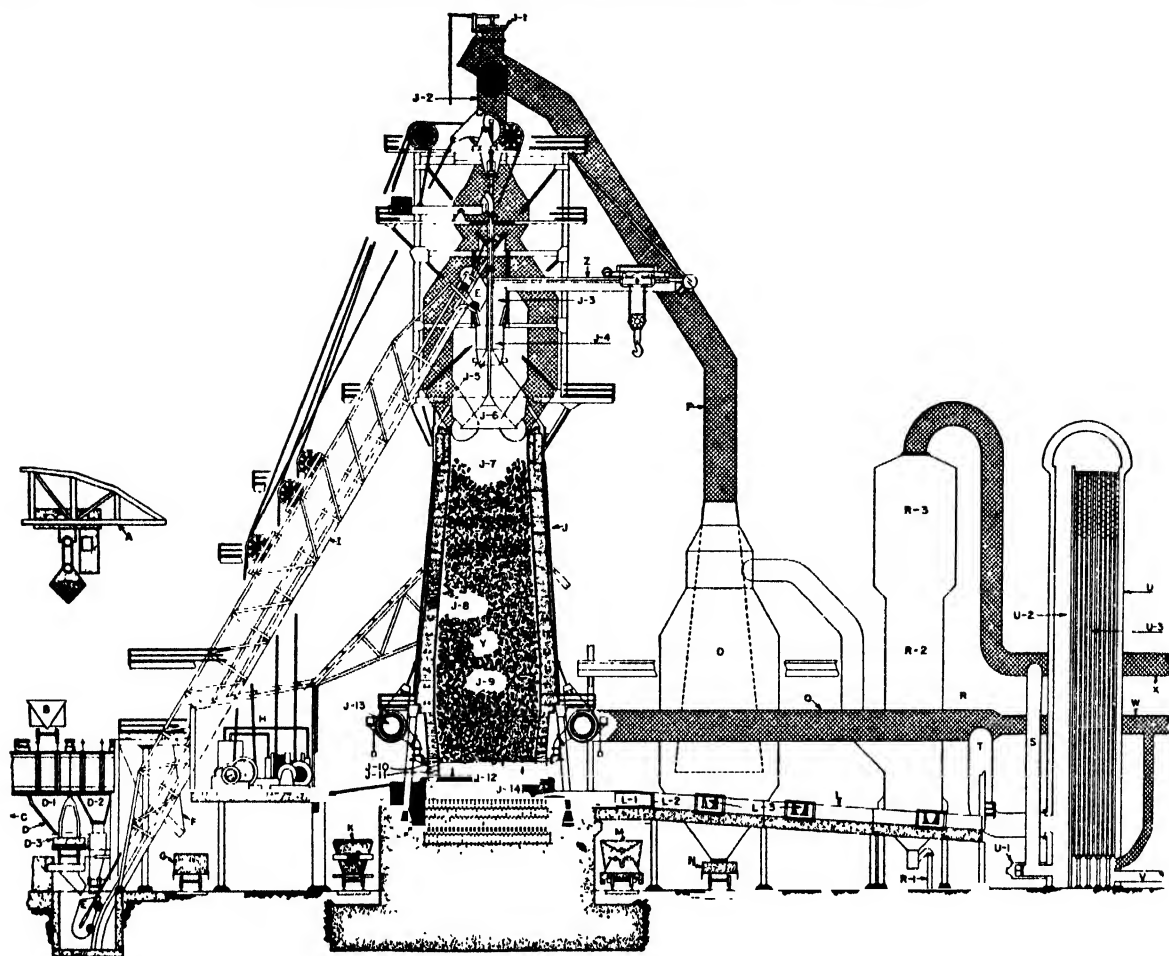


FIG. 12—2. Idealized cross-section of a typical modern blast-furnace plant. Details may vary from plant to plant.

Legend

- | | | |
|-----------------------------|--------------------|------------------------------------|
| A. Ore bridge | J-5 Small bell | P. Downcomer |
| B. Ore transfer car | J-6 Large bell | Q. Hot blast line to furnace |
| C. Ore storage yard | J-7 Stock line | R. Gas washer |
| D. Stockhouse | J-8 Inwall | R-1 Sludge line to thickener |
| D-1 Ore and limestone bins | J-9 Bosh | R-2 Spray washer |
| D-2 Coke bin | J-10 Tuyeres | R-3 Electrical precipitator |
| D-3 Scale car | J-11 Slag notch | S. Gas offtake to stove burner |
| E. Skip | J-12 Hearth | T. Hot blast connection from stove |
| F. Coke dust recovery chute | J-13 Bustle pipe | U. Stove |
| G. Freight car | J-14 Iron notch | U-1 Gas burner |
| H. Skip and bell hoist | K. Slag ladle | U-2 Combustion chamber |
| I. Skip bridge | L. Cast house | U-3 Checker chamber |
| J. Blast furnace | L-1 Iron trough | V. Exhaust gas line to stack |
| J-1 Bleeder valve | L-2 Slag skimmer | W. Cold blast line from blower |
| J-2 Gas uptake | L-3 Iron runner | X. Surplus gas line |
| J-3 Receiving hopper | M. Hot-metal ladle | Y. Stock—Iron ore, coke, limestone |
| J-4 Distributor | N. Flue dust car | Z. Jib boom crane |
| | O. Dust catcher | |

electric furnace. The transfer and use of the pig iron in the liquid state is common for steelmaking; in this form it is referred to as hot metal. When it is desired in small shapes, or pigs, for convenience in handling in the cold, solid form, it generally is cast into traveling molds attached to an endless conveyor forming part of a pig machine or pig-casting machine, and the name pig iron is retained. The qualifying phrase "smelting iron ore" is required in the definition to distinguish the

blast-furnace operation for pig iron from its operation for the other products of the blast furnace, such as ferromanganese, spiegeleisen, silicospiegel, ferrophosphorus, and alloy iron, in which cases manganese ore or iron ore mixed with other ores is charged. "Direct metallic product . . . of the blast furnace" distinguishes pig iron in any form from cast iron, which may be made solely of pig iron, with or without refining treatments, or from mixtures of pig iron with steel, or with other

alloying elements, such as copper, nickel, and chromium. Pig iron is the crudest commercial form of iron, but is made nevertheless in many grades as shown in Table 12—II, along with other products of the blast furnace.

Chief Metallic Products of the Blast Furnace—The chief metallic products of the blast furnace may be classified as follows:

(1) according to raw materials used, as **charcoal pig**, **anthracite pig** (both now practically extinct), **coke pig**, **hematite pig**, **titanium pig**, **chrome-nickel pig**, etc.;

(2) according to the use for which it is intended, as **acid pig** (in the United States, **Bessemer pig**), for the acid Bessemer or acid open-hearth process; **basic pig**, for the basic open-hearth or basic Bessemer (Thomas) process; **forge pig**, or **puddling iron**, for wrought-iron manufacture; **malleable pig** for making malleable cast iron; and **foundry pig** for use in the iron foundry for making castings of various kinds; and

(3) according to chemical composition, as **silicon pig**, **low-phosphorus pig**, **high-phosphorus pig**, **alloy pig**,

spiegel or **spiegeleisen**, **silicospiegel**, **blast-furnace ferrosilicon** and **ferromanganese**.

Formerly, ordinary grades were judged for quality or suitability by the appearance of the fractured surface of broken pigs, the color of which may be white, silvery gray, gray, or mottled, with either small or large crystals in the last two. Although the fracture is a convenience in sorting or selecting iron, grading in the United States is now on the basis of chemical composition, the chief products of the blast furnace according to this classification being represented in Table 12—II. Study of this table shows that blast-furnace products fall into three general types: (1) **conversion pig irons** which are used to make wrought iron and the many varieties of steel, in which processes they lose their identity, being converted into an entirely different product; (2) **pig iron** for castings, the characteristics of which are discussed in detail in Chapter 18; and (3) **ferroalloys**, which contain a high percentage of one or more metallic elements and are used as **addition agents** to regulate the composition of other ferrous metals.

SECTION 2

OUTLINE OF THE BLAST-FURNACE PROCESS

Furnace Input and Output—Essentially, the blast-furnace process consists of charging iron ore, fuel (coke), and flux (limestone and dolomite) into the top of the furnace and blowing heated air (blast) into the bottom. A rough statement of the input of each material per ton of iron produced would be 2.0 tons of ore and iron-bearing materials, 0.9 ton of coke, 0.4 ton of a mixture of limestone and dolomite, and 3.5 tons of air. From these materials, there is produced 1.0 ton of iron, 0.6 ton of slag, 0.1 ton of flue dust, and 5.1 tons of blast-furnace gas. These figures are for present materials which are available to the territory that borders the Great Lakes and which produces about 75 per cent of the pig iron in the United States. As the purity of the materials used varies, the proportions of both the materials charged and the materials produced will vary accordingly.

Functions of the Charged Materials—The function of the ore, as well as the other iron-bearing materials, is to supply the element iron which, in turn, represents about 93½ per cent of the pig iron. The ore is usually in the form of the oxide, either hematite (Fe_2O_3) or magnetite (Fe_3O_4), although there may be small proportions of limonite ($2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$) or siderite (FeCO_3) present. Hematite represents the largest proportion and usually comes from the Lake Superior region. When chemically pure, hematite contains 70 per cent iron, but present ores contain about 50 per cent iron, the difference being represented by **gangue** which consists mostly of silica and alumina, about 12 per cent moisture, and 6 per cent chemically-combined water.

Other iron-bearing materials are mill-scale, sinter, slag from open-hearth furnaces or Bessemer converters, and scrap. Scale (Fe_3O_4) is a by-product of hot-rolling operations, which drops from the heated piece as it passes through the rolls. Sinter is blast-furnace flue dust (or a mixture therewith of fine ore) which has been agglomerated to form a granular, relatively coarse material. Open-hearth slag is a by-product from the steel-making process containing about 25 per cent iron and an excess of bases over acids, thereby replacing a certain quantity of basic fluxes, but increasing somewhat the total quantity of slag per ton of iron. Basic open-hearth slag also contains sufficient manganese to make

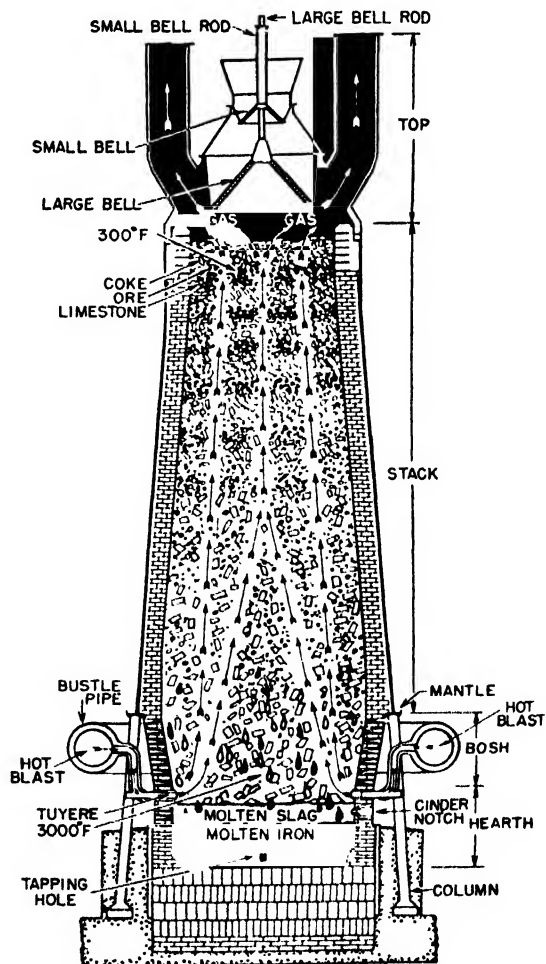


FIG. 12—3. Diagrammatic cross-section of a modern blast furnace, identifying principal structural elements, and indicating flow of materials.

it a valuable source of this element. Its use is limited to the production of irons wherein the phosphorus content is not raised too much thereby; also, it is not charged into blast furnaces producing "blowing iron" for the acid Bessemer process because the manganese it contains would contribute to making a too-fluid slag in that process by raising the manganese content of the iron to an undesirable degree. Converter slag, a by-product of the Bessemer process, does not contain the excess bases or phosphorus found in open-hearth slag. Scrap arises in the casting of the furnace, or it may be iron or steel from external sources and used in the blast furnace to increase production.

The function of the coke is two-fold; first, to supply enough heat to attain the necessary temperature for the metallurgical reducing reactions to take place; and second, to supply the reducing agent for the process of reducing or removing the oxygen from the iron oxide in the ore. Sufficient heat is necessary to reduce and melt practically all the iron, most of the manganese, and a small part of the silicon, and to flux and melt the gangue of the ore and the ash of the coke. The incandescent carbon of the coke is inherently a strong reducing agent, and it is thought, itself, to account for about 20 per cent of the reduction taking place in the blast furnace.

The oxygen of the blast unites almost immediately on entering the furnace with the carbon of the coke to form carbon monoxide, which accounts for the balance (perhaps 80 per cent) of the reduction taking place.

The function of the limestone and dolomite is to form a fluid slag of a composition which will restrict appropriately the other elements (silicon, sulphur, etc.) entering the pig iron. These so-called metalloids in turn also control, within limitations, the mechanical and chemical properties of the pig iron.

With this outline of the process of blast-furnace smelting, it is proper to pass on to the apparatus which makes this process possible.

Parts of a Blast-Furnace Plant—The heart of the ap-

paratus is, of course, the blast-furnace structure itself. The sections of the furnace are (1) the lowest part, known as the hearth, (2) the inverted frustrum of a cone immediately above it known as the bosh, and (3) the part known as the stack or shaft (see Figure 12-3). Means are provided at the bottom for the admission of the blast and removal of iron and slag; at the top, apparatus is provided for the charging of the ore, coke, and flux and for the removal of the gas generated in the process. The gas is led to the dustcatcher for the mechanical separation of the major portion of the dust; thence, through the primary and secondary cleaners it is conducted away for use in hot-blast stoves, in metallurgical heating, the generation of steam under boilers, or to operate blowing engines. The blast air is compressed by turbo blowers, steam engines, or gas engines, and driven to the stoves wherein it is heated regeneratively before its entry into the upper part of the furnace hearth. Bins for the storage and selection of the proper proportions of ore, coke, and flux are provided together with means for conveying these materials to the top of the furnace. Incoming materials arriving in railroad cars are either unloaded directly into the bins or by a car dumper from which they are handled for storage by an ore bridge. Incoming materials in boats are handled either by Hulett unloaders, belt conveyors, or modified ore bridges. The molten iron is carried away from the furnace in iron ladles which deliver it to the steelmaking or foundry departments for use in the molten condition, or to the pig machine where it is cast into small blocks for use in the solid condition. The slag is either disposed of molten in cinder ladles or granulated by a high-pressure water spray, in which condition it is used for fill or in the manufacture of cement. The flue dust, collected from the dustcatcher, the primary and the secondary cleaners, is usually recovered and sintered and then returned to the furnace. Besides these, the necessary utilities are steam, power, water, and railroad service as indicated schematically in the flow diagram reproduced as Figure 12-4.

SECTION 3

CONSTRUCTION OF THE FURNACE PROPER

Foundation—The foundation of a large, modern furnace, typified by Furnaces Nos. 11 and 12 at South Works, has to carry a structure which generally rises over 200 feet above the point of pig iron delivery (center line of the iron notch), has a side reaction from the weight of the skip incline and the downcomer pipes, and carries inside of the furnace a column of material about 90 feet high which will vary in temperature from 2800° F (1515° C) down to 300° F (190° C). Such a furnace foundation carries a load of about 20,000,000 pounds. Under the foundation, H-beam piles are driven to bedrock with the top of the piles about 10 feet below yard level. Upon the piles is placed a reinforced-concrete pad some 11 feet thick and 60 feet in diameter. A ring of reinforced concrete 10 feet high and 12 feet wide, with an internal diameter of about 36 feet, is placed upon the pad. The bases for the furnace columns rest on the top surface of the ring. The column bases are tied together structurally and the columns are welded to the column bases, thereby providing a structure which will not move or slip. The columns and bases are protected against a furnace breakout by a masonry sheath, one brick thick. Inside the circular wall or ring of concrete which supports the column bases, about 2 feet thickness of second-quality firebrick are laid—

the top course being leveled off to provide a suitable base for the lowest course of first-quality brick called bottom block that begins the construction of the actual hearth.

Columns and Base Plates—The column bases are 6-inch thick steel slabs, about 5 feet square. The top of each column is covered by a steel cap which is about 3½ inches thick and somewhat less than 4 feet square. The mantle rests on these caps. The columns are provided at both top and bottom with thick angle plates which are welded to the web of the column and to the caps and bases. Each column consists of a 14-inch H-beam core, to the flanges of which 3-inch thick cover plates are welded. The distance between the bottom of the column base and the top of the column cap is about 36 feet. On a large furnace, such as being described, there would be eight or ten columns.

Mantle—The mantle is a horizontal heavy steel ring resting on the column caps. It supports the furnace shell and the brickwork of the inwall or shaft so that, when necessary, the hearth and bosh brickwork may be removed without disturbing the inwall brickwork. It is built of heavy steel plates which are reinforced circumferentially by circular, heavy, welded-steel bands and by web plates.

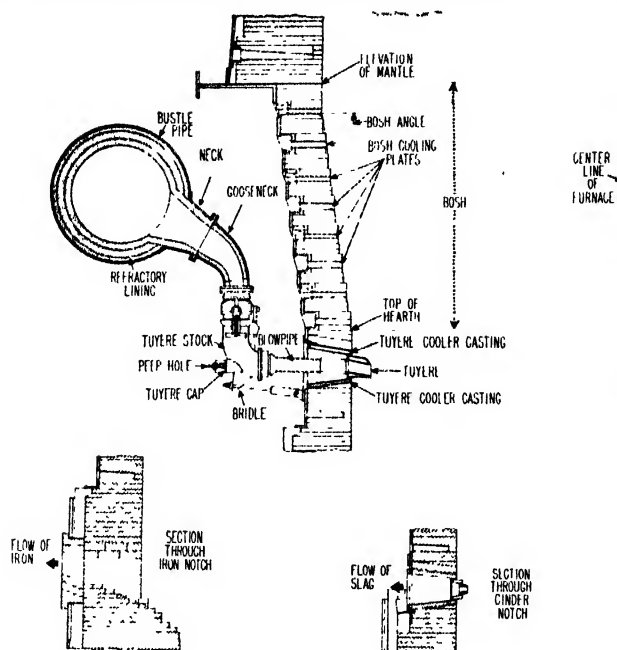


FIG. 12-5. Section through the bosh and upper part of the hearth wall of a large blast furnace showing details of connections between the tuyere and bustle pipe, and of the iron notch and cinder notch.

Shell—The shell of a blast furnace is in the form of a truncated cone with steep sides, topped by a flatter truncated cone called the **top cone**. The part below the top cone encloses the shaft of the furnace (Figure 12-3). Formerly, the shell was constructed in "shingle" fashion; that is, the edges of adjacent plates were overlapped and riveted. However, on the furnaces being described, the shell is an all-welded structure, with the edges of adjacent plates matched and butt-welded. This design gives the shell a continuous, relatively smooth surface free from lapped and riveted joints; it also uses less steel, is cheaper to erect, and is stronger than the previous construction. The plates used to construct the top and bottom parts of the shell usually are thicker than those used in the middle section, perhaps $1\frac{1}{2}$ inches compared to 1 inch. The shell is welded at the bottom to the mantle. Openings at selected regular intervals in the shell allow for the insertion of cooling plates into the brickwork with which the shell ultimately is lined. The bottom of the cone at the top of the shell is welded to the main shell structure. The top opening in this cone receives the **lip ring**, a heavy steel casting that may be of one piece or of joined segments. The lip ring is bolted to the top cone, and must be located concentrically; it supports the large bell hopper and the other filling equipment on the top of the furnace. There are regularly-spaced openings in the top cone to which the offtakes, usually four in number, are welded and which conduct the gas formed in the smelting process out of the furnace.

Hearth—It has already been mentioned that second-quality firebrick are laid in the concrete ring on the foundation to form a base for the bottom of the hearths of the furnaces being described. On top of this base, seven rows of first-quality 18 by 9 by $4\frac{1}{2}$ -inch fireclay bottom block were laid on end giving a total thickness of 10 feet, 6 inches. On top of these bottom blocks, two rows of large carbon blocks ($22\frac{1}{2}$ inches thick, 30 inches

wide and up to 15 feet long) were laid, giving a total thickness of carbon of 45 inches. Again, on top of the carbon a single course of fireclay blocks was placed on end. The sides of the crucible or hearth were constructed of two courses of carbon brick, with a total thickness of 33 inches, up to the center line of the cinder notch.

Not all modern furnaces employ carbon refractories in construction of the hearth; those that do not are built entirely of fireclay refractories. Neither do all furnaces using carbon refractories employ exactly the construction just described.

Located about 42 inches above the surface at the bottom of the hearth is the **iron notch** or tapping hole (Figure 12-5) through which the iron accumulated in the smelting process is drawn off at periodic intervals—usually four or five times per day. The slag or cinder, being lighter, floats on top of the molten iron and is drawn off at intervals through the **cinder notch** or **monkey**. The cinder notch assembly consists of three water-cooled, concentric, cast-copper frustrums of cones, the largest, known as the **monkey cooler**, supporting on its inside a smaller frustrum, known as the **intermediate cooler**, which in turn supports the **monkey** itself (Figure 12-5). The central opening in the monkey is closed by a **bott**, which is a tapered water-cooled plug on the end of a steel rod. The bott, on large furnaces, is operated mechanically by a device known as the **cinder-notch stopper**. On the larger furnaces, such as these, two cinder notches usually are installed.

In time, the bottom is eroded to various degrees and is replaced by a large mass of metal known as a **salamander**.

A series of 6-inch thick, cast-iron **staves** surround the hearth sidewalls, extending about 4 feet above and 12 feet below the center line of the iron notch. The tops of the staves are at about the level of the working floor around the furnace; in most modern furnaces, they

are embedded in the brickwork or concrete below floor level. Water-cooling pipes are cast into the staves to reduce the temperature of the hearth sidewalls. A 1½-inch thick steel **hearth-jacket plate** surrounds the staves for approximately their full height to resist the bursting stresses from the inside of the furnace. The staves are modified for the iron notch and cinder notch openings. At the iron notch, it is the practice to place a removable stove, about one-half the height of the normal stove, as a means for a quick repair should a breakout occur at this location.

Above the tops of the cooling staves, the brickwork of the hearth wall is enclosed on the outside by a **tuyere jacket or breast plate**, made of welded 1½-inch thick steel plate. Reinforced openings are provided in this jacket for insertion of the **tuyere coolers**, **cinder notch** and **cooling plates**, all of which fit their respective openings closely. Some of these details may be observed in Figure 12-5, which shows a section of the bosh and the upper part of the hearth. The cooling plates are flat, water-cooled copper castings which extend into the brickwork of the hearth sidewalls. Usually, there are one or two rows below the tuyere-cooler level and three rows between the tuyere-cooler openings. Another method of construction is to use steel castings for the tuyere jacket instead of the welded-steel plate construction. Recently, on furnaces with carbon-hearth linings, water-cooled iron castings in which pipes have been cast (similar to a hearth-cooling stove) have been used. In this latter case, a heavy, welded, steel-plate jacket surrounds the castings to resist the stresses exerted by the outward pressure of the furnace contents against the walls of the hearth.

The heated blast enters the furnace through nozzle-like water-cooled copper castings known as **tuyeres**, usually about twenty in number in a large furnace, spaced around the hearth about a foot below its top (Figure 12-5). The tuyere cooler supports the tuyere in the same manner as the cinder-notch cooler supports the intermediate cooler. For different furnaces in the same plant, the length and diameter of the tuyeres may be different but in all cases they are designed to fit inside the same tuyere-cooler casting. A horizontal cast-iron pipe called the **blowpipe**, about 5 feet long and with one end fitting closely against the tuyere, carries hot blast from the tuyere stock to the tuyere, thence into the furnace. The blowpipe may be slightly larger in diameter at one end than at the other and both ends are turned to fit into slight sockets in the tuyere and the stock. It is held in place with the smaller end fitting into the tuyere by pressure from the tuyere stock which, in turn, is held tightly against the other end of the blowpipe by a heavy spring and rod, called the **bridle**, which is attached to the jacket plate and allows limited motion due to expansion and contraction of the stock and blowpipe with changes in blast temperature. The tuyere stock curves upward on leaving the blowpipe to fasten to the **gooseneck** and is lined with firebrick as a protection against the heat of the blast. In the outer part of the curve in the tuyere stock and on the center line of the blowpipe and tuyere is a small opening closed by the **tuyere cap** or **wicket** through which a small rod may be inserted to clean out the tuyere without removing the blowpipe. The tuyere cap must be of such form that it may be opened readily at any time and still be practically gas-tight. This is accomplished by a tapered plate inserted in guides, in which a small glass-covered opening called the **peep hole** or **eyesight** is placed to permit the inspection of that portion of the interior of the furnace directly in front of the tuyeres. The upper part of the stock is con-

nected by a swivel joint to the refractory-lined nozzle of the gooseneck to which it is clamped by lugs and keys that fit into seats of hanging bars. Each gooseneck in turn is connected by flanges and bolts to a neck extending radially from the inside diameter of the **bustle pipe**. The bustle pipe is a large, circular, refractory-lined and insulated pipe which encircles the furnace at about mantle level and equally distributes the heated blast from the hot blast main to each tuyere connection.

Bosh—Although previously mentioned, the bosh may be described more specifically as that portion of a furnace formed by an inverted frustrum of a cone starting at the top of the vertical sidewalls of the hearth and extending upward to the level of maximum furnace diameter at the mantle. A standard bosh is constructed in the following manner: Starting at the top of the hearth, the brickwork, 27 inches in thickness, is stepped outward, externally, nearly six inches for each twelve inches of vertical rise. Each step-out is supported on the outside by a heavy steel band called a **bosh band**. Extending through the walls of the bosh, fitting closely into cast-iron "boxes" placed in the brick spaces between bosh bands, are cooling plates called the **bosh plates**, in horizontal rows from 18 inches to two feet apart, measured vertically. The plates in each row are several inches apart, and the plates in the different rows are staggered vertically, "breaking joints" like brickwork. This construction adds to the cooling efficiency of the plates. At some plants, it is the custom to place the plates directly in the brickwork, eliminating the cast-iron box entirely. As a safety precaution, a means is usually supplied to prevent the plates from being forced out of the furnace. There are several different makes of bosh plates, but the more common ones are somewhat wedge-shaped, with a flat bottom and oval top, and about four inches thick at the point of their greatest thickness. They are hollow and have inserted in them, usually at opposite corners, two pipes through which water flows continuously; the water enters through one pipe and leaves by way of the other. These plates are necessary to cool and thus protect the brickwork for, being just above the tuyeres, the bricks here are exposed to the highest temperatures in the furnace. In American practice, the bosh has been built of first-quality fire-clay brick, but consideration is now being given to extending the carbon lining of the hearth up to the top of the bosh, which practice is used extensively in Europe.

Regardless of the size of the hearth, the modern furnace has a bosh 9 to 12 feet in height and a **bosh angle** between 80 and 85 degrees (Figure 12-5). In the older furnaces, when a large part of the ore consisted of lumps, the usual bosh angle was about 10 degrees less than at present; but it was found with the use of the finer ores, such as Mesabi, that better results were obtained with a steeper bosh. Also, the height of the older boshes ranged between 15 and 20 feet.

Inwall—The shaft comprises all that part of the furnace which is located above the bosh. The wall of this shaft is usually, in an imaginary way, divided into three almost equal parts, called the **upper**, **middle**, and **lower inwalls**. Up to this point, the construction for all furnaces is fairly uniform in general features, but as to the inwalls, three types are employed, namely, the thick, the intermediate, and the thin wall. The construction necessarily differs among these three types, and each is described separately.

Thick-Wall Type—The inwalls of this type, of which there are many in the United States, have a maximum thickness of five feet, and are enclosed in a heavy riveted or welded steel shell about one inch thick. A space is left between the brickwork and the shell, in

which space is tamped a packing of compressible material to allow for the expansion and contraction of the inwalls. Thick-walled furnaces usually are not water-cooled extensively above the bosh. In case their walls are cooled, this is done by inserted cooling plates.

Intermediate, or Semithin, Wall Type—In the intermediate type, the walls are about three feet thick and, to protect the brick as much as possible, cooling plates (similar to bosh plates) are inserted in the lower inwall. They may extend for a distance of from twenty to forty feet above the bosh. The inwalls are surrounded by a steel shell as in the thicker type, the only difference being in the necessary openings for the coolers. Many American furnaces have linings of this type.

Thin-Wall Type—In this type, the inwalls are from 9 to 18 inches thick, the top is always supported by structural work, and the shell must be cooled throughout its entire length. This cooling may be done in three ways. One method consists in spraying the steel shell with water, conducted through suitably arranged pipes and enclosed by a light splash jacket which conforms to the size and shape of the stack. In the second method, the shell is encircled by a series of deep and narrow horizontal troughs through which water is kept flowing from the top to the bottom of the furnace, overflowing from each trough to the next succeeding lower one. Each of three or four of these troughs drains separately to a common point where the temperature of the water can be noted. In the third type, the entire outer surface of the stack is kept covered with water by a spiral trough which, slightly separated from it, encircles the stack from top to bottom. This trough is kept full of water by a series of feed lines that enter it at various points in the spiral. Few furnaces of this type have been built in the United States, and these gradually are being dismantled.

Relationship of Furnace Dimensions—The relationship of the furnace dimensions, one with another, is most important. Practically all of these relationships are the result of "cut and try" methods, for they do not lend themselves to theoretical calculation. As an example of good furnace design, South Works Furnaces Nos. 11 and 12 have been selected, whose principal dimensions are as follows (see also Figure 12-6):

Diameter of hearth—28 ft.
Diameter of bosh—31 ft.
Diameter of stockline—21 ft. 6 in.
Diameter of big bell—16 ft. 6 in.
Bosh angle— $81^{\circ} 40' 27''$
Inwall batter—1.00 in. per 12 in.
Hearth area—616 sq. ft.
Working volume—47,550 cu. ft.
Number tuyeres—20
Centerline iron notch to top hearth—9 ft. 3 in.
Centerline iron notch to centerline cinder notch—4 ft. 8 in.
Centerline cinder notch to centerline tuyeres—3 ft. 8 in.
Centerline tuyeres to bottom bosh—11 in.
Height of bosh—10 ft. 3 in.
Height straight section lower inwall—9 ft.
Height sloping section inwall—57 ft.
Top sloping section to top of hopper—22 ft. 9 in.

In many quarters, the rated daily production of a furnace is the hearth area in square feet multiplied by a factor from 2.0 to 2.6 which varies primarily with the materials used. The rated daily production of the above furnaces is $616 \times 2.45 = 1,500$ net tons per day. The theory has been advanced that the daily production is a function of the coke burned per day, which in turn is determined by the area in square feet of a ring six feet wide in front of the tuyeres, multiplied by 6,300.

There are two methods for calculating the proper number of tuyeres for a given hearth diameter: the first being to divide the hearth area in square feet by 30; and the second being to divide the inside circumference of the hearth in feet by 4.0 to 4.5.

The working volume of a furnace may be defined as the volume of that part of the furnace through which the gases produced at the tuyeres pass in contact with the materials in the charge. In other words, this volume may be considered as the volume between the centerline of the tuyeres and the stockline. The stockline in this case is assumed to be a horizontal plane located six feet below the big bell in its closed position. The ratio of the working volume to the hearth area should range between 75 and 80, and in this particular case is $47,550 \div 616 = 75.2$. For furnaces producing ferromanganese, this

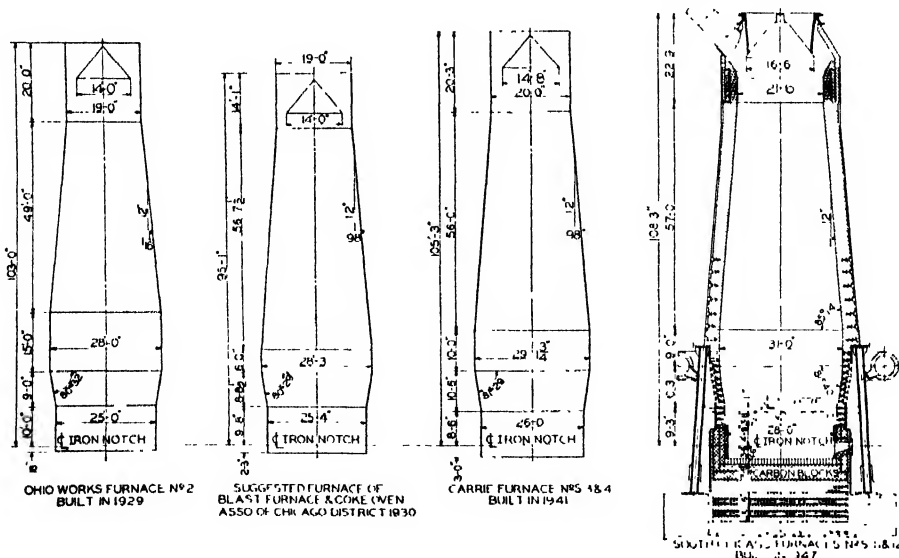


Fig. 12-6. Lines of typical modern American blast furnaces designed and built during the period 1929-1947.

ratio is even higher than the 80 figure indicated above, being about 90.

An inwall batter of 0.9 to 1.1 inch per foot is desirable which, with the ratio of working volume to hearth area, defines fairly well the stockline diameter. It has been stated that the stockline area should equal the area of the 6-foot horizontal ring in front of the tuyeres mentioned earlier.

The big-bell diameter should be from 4 to 5 feet less than the stockline diameter for smaller furnaces and about 5 to 5.5 feet less for larger furnaces.

The time that elapses from the time the materials are charged at the top until they reach the tuyeres is important because if just sufficient time is available, they will be thoroughly preheated and reduced to the optimum extent by the ascending gases. A derived figure, the theoretical speed at which they descend in the furnace, is also important. The materials next to the wall are in the furnace a shorter time than those in the center since their rate of descent is faster. An average or theoretical figure gives a good check on proper relationship of furnace lines. An example of these theoretical figures is shown below:

	Per Net Ton Iron		Lbs. Per
	Lbs.	Cu. Ft.	Cu. Ft.
Group 3 Ore.....	2748	22.4	122.7
Group 7 Ore.....	923	7.4	125.0
Total Ore.....	3671	29.8	123.3
Open-Hearth Slag.....	395	2.6	150.0
Flux and Miscellaneous...	1055	10.3	102.0
Net Coke.....	1782	59.4	30.0
Gross Total.....	6903	102.1	67.5
Less Flue Dust Produced.	284	2.6	110.0
Net Total.....	6619	99.5	66.5
Iron production, net tons per day.....			1,324
Cubic feet materials processed per day.....	$1,324 \times 99.5 =$		131,738
Working volume, cubic feet.....			44,849
Furnace fillings, number per day.....	$131,738 \div 44,849 =$		2.94
Time material in furnace, hours.....	$24 \div 2.94 =$		8.16
Working height, feet.....			79.52
Speed of material descent, feet per hour.....	$79.52 \div 8.16 =$		9.75

In the above calculations, the weights of ore per cubic foot used are less than the weights as they lie in the pile, due to the ore being "fluffed up" in handling.

Furnace Linings—The brickwork which forms the hearth, bosh, and inwalls of a furnace is referred to as its lining. All the brick used in the construction of these parts are made of fire clay. The brick in the hearth and bosh are required to resist a very high temperature and the action of flux and slag; brick in the inwall zone must be able to withstand abrasion at a moderately high temperature; and top brick, though always at a comparatively low temperature, must resist the impact and abrasive forces of the charges as they are dropped into the furnace.

The bricks are inspected carefully before being put in place in the furnace. Great care is exercised with respect to brick, because the life of the furnace depends in a large measure upon the lining, and the item of cost for brick is not a small one. In the construction of one of the large thick-wall furnaces, the equivalent of ap-

proximately 1,000,000 nine-inch bricks is required. Fire-bricks are always laid in a thin slurry composed of refractory mortar. The slurry is applied by pouring it on the top of each course with a dipper, and followed immediately by the next course of bricks, which is hammered into place to squeeze out all the fire clay except that required to compensate the inequalities of the brick.

Water Troughs—Encircling the furnace bosh, at about the level of the bustle pipe, will be found one or more water troughs into which the water supplying the numerous cooling plates is discharged in visible streams, thus providing means of determining when a plate, tuyere, or cooler is burned out. The water, flowing from these troughs into a well, may be reused by pumping to the condensers or gas washers. The better practice is to reuse the water from the condensers for furnace cooling.

Top—At the present time furnace tops are somewhat complicated affairs. In olden times the tops of furnaces were left open, the escaping gases being allowed to burn in the air. In the year 1809, these waste gases began to be employed in France for the purpose of burning bricks and heating small furnaces. The first attempt to heat the blast by utilizing these gases was made in 1829. It was not until 1845 that a plan was evolved by which the gases could be used to heat "stoves", which are separate structures designed for the sole purpose of preheating the air blown into the furnace through the tuyeres; these are described later. To effect this purpose, changes in top construction were necessary. At first the gases were merely drawn off by the chimney draft of the stoves through openings below the stock line. The arrangement, known as the bell and hopper, or cup and cone, was not put into use until 1850. It consisted in finishing the top of the furnace with a large circular hopper, the smaller opening of which was closed by the bell that could be lowered and raised at will. With the bell raised against the hopper, the materials were dumped into the hopper; then when the bell was lowered, the charge dropped into the furnace. As large quantities of gas escaped with each lowering of the bell, this device was improved by the double bell and hopper, which is of comparatively recent origin. Essentially, this improvement consisted in placing a second but smaller bell and hopper above the first, and providing a gas-tight space of large size between the two. The raw material, upon being hoisted to the top, is first dropped or dumped into the upper hopper, whence it may fall into the larger hopper below when the small bell is lowered. The small bell being raised against the upper hopper, the large bell is lowered, and the charge falls into the furnace without the escape of gas. The bells are made of cast steel, in one piece, and of such a slope, 45° to 55°, as to permit the charge to slide off readily. They usually are supported from their top centers by a rod and a sleeve, each attached to a counterbalance lever operated by a steam or air cylinder or an electric motor, controlled from the ground. The large bell is attached to the rod and the small bell to the sleeve. The hoppers, of cast steel or iron, generally are made up in sections, which are securely bolted together. The details of this construction differ somewhat from furnace to furnace to conform to the introduction of improvements, with the type of hoist, and with the preferences of the different builders. Operation of a typical design is shown in Figure 12-7.

Hoisting Appliances—The old-time method of charging by hand having been superseded by mechanical charging, there are now in use two types of these devices, namely, the skip hoist and the bucket hoist. In

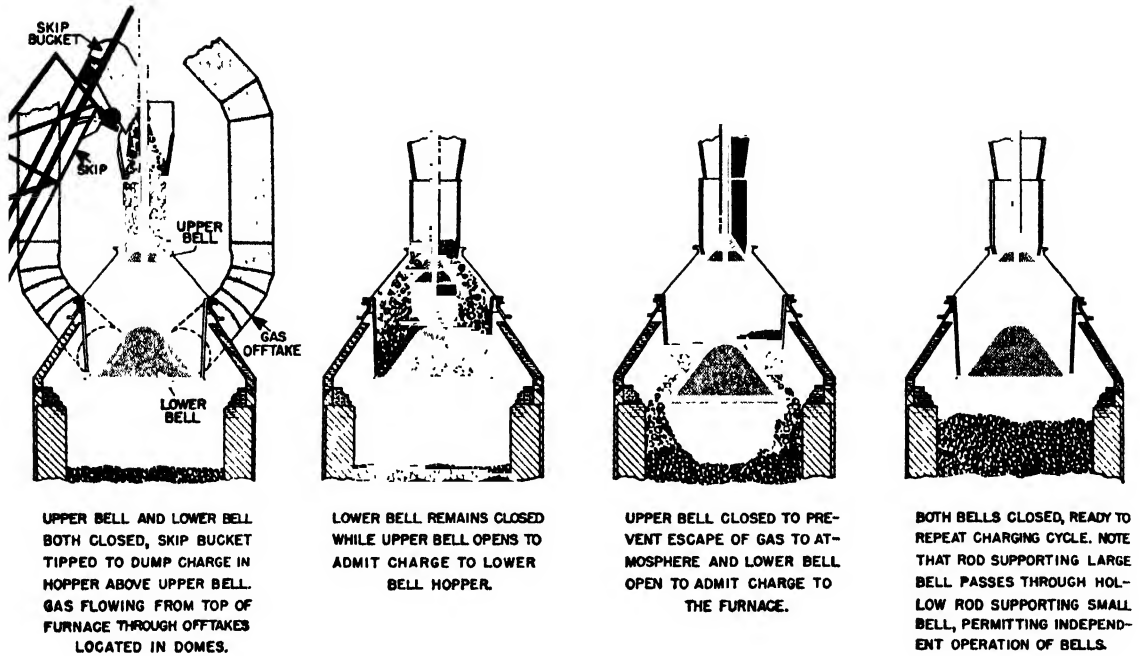


FIG. 12-7. Diagram of progressive steps by which the double bell and hopper permits charging raw materials into a blast furnace without escape of furnace gases.

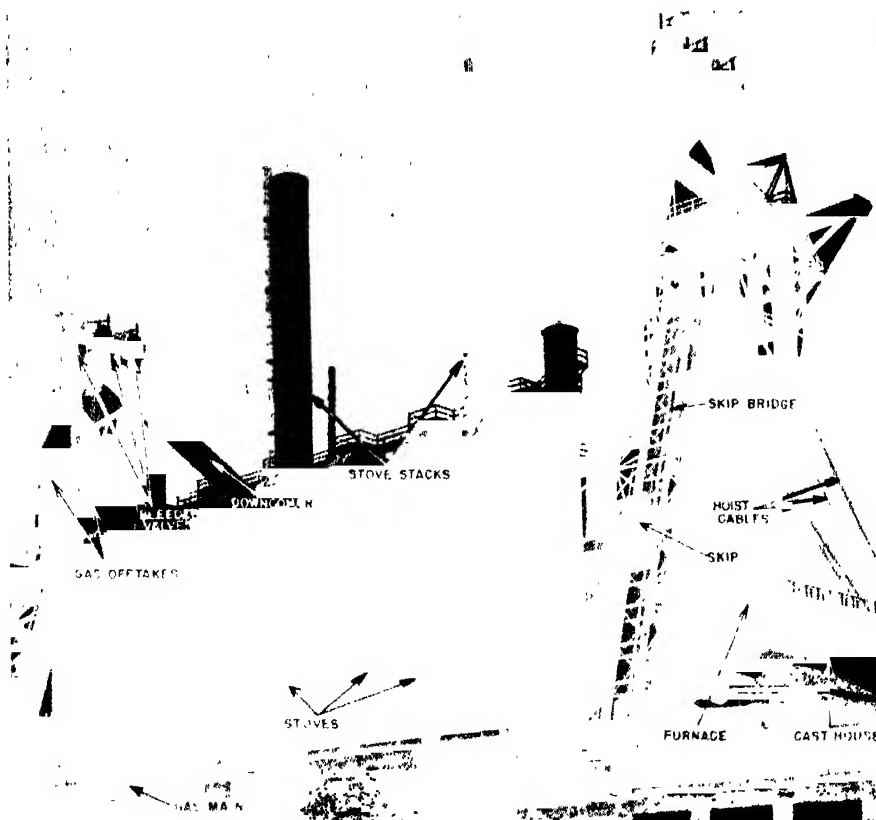
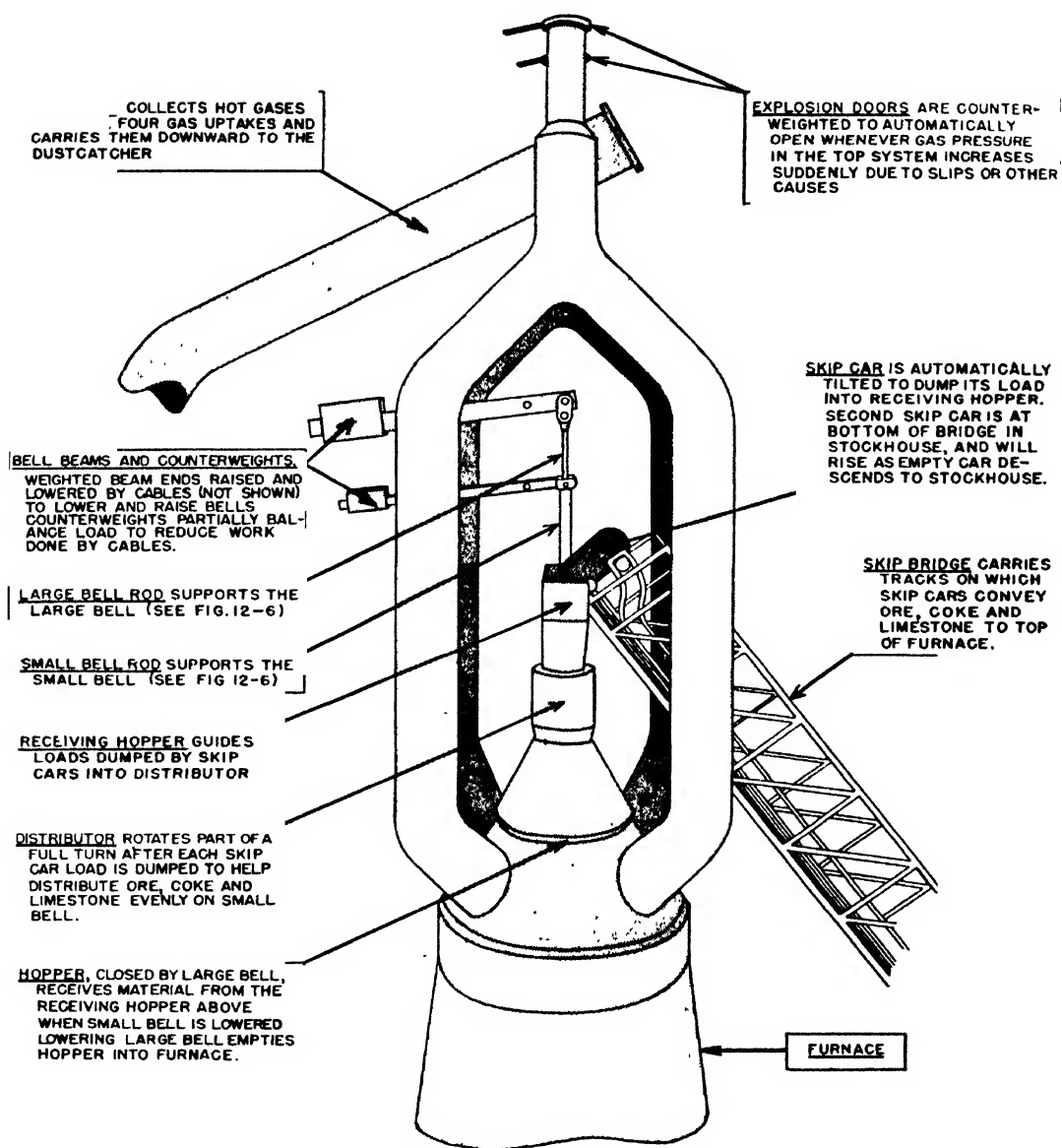


FIG. 12-8. Overall view of a blast-furnace plant, showing two furnaces and the stoves that operate in conjunction with them.

PRINCIPAL OPERATING PARTS OF A BLAST FURNACE TOP*



* THE VARIOUS PLATFORMS AND STRUCTURES SUPPORTING THE TOP EQUIPMENT HAVE BEEN OMITTED FROM THIS DRAWING, AS WELL AS THE CABLES AND SHEAVES THAT OPERATE THE SKIP AND BELLS. ALSO OMITTED ARE THE JIB BOOM CRANES THAT ASSIST IN HOISTING REPAIR AND REPLACEMENT PARTS TO THE TOP OF THE FURNACE.

FIG. 12-9. Principal operating parts of a blast-furnace top.

both cases there is an incline (a fabricated steel structure) extending from the top of the furnace to or below the bottom of the stock house; and over the tracks of this incline the materials charged into the furnace must pass. In the **skip hoist**, the conveying vessel is a small open-ended steel car, called a **skip**, that automatically dumps the materials into a hopper above the small bell. Skip hoists are generally provided with double tracks, so that while a loaded skip is passing up the incline an empty one is descending. In the **bucket hoist** the solid materials are raised in a bucket, suspended from a truck or carriage, that drops the charge directly into the space above the large bell. When in position for dropping the charge, the bucket, being itself provided with a small bell at the bottom, takes the place of the little bell and hopper. While the bucket is being filled at the stock house, the opening left in the top is closed with a special gas seal. Figure 12—8 shows furnaces equipped with skip hoists.

Top Appliances for Distributing the Stock—It is apparent that, in a mechanically filled furnace, when the raw materials are dropped into the receiving bell, the larger lumps of ore and stone will have a tendency to roll and thus collect around the edges, or perhaps at one side or the other. The same segregation, if uncorrected, will also occur upon dropping the charge into the furnace. This tendency results in more or less open and continuous **channels** being formed through the materials and extending from the top towards the bottom of the stack. These channels offer less resistance than the remainder of the materials to the passage of gases through the furnace, with the result that a disproportionately large quantity of gas passes through them. This condition, called **channeling**, also results in excessive temperatures along these passages. To overcome this defect, various plans and devices have been employed to introduce the materials into the furnace in such a way that this segregation of coarse and fine material will not occur.

Rotation of the position of the charge on the large bell by various means has been most effective in minimizing channeling. This can be accomplished in several ways, depending upon whether the furnace is bucket or skip charged. One of the most popular and effective devices is a rotating hopper above the small bell which can be rotated through a prescribed arc before the small bell is lowered.

Some designers concentrate upon details of construction of the furnace top and of the large bell and hopper with good results. For example, it can be shown that as the diameter of the large bell approaches the internal diameter of the furnace top, the fines are concentrated next to the wall, and as the bell diameter is decreased, a dimension is reached that throws some lumps against the wall and some toward the center, while a further decrease in diameter throws more and more lumps toward the wall until all the fines are concentrated at the center. Other factors, such as the angle of the bell, the angle of the hopper, the extension of the bell below the hopper, the distance below the hopper the bell can be dropped, and the speed with which it can be lowered, all may be adjusted to produce the best charge distribution.

Top Openings—The smallest opening in the top of a furnace is the **try hole**. In operating a furnace it is necessary to be able to determine the position of the stock line. This is done by means of the **stock indicator**, which is a rod of steel passing through and fitting the try hole loosely so that one end rests upon the stock, while the other is attached to a small steel cable that leads to the stock house or the cast house below. Some stock indicators are automatic and self-recording. Some modern furnaces use two stock indicators, mounted 180° apart.

Gases resulting from the smelting operation are conducted to the top of the furnace by large, vertical pipes called **offtakes**, each of which is about 6 feet in diameter. The offtakes, usually four in number, are connected to openings in the furnace top at about the level of the big hopper. They may be constructed either of abrasion-resistant steel or may be lined with firebrick. The tops of the offtakes are connected in pairs, and pipes lead from these connections into one large descending pipe called the **downcomer** or **downtake**. Rising above the junctions of each pair of offtakes are tall, vertical pipes called **uptakes** or **bleeders**. These usually are lined with one course of firebrick. The tops of the bleeders are equipped with counterweighted valves. These valves have a double purpose: first, they may be opened by mechanical means from ground level to permit surplus gas to escape from the furnace system; secondly, they may be adjusted, either by weights or mechanical means, to act as **explosion doors** and open at a predetermined pressure to relieve sudden increases in gas pressure occasioned by "slips" in the furnace, and thus prevent possible injury to the top. A third bleeder, with explosion door, may rise above the junction of the pipes from the offtakes with the downcomer (Figure 12—9). Some furnaces have a separate bleeder for each offtake. The explosion doors are at the highest point of the furnace (Figures 12—8 and 12—9).

General Considerations for Top Construction—As previously pointed out, there are many types of tops, and the description above is intended to give a general idea of the essential parts and their uses. The chief endeavor in top construction is to perfect the distribution of the stock entering the furnace stack, and either eliminate or compensate for as many irregularities as possible, while a second aim is to hold to a minimum the amount of solids (coarse and fine dust from the raw materials) carried out with the gas. However, in attaining these ends, simplicity must be considered, as any great amount of mechanism on the top of a furnace is objectionable. It is important to prevent large material from being thrown out of the furnace in case of slips, and as little dust as possible at any time should be carried out by the gases. In locating the offtakes, care is taken to see that they do not enter the furnace directly over the tapping hole, cinder notch, or the entrance of the blast main to the bustle pipe, because, these being the most active points in the furnace, entering the furnace over these points would tend to cause more uneven distribution of the gases through the stock. In the latest furnaces, the top diameters have been increased in an endeavor to reduce the relative amounts of flue dust produced.

SECTION 4

CONSTRUCTION OF FURNACE AUXILIARIES

Stoves—Blast-furnace stoves, usually three per furnace, are brick-lined regenerators enclosed in a circular steel shell with a flat bottom and a dome-shaped top.

Their function is to preheat the blast before its admission into the furnace through the tuyeres. The result of preheating the blast is to intensify and speed

up the burning of the coke at the tuyeres with a consequent reduction in the coke required for the smelting operation; this reduction in coke consumption is more than would correspond merely to the additional sensible heat carried in by the heated blast. Essentially, a stove consists of two parts; the first being the combustion chamber which is a vertical passageway (in which cleaned blast-furnace gas is burned) extending from a point near the bottom of the stove to the bottom of the dome, and through which the hot products of combustion pass upward to the dome. The second main part is the checkerwork, which contains a multiplicity of small passageways through which the products of combustion from the blast-furnace gas pass downward from the dome to a point near the bottom of the stove. This description applies to the two-pass design which is the most prevalent today but there still exist quite a number of stoves of the three-pass design, the difference being that an additional upward pass of checkerwork is added through which the products of combustion pass before their exit at the top of the stove. In a few rare cases, a four-pass design has been used which again simply adds a downward pass for the products of combustion over the three-pass design. In all designs, there is only a single combustion chamber. The two-pass design offers a larger percentage of the cross-sectional area for checkerwork than either the three- or four-pass designs because of the space taken up by the walls which separate the passes. With sufficient volume of checkerwork installed, the two-pass design permits the spent gases to leave the stove at a temperature no higher than that from stoves with the more complicated and more costly three- and four-pass designs. The temperature of the exit gases is, of course, a measure of the efficiency of the stove. The design of stoves may be broken down further into either the side-combustion or the center-combustion types, which refers to the location of the combustion chamber inside the shell itself. The preference seems to be for the side-

combustion type. Figure 12—10 shows vertical and horizontal sections for the side-combustion type.

Modern stoves for large furnaces are 26 to 28 feet in diameter and about 120 feet high from the bottom of a stove to the top of its dome. Depending upon the type of checkers used, the stoves contain between 250,000 and 275,000 square feet of heating surface. If the checkerwork walls are too thin, structural failures will result. It has been found that a ratio of weight of brick (in pounds) to square foot of heating surface of about 10 is the minimum. Below this ratio structural difficulties are encountered. Formerly the checkerwork was supported several feet above the bottom of the stove by brick piers but present construction uses cast-iron or steel columns to support steel grids which in turn carry the checkerwork. Insulation between the brick and the steel shell prevents the heat in the stove from distorting the steel plates. The blast-furnace gas used to heat the stoves is cleaned almost universally by wet washing, primarily in a tower-type washer and finally in a precipitator or disintegrator. Primary gas will contain about 0.2 grain per cubic foot of solids and should not be used in stoves with openings less than $4\frac{1}{2}$ in. square. Final or secondary gas will contain less than 0.02 grains per cubic foot and can be used successfully in checkerwork with openings slightly less than 2 inches by 2 inches. An empirical factor of about 500 square feet of total heating surface per net ton of daily furnace production has been found to give adequate stove capacity. This excludes the area of any cross flues in the checkerwork.

The stoves for a furnace, when of the two-pass design, usually are provided with a common stack about 200 ft. high, while three-pass stoves have an individual stack for each stove mounted on top of the dome of the stove shell.

Air blast for the furnace passes through the heated stove in a manner counter-current to the passage of the gas. In a two-pass stove the gas passes upward through

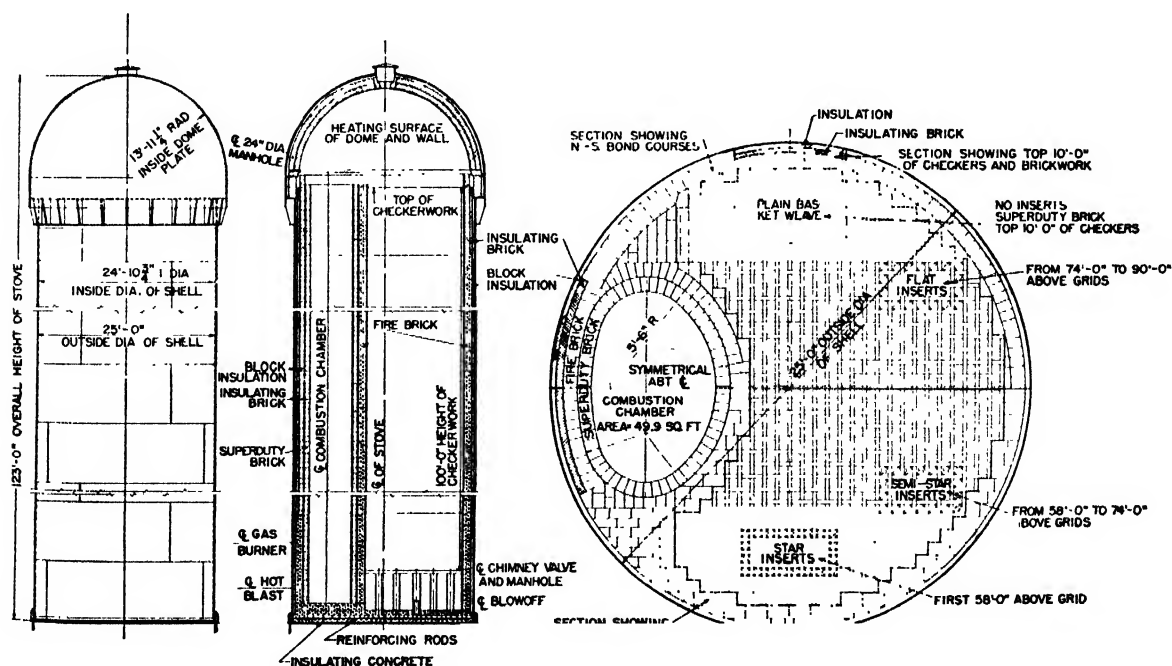


FIG. 12—10. Details of a modern blast-furnace stove, showing: (left) dimensions; (center) principal details of stove-lining construction; and (right) details of checkerwork at various levels. Checkers of this type are designated as of basket weave with inserts, as shown in detail at left of Figure 12—11.

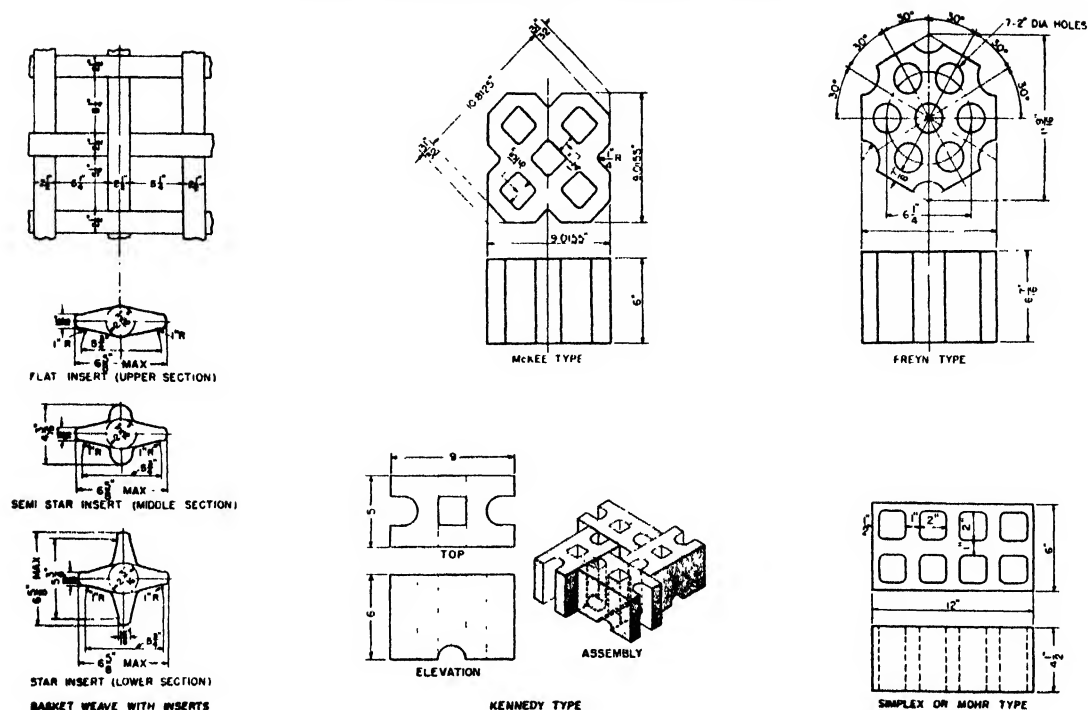


FIG. 12-11. Five different designs of checker-building shapes employed in the construction of modern blast-furnace stoves. The basket-weave design with inserts is shown in Figure 12-10 also.

the combustion chamber, downward through the checkers, and out to the stack while the blast passes upward through the checkers, downward through the combustion chambers, and out into the hot-blast main. With three stoves to a furnace, one stove is "on blast" at a time while the other two stoves are "on gas" with the result that each stove is "on gas" twice as long as it is "on blast." In order to accomplish the changeover from gas to blast, and vice versa, certain valves are necessary.

The hot-blast valve is usually of the water-cooled mushroom type, closing on a water-cooled seat. Its function is to open the passageway for the heated blast from the combustion chamber to the hot-blast main. The cold-blast valve admits cold air from the blast line into the space directly below the checkerwork and, not being subjected to high temperatures is usually of the gate type. The chimney valves, usually two in number, open and close the passageway for the products of combustion from the stove to the stack and are subjected to only moderately high temperatures. The blow-off valve releases the pressure in the stove when changing the stove from "on blast" to "on gas." Since the heated blast leaves the stove at a higher temperature after the stove has just been put "on blast" than after it has been cooled somewhat by the air undergoing heating that passes through it, and since it is desired to maintain, in most cases, the hot blast at a constant temperature, the mixer valve by-passes a certain amount of the cold blast into the heated blast and by its regulation maintains a constant hot-blast temperature. A pyrometer is installed in the hot-blast line beyond the point of entry of the mixer line; it can be set to the temperature desired and will operate a mixer valve automatically to maintain this temperature. The burner valve or door opens and closes the passageway from the stove gas burner into the combustion chamber. Most gas burners

today are of the pressure type, in which the air for the combustion of the gas is blown by a fan into the burner in such quantities as to give theoretically perfect combustion. In the older type burners, the combustion air was aspirated into the combustion chamber with the gas with resultant mediocre combustion. The nose of the burner is movable so that it may be moved tightly against a flange on the burner door to prevent any infiltration of air or puffing out of the gas when the stove is "on gas." Inspection or cleanout doors are provided at the top and at the bottom of the stove. The snort valve, which derives its name from the noise made by blast escaping into the atmosphere, is located in the cold-blast line leading from the blowing room to the stove. Its function is to relieve the pressure of the blast entering the furnace when a "slip" occurs in the furnace, during casting, or when any emergency occurs. It consists of two interconnected butterfly valves, one closing the line against the blast, and the other opening a by-pass line to the atmosphere.

The brick lining of a stove, which includes all the brickwork enclosed by the shell, usually lasts from 15 to 20 years before it becomes necessary to replace it because of spalling, deformation, or clogging of the checkers. In addition, it may be necessary to replace the upper 10 to 20 feet of the checkerwork at the end of each furnace campaign, or from five to seven years. An insulating space of about 3 inches between the brickwork and the steel shell is filled with high-grade loose insulation or insulating strips. The bottom is insulated with insulating concrete, and the dome is covered by small insulating squares on top of which loose insulation is placed. The dome of a modern stove is separately supported from the outside of the shell, which allows the brickwork below to "breathe" and at the same time permits replacement of the lower brickwork without disturbing the dome (see Figure 12-10).

The brick is first-grade firebrick and should possess high thermal capacity and conductivity. The skin wall of the combustion chamber and the top 10 feet of checkerwork are constructed of a higher alumina brick than first-grade firebrick. Some designers prefer semisilica brick, made from New Jersey clay, over the higher alumina brick. Formerly, checkers were built of the conventional rectangular formed brick and provided relatively large channels for passage of gases. With the advent of the use of secondary-cleaned gas in the stoves, the use of smaller openings has become feasible. Inserts may be used in the passages of checkers built of rectangular brick, as shown in Figure 12-10, or special brick made in clusters with each cluster containing several openings may be used; the latter facilitate laying and handling. Figure 12-11 shows several types of the more common clusters now in use. The thermal efficiency of a stove is about 85 per cent. The stoves use between 18 and 24 per cent of the gas produced by the furnace for heating the blast.

Dustcatcher and Gas Mains—The object of the dustcatcher is, as implied by its name, to remove as much as possible of the flue dust blown over from the furnace, with which the gas is heavily laden. If this dust is not removed, at least in part, it puts an unnecessary load upon the primary and secondary washers and, in addition, the dust is more easily and economically handled in the dry state as it comes from the dustcatcher. The dustcatcher is from 35 to 40 feet in diameter, usually lined with brick to prevent cooling and the attendant precipitation of water from the gas. It is a cylindrical structure fitted with a 60-degree inverted cone on the bottom and a cone-shaped top.

The single downcomer from the top of the furnace carries the gas to the top of the dustcatcher, where it enters a vertical pipe that is centrally located inside the dustcatcher and extends nearly to the bottom (Figure 12-2). This vertical pipe flares outward about 8 degrees and is thus larger in diameter at the bottom than at the top. As the gas passes downward through this pipe, its velocity (and therefore its ability to carry dust) is lowered, causing some of the dust to drop out of the gas stream and be deposited at the bottom of the dustcatcher. When the gas emerges from this pipe, it must change direction of flow suddenly and rise to the top of the dustcatcher, since the gas is removed from a convenient location at the top. The sudden change in flow direction causes more dust to be dropped from the gas stream. Efficiency of the dustcatcher is approximately 60 to 75 per cent, depending upon the type of ores used and the blast volume. The accumulating dust usually is removed from the dustcatcher by a double-shaft pug mill which is rigidly fastened to the cone bottom. Water is added to the pug mill until the dust is slightly moistened, which prevents the dust from blowing around the vicinity as it drops into a railroad car. An auxiliary hand-operated gate also is provided which permits dust to be removed in case the pug mill is down for repairs. The exit gas passes from the dustcatcher into the bottom of the primary washer.

Gas-Cleaning Plants—Neglecting the larger lumps blown out occasionally when the furnace "slips," the dust particles in blast-furnace gas vary in size from $\frac{1}{4}$ inch to a few microns (1 micron = 0.00003937 inch) with practically all of the dust passing the dustcatcher being minus No. 20-mesh sieve size. The difficulty of cleaning increases as the particle size decreases, the removal of the very fine particles being comparable to the removal of smoke from air. On this account, plants generally are planned to clean the gas in two stages, a **primary stage** to remove the coarser particles, which

form the larger proportion of the dust, and a **secondary** or **final stage** to remove as much of the remaining dust as possible. The approximate removal of incoming dust by the usual gas-cleaning system is 60 to 75 per cent in the dustcatcher, 90 to 95 per cent of the remainder in the primary washer, and 90 to 95 per cent of the dust still in the gas leaving the primary washer is eliminated in the secondary washer. The gas entering the dustcatcher contains from 7 to 30 grains per cubic foot, an average dust content being about 16 grains per cubic foot. In some cases, a centrifugal cleaner is installed between the dustcatcher and the primary washer.

Either of two methods is used to clean the gas passing the dustcatcher, wet cleaning or dry cleaning. In **wet cleaning**, the aim is to wet the dust particles and wash them out of the gas with water. Incidentally, the gas is cooled to about the temperature of the wash water, and any moisture in excess of saturation at this temperature is precipitated. In **dry cleaning**, one of the aims is to remove the dust without cooling, thus conserving the sensible heat. In many cases, the temperature of the gas leaving the furnace is under 300° F (150° C) as contrasted with 500° F to 600° F (260 to 315° C) in former practice. The elimination of the water, which exists in the gas as superheated steam, and which is reduced to its dew point concentration at the cleaning-water temperature, more than balances the loss in sensible heat in wet cleaning. For these reasons, dry cleaning presently is not used to any great extent.

Primary wet cleaners include stationary-spray towers, revolving-spray towers, Feld washers as described later, baffle towers, spray fans and water separators. The stationary-spray tower is most used, although many Feld washers remain in use, and, sometimes a washer consists of a combination of one or two Feld washer revolving cones in the bottom, above which are several banks of stationary sprays. In almost all cases, a water separator is mounted on top of the washer as an integral part thereof.

The **stationary-spray tower** consists of a supported steel cylinder with conical bottom and conical top with the gas entering at a point near the bottom of the cylinder and leaving through a centrally located outlet on top of the top cone. Inside the cylinder are three or four banks of ceramic tile (with about 5-inch diameter round holes in them) which split up the rising gas and tend to prevent channeling. Above each bank are water sprays which uniformly cover the cross section of the washer with a falling "rain." A U-shaped seal pipe at the outlet of the bottom cone carries the dirty water away to a Dorr thickener or settling basin and prevents the gas from blowing to the atmosphere. A centrifugal-type water separator above the topmost bank eliminates any entrained water in the outgoing gas.

The **Feld washer** is a cylindrical shell with a flat bottom and a conical top in which there is a series of from 5 to 7 inverted frustrums of cones which rotate on the same centrally-located vertical shaft, driven by a motor and reduction gear on top of the washer. The lower edge of each cone is immersed in a separate water tray. The centrifugal action of the rotating cones picks up water from each tray and throws it outward toward the shell as a spray through which the rising gas must pass. Baffles guide the water back into the trays for reuse, excess water spilling into the next tray below. A water separator similar to that on the stationary-spray tower is located above the topmost rotating cone.

Primary dry cleaners include various modifications of cyclone separators, centrifugal machines, metallic-wool-pad filters, and electrical or Cottrell precipitators. The dry-type Cottrell precipitator may also be classed

as a secondary dry cleaner and will be discussed under that heading. The cyclone and centrifugal types present the problems of excessive wear and difficulty in handling the fine dust removed. The filters have a tendency to clog shortly after being placed in operation.

Secondary wet cleaners have consisted of wet-type Cottrell precipitators, high-speed disintegrators, and Theissen disintegrators. Formerly, Theissen disintegrators were used almost exclusively, but on later installations have been replaced by high-speed disintegrators or by the wet-type Cottrell precipitators, with preference being given to the latter type. The Theissen disintegrator is a type of fan that dashes water, introduced through small jets, into a spray which is forced to travel in one direction while the gas is forced through the same channels in an opposite direction.

The high-speed or rotary disintegrator is a smaller diameter machine than the Theissen but rotates faster (500 to 700 rpm) and consists of a casing in which is mounted a rotating "squirrel cage." This cage is made up of two heads fastened to the shaft with a series of bars parallel to the axis fastened to the heads. Vanes mounted on the heads reduce the pressure-drop through the machine and force the incoming gas through the rotating bars, upon which water is sprayed, to the center of the cage where it is taken off to the clean-gas main. These machines are built up to 40,000 cu. ft. per min. rated capacity, require from 400 to 500 horsepower to drive them, and will clean the gas to under 0.02 grains per cubic foot.

In the Cottrell wet method of cleaning, the primary-cleaned gas is forced to pass through narrow channels or ducts across which an electrostatic field is maintained. The dust particles are "precipitated" or separated from the gas through the action of electrostatic charges. The molecules of gas are ionized and, in turn, induce electrostatic charges upon the surfaces of the small dust particles. Under these conditions, the dust particles are attracted toward the electrode of opposite polarity. There is also an "electrical wind" or corona effect. Current is supplied from an alternating-current source, stepped up in voltage by a transformer, and made unidirectional by a mechanical rectifier or by vacuum tubes. In most precipitators cleaning cool gases, the unit collecting electrode is a vertical tube, 8 to 12 inches in diameter, or it may be of parallel steel plates through which the gas is forced upward, or sideways. In the former type the discharge electrode is a wire suspended coincident with the long axis of the tube and, in the latter, multiple wires midway between the plates. A thin film of water flows over the inside edge of each tube or plate which washes it free of dust that has been deposited thereon, the dirty water being conducted to a Dorr thickener or settling basin. The precipitator usually is divided into two units with valving so arranged that one unit may be shut down for inspection or repairs while the other unit is operating. In some installations, the precipitator is mounted directly above the primary washer. The rotary disintegrator has the advantage of lower first cost but has higher power cost than the wet-type Cottrell precipitator. The cleanliness of the gas may be slightly in favor of the precipitator.

Secondary dry cleaners have included bag filters and Cottrell precipitators. The bag-type filters, sometimes called the Halberger-Beth type, consist of a number of fabric "socks" made of cloth, glass threads, asbestos threads, or very fine Monel metal screen (or a combination), supported at the top and closed on the bottom, into which the gas passes from the outside forming a film of dust on the sock which acts as the filtering

medium. When the film of dust becomes too thick, the exit end of the sock is closed thereby shutting off the gas flow and the sock is shaken or vibrated to drop the excess dust into a collecting hopper. The socks are stiffened both circumferentially and longitudinally to prevent their collapse. It has been difficult to find a fabric which does not break down under the temperature encountered with hot blast-furnace gas. Also, if the temperature of the gas should reach its dew point, moisture would deposit in the fabric thereby forming a deposit of dust on the sock which cannot be dislodged.

The dry-type Cottrell operates upon the same principle as the wet type but is constructed somewhat differently. The modification consists in using slabs of reinforced concrete as collecting electrodes to form channels for the passage of the gas, with the discharge wires suspended along the center lines of the channels. Periodically, each unit is taken out of circuit to permit the dust on the collecting electrode to fall into a hopper below. In this type, the dust is detached from the collecting electrode by suspended chains which are used as scrapers and which are operated by air cylinders. The dry-type precipitator has not cleaned the gas to the cleanliness required for small-checker stoves, gas engines, or for underfiring of coke-oven batteries.

Wash-Water Disposal—The water used to clean the gas in the wet-cleaning system contains about 40 per cent of the dust recovered, the other 60 per cent being recovered in the dry state. The wash water from all units is gathered together and conducted to a settling basin or thickener, present preference being given to the thickener. The usual settling basin is rectangular in plan and divided into two units so that one may be filling while the other is being emptied by a tractor or locomotive crane or by a permanent overhead crane. Each unit consists of several weirs over which the water flows in series, depositing a portion of the solids in each compartment. When one unit is full the water is then switched to the other unit and the full unit emptied. The handling of the wet solids into railroad cars is a difficult task, both at the basin and in the vicinity, due largely to the inability to make the railroad cars water tight.

A typical thickener consists of a circular, reinforced-concrete tank which may contain one or several compartments in each of which one or several arms revolve. The arms are driven from a central vertical shaft. Water enters at the center of the thickener and leaves over a continuous weir which follows the circumference of the compartment. Each arm carries a series of rakes or vanes set at such an angle that the solids which settle out are gently pushed toward the center of the compartment. The thickened solids, containing about 60 per cent water, are pumped out and delivered either pneumatically or by pump to a filter for further processing. The thickener may be built with either an open or closed top. Two types of filters are in general use, the cylindrical or drum type and the disc type. Both operate upon the underflow from the thickener, discharged into a basin into which an arc of the slowly rotating filter dips. The drum or disc consists of a framework over which canvas is stretched, and a partial vacuum is applied on the inside of that part which dips in the basin, thereby drawing the liquid through the canvas while the solids are retained by the canvas. When the drum or disc reaches another point in its rotation, a slight air pressure is applied on the inside of the canvas thereby bulging it and causing some of the filter cake to crack and drop off into a chute, the balance being scraped off by a scraper. The filtrate returns to the thickener and the filter cake, containing

about 25 per cent moisture, is delivered by railroad car to the sintering plant or, if the sintering plant is located adjacent to the furnace, the filters are designed as an integral part of the sintering plant itself.

Sintering Plant—A sintering plant's primary function is to agglomerate the flue dust and filter cake produced in the operation of the blast furnace into a product more acceptable for recharging into the furnace, and its secondary function is to beneficiate some of the finely divided ore. As the supply of high-grade ore is depleted, the secondary function may transcend the primary function and for purposes of description it is assumed that part of the charge contains a portion of fine ore for beneficiation. Modern sintering methods and equipment already have been described in Chapter 6 on iron ores, along with other methods of agglomerating fines.

Cast House—Since the molten iron is heavier than the slag or cinder, it lies in the bottom of the hearth and is cast or removed 4 or 5 times a day through the iron notch. This is an opening which is plugged with clay except when the furnace is casting. The usual method of casting is to drill into the clay with a pneumatic drill until the skull of iron is met, when an oxygen lance is used to burn through the iron skull. The iron flows into the main trough which has a skimmer located near its end. The skimmer separates any slag flowing with the iron and diverts it into the cinder ladles or to the cinder granulating pit. The iron continues to flow down the main runner from which it is diverted at intervals into the iron ladles, the control of this operation being accomplished by gates or shutters located in the runners (Figure 12—12).

The closing of the hole at the finish of the cast is accomplished by means of the **clay gun**. The gun is swung from an arm mounted on a pedestal in the cast house or on one of the furnace columns. It consists of a cylinder narrowed to a nozzle at the exit end and a capped opening on top into which soft clay is placed preliminary to casting time. The arm is normally parallel to the main trough but when placed in operation the arm is swung through 90° and the gun is tilted so that the nose enters the iron notch. The clay is forced from the gun by a plunger or screw; the plunger usually is operated by steam and the screw by an electric motor. Figures 12—13 and 12—14 show a typical clay gun.

The slag is flushed from the furnace through the monkey two or three times between successive casts and it flows through runners, in much the same way as the iron, into cinder ladles, into a granulating pit, or into a slag machine. The monkey is closed by pushing into the hole by hand, a steel, cone-shaped knob on the end of a long steel bar, called the **cinder bott**, or this may be accomplished mechanically by arms attached to a framework which supports and guides a water-cooled bott (Figure 12—15).

Iron Disposal—Iron ladles are of three types: the open-top ladle, the Kling-type ladle, and the mixer or Pugh-type ladle, all of which are refractory-lined. The **open-top ladle**, as the name implies, is an inverted frustrum of a cone with the top open to the atmosphere. The maximum capacity of this type is about 75 tons. The ladle is supported on the car or carriage by trunnion castings on the ladle, resting in corresponding castings on the car. In all types, the car usually is equipped with

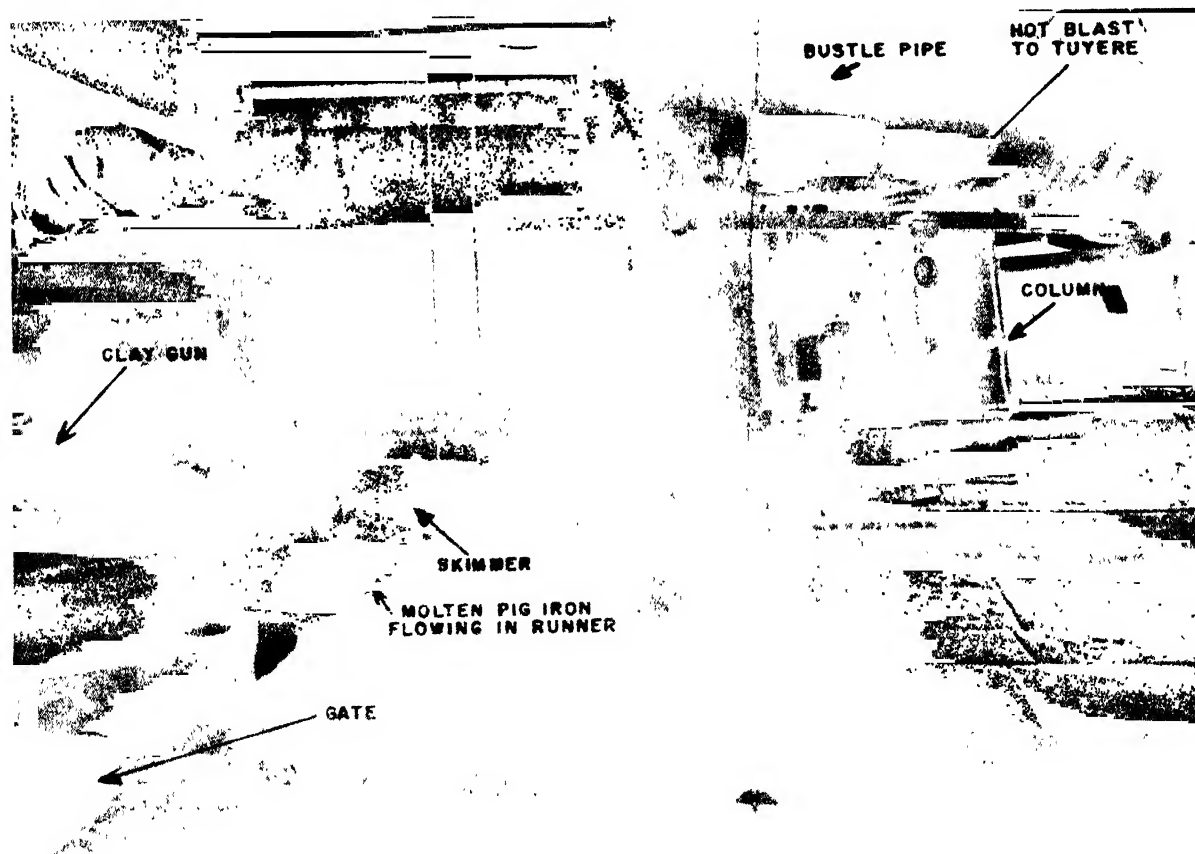


FIG. 12—12. General view inside the cast-house of a blast-furnace plant.



FIG. 12-13. Automatic clay gun, ready to be swung into position for closing the tap-hole of a blast furnace at the end of a cast. Clay is ejected under pressure from the nose of the gun, which is inserted in the tap-hole (see below).

air brakes and automatic couplers. The Kling type is somewhat spherical in shape with an opening at the top through which the iron enters. Its advantage is less radiating surface with consequent less skull formation. It is built up to about 115 tons capacity and is supported on its car similarly to the open-top ladle. The Pugh-type ladle is cigar-shaped with an opening at the top. It is built up to 200 tons capacity and differs from the other two types in that it is not removed from the car, the ladle itself forming part of the car between trucks. To empty the Pugh ladle, it is rotated about its longitudinal axis and, due to its large capacity in relation to the opening, offers the least radiating surface. In some cases, this type ladle functions as a metal mixer and the regular mixer is eliminated, with the iron going directly from the mixer ladle to the steel-making process. A

ladle of the Pugh type is shown in Figure 12-16.

Normally, the iron ladles carry the molten iron to the metal mixer but there are times when it is necessary to dispose of the iron by casting it over the pig machine. Present pig machines differ only slightly from each other;—the chain in one type carrying rollers on each side that ride on a track; while, in the other type, the sides of the chain form a track which rides on stationary rollers. The endless chain, carrying a series of parallel cast-iron molds or troughs with overlapping edges, passes over a head and a tail sprocket wheel. The molten iron is poured into the molds near the tail sprocket, solidifies, and is cooled by water sprays as the chain rises to the head sprocket, and falls from the molds into cars as the chain passes over the head sprocket. On the return travel of the chain, the molds are sprayed with a

FIG. 12-14. Automatic clay gun in operating position, closing up tap-hole at the end of a cast. The gun is operated entirely by remote control.



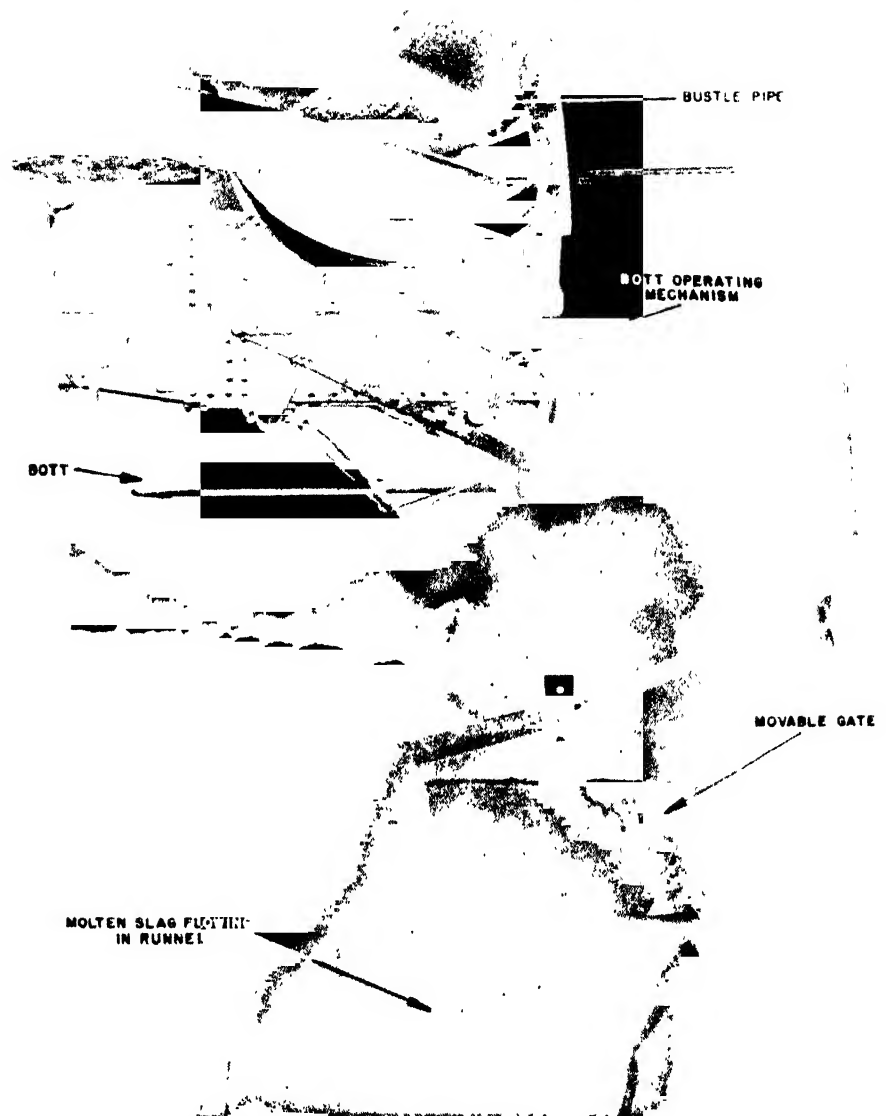


FIG. 12-15. Mechanical cinder bott withdrawn from cinder notch, with stream of molten slag running down cinder trough.

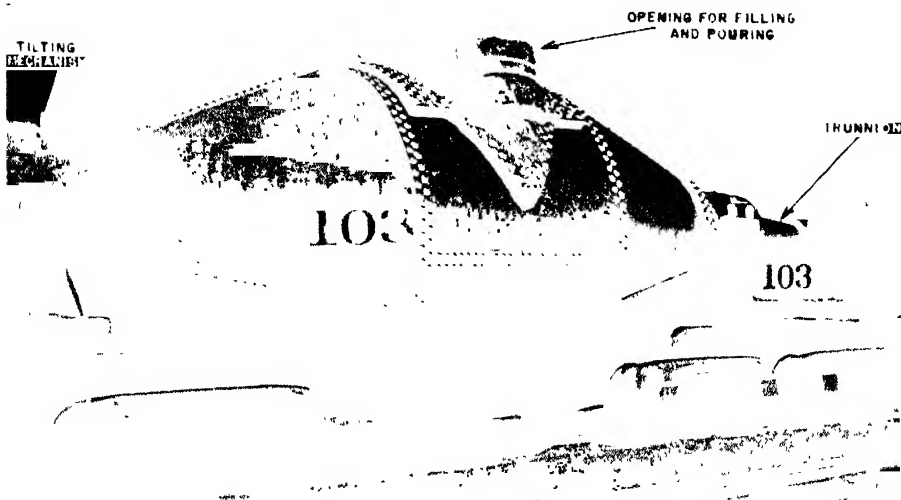
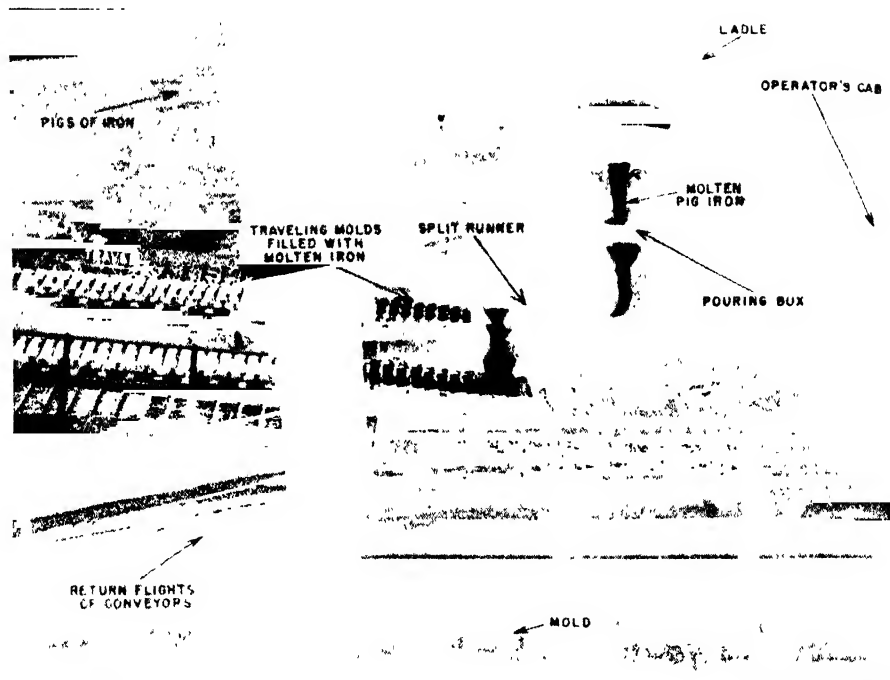


FIG. 12-16. A Pugh-type hot-metal transfer car of 150-ton capacity.

FIG. 12—17. Ladle pouring molten iron into traveling molds of a pig-casting machine.



lime wash to prevent sticking of the iron to the molds but, in cases where the pigs do stick to the molds, a "pig sticker" at the head end loosens the pig from the mold. Most machines are of the double-strand type, that is, they are built with two parallel chains. In the operation of the machine, iron is poured from the ladle into a small basin, which reduces the splash and which then divides the stream of molten iron to serve the two branches, one for each chain. The speed of travel of the chains and rate of pour of metal from the ladle are controlled so that a full-sized pig is formed in each mold. Pigs for use in the steel mills weigh almost 100 pounds but, for some commercial uses, pigs weighing as little as

30 pounds are poured. Operation of a pig machine is illustrated in Figure 12—17.

Slag Disposal—The slag is handled in one of four ways: it flows directly into cinder ladles; it is granulated; it flows directly into cooling pits; or it is made into lightweight aggregate for concrete or insulation. The cinder ladles are unlined, tulip-shaped, cast-iron (sometimes steel) pots mounted on cars or carriages on which they are moved from the furnace to the slag dump. The cars are equipped with air- or steam-operated cylinders which partially rotate the trunnion holding the ladle so that it can be dumped by the compressed air in the railroad train line (Figure 12—18).

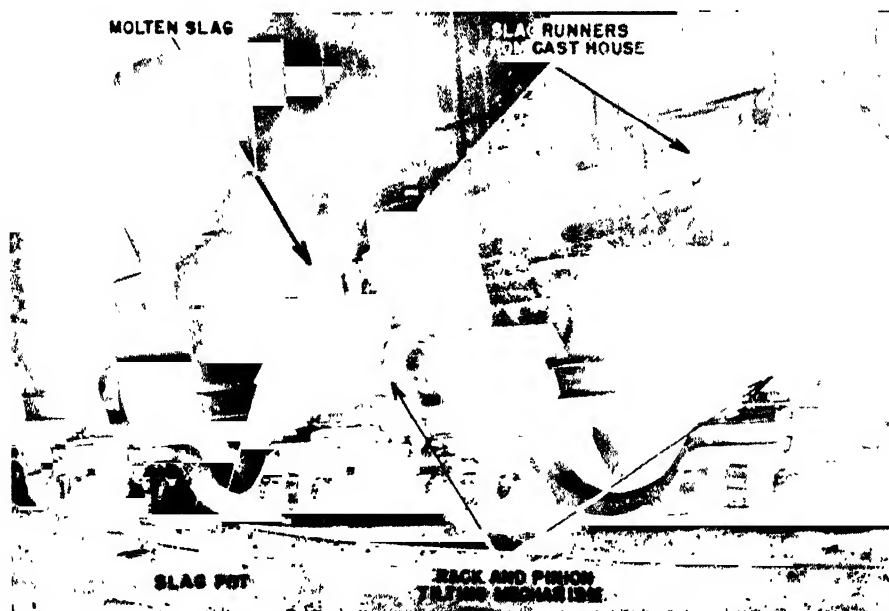
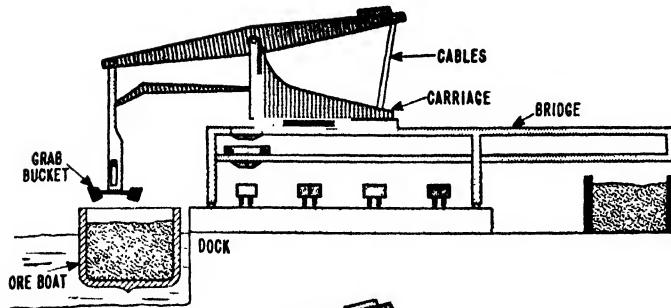
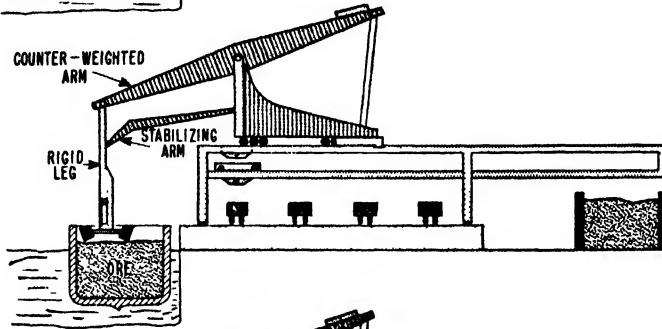


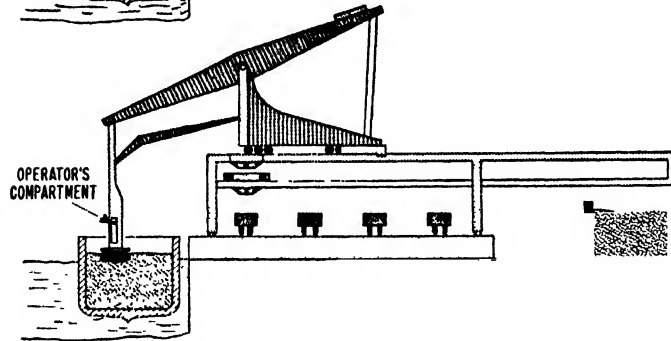
FIG. 12—18. Molten slag pouring from runners in floor of cast house into cinder ladle.



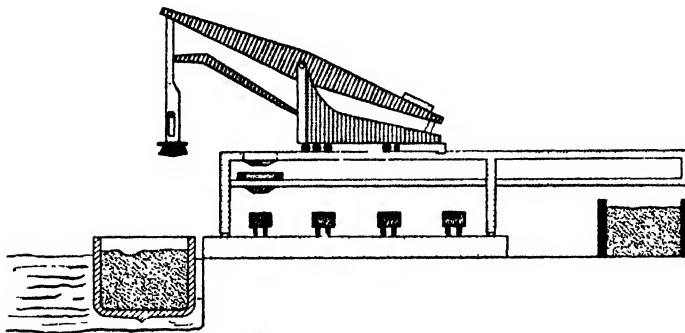
THE BRIDGE MOVES ALONG THE TRACKS PARALLEL TO THE DOCK WALL TO BRING THE CARRIAGE DIRECTLY OPPOSITE TO A HATCH OR OPENING IN THE DECK OF THE ORE BOAT. THE CARRIAGE ADVANCES ALONG TRACKS ON THE BRIDGE TO BRING THE OPEN GRAB BUCKET DIRECTLY OVER THE HATCH.



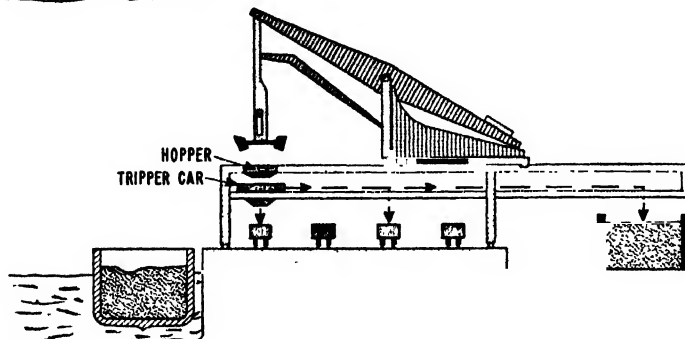
THE RIGID LEG SUPPORTING THE GRAB BUCKET IS LOWERED BY TILTING THE COUNTERWEIGHTED ARM UNTIL THE GRAB BUCKET CONTACTS THE ORE PILE IN THE HOLD OF THE ORE BOAT.



THE GRAB BUCKET IS CLOSED, TAKING A "BITE" OF UP TO 20 TONS OF IRON ORE.



THE COUNTERWEIGHTED ARM RAISES THE LOADED GRAB BUCKET OUT OF THE HOLD OF THE ORE BOAT.



THE CARRIAGE MOVES BACK TO PERMIT THE GRAB BUCKET TO DISCHARGE ORE INTO A STATIONARY HOPPER ON THE BRIDGE. GATES OF THE HOPPER OPEN TO DUMP ORE INTO A DUMP-BOTTOM TRIPPER CAR WHICH CARRIES THE ORE EITHER TO RAILROAD CARS OR TO THE ORE TRENCH, WHILE ANOTHER "BITE" IS BEING TAKEN BY THE GRAB BUCKET.

FIG. 12-19. The sequence of operations of a Hulett unloader transferring iron ore from the hold of an ore boat to railroad cars or storage areas on the dock.

Sometimes the molten slag, if of suitable composition, is dumped in specially prepared dumps, from which it is removed by a power shovel (after weathering for a period of several weeks), screened, and sold as an aggregate for concrete or as ballast. In granulating slag, provisions are made for the molten slag to strike a stream of high-pressure water as it falls from the lip of the runner, forming a popcorn-like mass which later is loaded onto cars by grab bucket. In this form, it is used in the manufacture of cement. Some of the newer furnaces flush the slag directly in long, rectangular, concrete walled pits, open at one end, from which it is removed by a power shovel and loaded into railroad cars. The pits are usually two in number, one being filled while the other is being emptied, with from five to seven days being required to fill a pit. In this form the slag, after screening, is used as an aggregate for concrete. In a few cases, the slag as it flows from the runner is met by a small stream of water, steam, air, or a combination of these which partially granulates it, but with no excess water, so that the product formed consists of lumps which are filled with air cells. After screening out the fines, the product is used for insulation or, when cement is added, as a lightweight concrete. By adaptation of the method used, it is sometimes possible to make slag disposal a profitable operation.

Handling Ore from Vessel—The ore used in many furnace plants is mined at considerable distance from the plant and requires transportation either by rail or water, or a combination of both, before reaching the plant site. Coke and limestone, the other essential ingredients, are rarely stored in large quantities and are consumed as currently received. In the case of ore, weather conditions are often such as to prevent mining and transportation during the colder months so it is necessary to store during the warm months approximately half a year's supply adjacent to the furnaces. Since many of the furnaces in this country receive their ore from the Lake Superior District, as discussed at length in Chapter 6, the transportation of ore from there is described very briefly here as being typical of water transportation of ore. Ore, at the head of Lake Superior, is loaded by gravity, from long piers extending into the harbor and carrying overhead bins, into specially constructed ore boats capable of carrying up to 20,000 net tons of ore per trip. These boats have a speed of 12 to 15 miles per hour and a draft of 20 feet. Upon reaching their destination, the ore is unloaded by Hulett unloaders or by special unloading rigs if the plant is large enough to justify this special equipment, or, if the plant is smaller, by an ore bridge which extends over the vessel and permits the ore-bridge bucket to unload the vessel.

The Hulett unloader (Figure 12—19) consists of a carriage mounted on wheels which permit it to move in and out in a direction at right angles to the dock, and which carries a counterweighted pivoted arm, on the harbor side of which is located the grab bucket capable of handling 20 tons at a time. The operator rides in a compartment near the bucket and controls its movement from this location. The bucket itself can be rotated, and the entire unloader can be moved parallel to the dock. The bucket moves downward through the open hatch of the vessel, takes its "grab," and then moves upward and backward depositing its grab into a hopper. Under the hopper is a rotating feeder which places the ore into a small car that runs backward either to a hopper over railroad cars or to an ore trench in the ore yard. The ore bridge picks the ore from the ore trench and distributes it onto the ore pile.

The unloading rigs perform the same function as the

Hulett but differ somewhat in construction. Instead of the bucket being carried by a rigid arm, it is hung from a trolley resembling a crane trolley; consequently, it cannot be rotated and it cannot clean out the hold of the vessel as completely as the Hulett, and the additional use of a small bulldozer and hand labor is necessary to complete the job.

Car Dumper—For those plants located some distance from the point of vessel unloading, or where ore is received directly from the mines by railroad, a car dumper is the usual means used to unload ore. The ore, arriving at the plant in trainload lots, is switched to a siding ahead of the car dumper and the cars are unloaded one by one in rapid succession. A car, being pulled up an incline to the platform of the dumper by a steel cable of a "mule" or "barney," is lifted bodily and turned over so as to empty its contents into a large transfer car which, in turn, discharges into an unloading trough opposite the location in the stock yard for that particular grade. The dumper then resumes its former position and the empty car is pushed off the platform, by the next car of ore, to an incline, down which the empty cars move to a siding. The transfer cars are usually designed to hold two railroad cars of ore. A nozzle carrying high-pressure water and mounted upon a rigid swivel can be turned on and directed by the operator to any crevice in the upturned car bottom where the ore has not been dislodged. The quantity of water used is so small that it is insignificant in the ore. The ore falling from the car passes over grids that hold back any large lumps and then drops into a bin. There are gates



FIG. 12—20. Car dumper discharging iron ore from a railroad car into a transfer car.



FIG. 12-21. View of an ore yard serving the blast furnaces in the background.

at the bottom of this bin which are operated by the transfer-car operator in filling the transfer car. A few dumpers have been built which are movable, in that they are able to move to any location and dump the ore directly into the ore trough. These dumpers, however, must be attended by a locomotive when operating; consequently, only one car can be spotted on the dumper at a time. Preference seems to be for the stationary dumper with transfer cars. One type of car dumper is shown in Figure 12-20.

Ore Yard and Ore Bridges—The ore yard (Figure 12-21) is a large space, sometimes with concrete bottom and sides, which parallels the furnace bin structure and serves as storage for approximately a half-year's consumption of ore. Its width is determined by the practical maximum span of the ore bridge, which for a bridge supported at each end amounts to about 350 feet. Other types of bridges are supported by two legs which run longitudinally through the ore yard (dividing the yard into three separate piles); such bridges have cantilevers on the outer sides of the two supporting legs. This type gives a longer trolley travel but does not necessarily store any more ore per running foot because of the space occupied by the supporting structures. A trolley running on the bridge proper carries a grab bucket of about 15 tons capacity which, when stocking, lifts the ore from the dumping trough and distributes it on the pile, or removes the ore from the pile and deposits it either directly into the bins or into a bin or transfer car (Figure 12-22) which in turn distributes the ore. Some bridges are provided with a bin hanging from the bridge into which the bucket dumps permitting the bridge to keep working while the bin car is dumping its load. In order to obtain the maximum

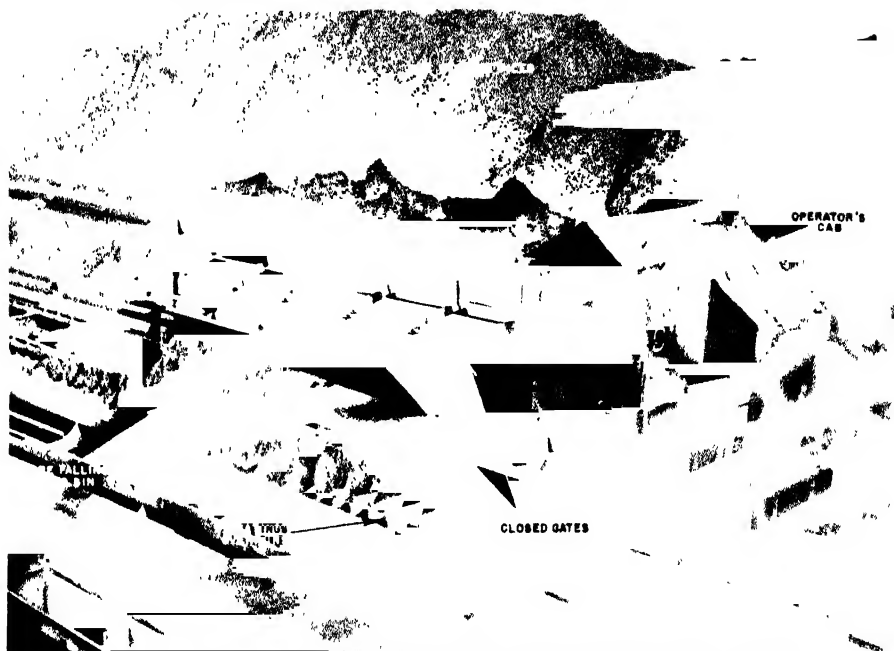
benefit of mixing, the ore is stocked in horizontal layers and removed in vertical slices. For this reason, the use of ore direct from hopper cars unloaded from the trestle into the bins is undesirable. The details of the ore-handling system will vary considerably but the general scheme is essentially as stated.

Trestle and Stockhouse—If it were not for the fact that all parts of a blast-furnace plant are essential to its operation, the stockhouse might be considered as the most important unit of the plant, with the exception of the furnace itself, since its function is to deliver to the furnace the correct quantities of ore, fuel and flux without which the furnace cannot operate.

The constructional details of the trestle differ considerably but a trestle of modern design consists of a reinforced concrete wall on the stock yard side and steel columns on the furnace side, between which is a crosswork of transverse and longitudinal girders. On top, these girders support three or four railroad tracks, and on the bottom, the bins proper. The track nearest the stock yard carries a side-dump bin car, the next track carries railroad cars containing materials such as limestone, dolomite, scale, scrap, etc., which are unloaded by hand, and the next one or two tracks carry coke cars. When there are more than six furnaces in a line, it is desirable to have two coke tracks so that the coke cars, not fully unloaded, may be by-passed. Where the coke bins are filled by a belt conveyer, one coke track for emergency use is sufficient. This construction results in two lines of bins in the stockhouse, one line containing ore, flux and miscellaneous materials and the other containing coke.

A scale car, equipped with scales to weigh accurately each material drawn, runs underneath the first line of

FIG. 12-22. Transfer or bin car emptying ore into stock-house pocket.



bins. The car may have a capacity of up to 40 tons and carries two pockets. The ore and miscellaneous material bins are equipped with gates on the bottom which may be operated either mechanically or by hand (Figure 12-23). The car delivers its load by discharging into chutes leading to the skips. Underneath the second row of coke bins, a belt conveyor, fed by vibrating feeders at each coke bin, runs toward the skip pit while another conveyor on the other side of the skip pit performs the same function for the coke bins on that side. Each conveyor discharges upon a vibrating screen where the coke fines are screened out and the oversize discharged into a weigh hopper. Here the coke is weighed and at a pre-

determined weight an electrical contact stops the belt and vibrating coke feeders. Provision also is made usually to stop the belt and feeders in case a predetermined volume, rather than weight, is desired. From the weigh hoppers, the coke is discharged directly into the skips. Considerable time is saved by this method, as the operator can be drawing ore while the coke loading is being taken care of automatically. The undersized coke from the vibrating screen is removed by a small bucket hoist, or by conveyor belt, to a storage bin where it is rescreened into coarse and fine. The coarse may be recharged separately into the furnace or may be sold as domestic coke, while the fines are used under boilers,

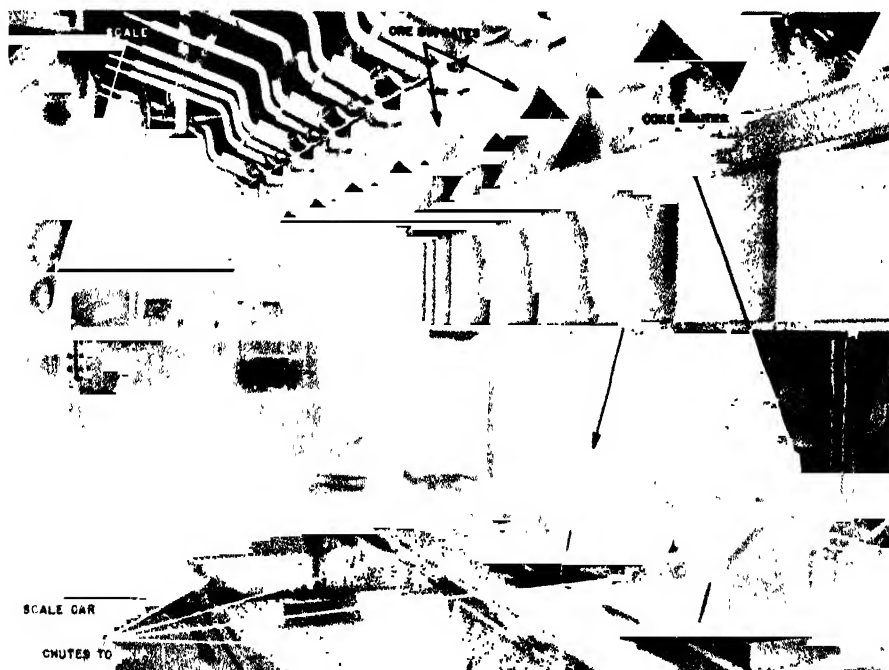


FIG. 12-23. Scale car in stockhouse, showing overhead ore bin gates from which car is filled, and chutes that carry coke from bins directly to skips.

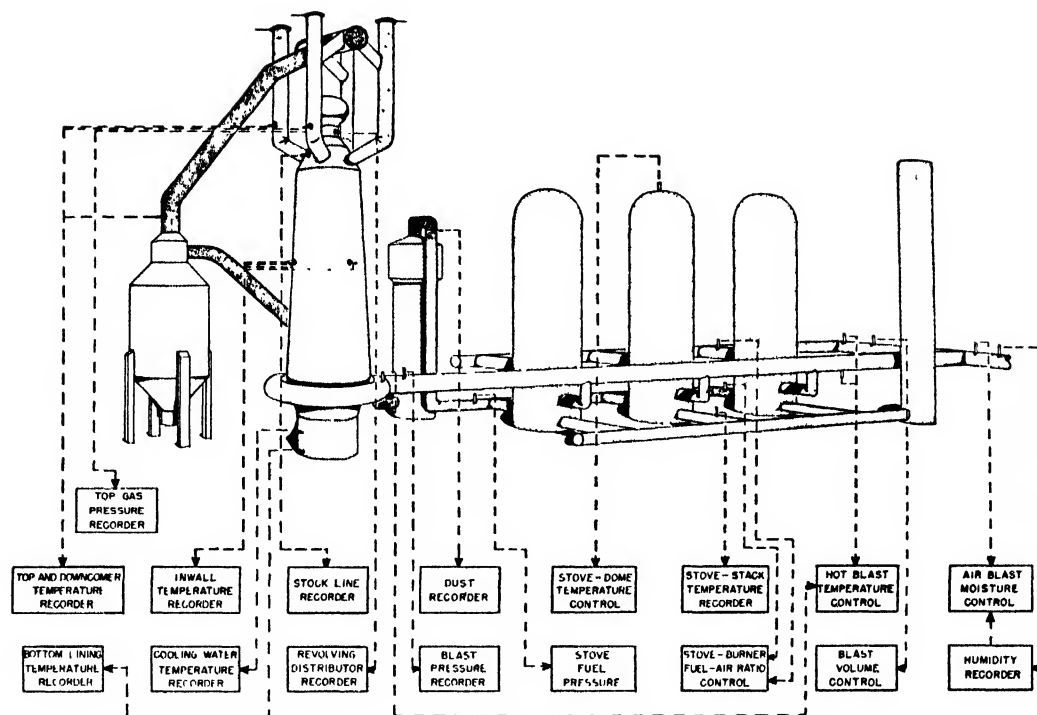


FIG. 12—24. Instruments assist the furnaceman by automatically indicating, recording and controlling temperatures, pressures, flows, etc., at the important points in the blast-furnace system indicated in this schematic representation of a blast furnace and its auxiliaries.

as fuel in the sintering processes, and to make up the bottom of soaking pits. In the bucket hoist the same scheme as that just described is used for the ore, flux and miscellaneous materials, but the bucket itself is placed, on descending, upon a revolving cone which keeps the bucket revolving during the time it is receiving the charge. A **charge**, or **round**, is that minimum combination of skip or bucket loads of material which together provide the balanced complement necessary to produce hot metal of the desired specification. Normally, the entire charge is deposited on the large bell by successive skips or buckets and enters the furnace upon lowering of the bell. However, through preference or because of the limited capacity of the large bell hopper, the charge or round may be "split" and require two drops of the large bell.

Blowers, Boilers, Pump Houses, Etc.—These vital parts of the blast-furnace equipment present features of more interest to engineers than to metallurgists. The blowing of the furnace, on all later installations, is by turbo blower, this type of equipment having almost com-

pletely displaced internal-combustion gas engines and steam reciprocating engines. It is noted that many installations now receive steam at 700 pounds per square inch pressure with a total temperature of 750°F. There has been at least one installation abroad where a gas turbine has been used to blow a blast furnace. The boilers almost universally burn secondary clean gas, with unit coal pulverizers provided at each boiler for auxiliary firing. The water system is usually designed to deliver water at tuyere level at 30 pounds per square inch head. The water is treated with lime if it is of a corrosive nature, and may be chlorinated sufficiently to prevent algae growth.

Instrumentation and Control—The blast-furnace operator is aided and guided in establishing and maintaining the best possible balance between the many variable factors affecting furnace operation by numerous automatic devices which indicate or record conditions at various points in the furnace system (Figure 12—24) and may operate control mechanisms that automatically regulate important variables.

SECTION 5

OPERATION OF THE FURNACE

When a blast furnace is completed and provided with as much of the equipment described above as is necessary or desired, the active campaign to produce iron may begin. In blast-furnace parlance, the process of

starting a furnace is called **blowing in**. It is carried out in three steps, drying, filling, and lighting.

Drying—Newly constructed furnaces and stoves, or new linings, must be carefully and thoroughly dried

before they are put into service. In the case of the blast furnace, any one of several methods may be employed. In one method the drying is accomplished by a gas fire in the hearth but this method has become obsolete because of some hazards involved. Another and safe method is to build three or four "Dutch ovens" at different points outside of the hearth, fire these ovens with coal or coke, or a combination of the two, and allow the products of combustion to be drawn through the cooler openings into the bottom of the furnace. This method requires the assembling of the fuel at the Dutch ovens and the disposal of the ashes. A third method makes use of the hot blast for drying. In carrying out this method, the stoves are heated as when the furnace is operating, and air from the blowers is passed through the stoves, then to the bottom of the furnace through the regular connections and a number of special pipes extending from the tuyeres inside the furnace to the bottom. While this method also is perfectly safe, it can be used only where gas is available for heating the stoves, and is not practicable when the drying must be extended over long periods of time. Still another safe method is to build a fire of scrap lumber in the hearth of the furnace. The fire may be kept burning for several weeks. During the drying out, the openings at the bottom of the furnace are kept covered to prevent the drawing in of an excessive amount of air, while the bleeders at the top are kept open. In all cases, the heat must be applied very gradually. About ten days are required with these methods, but when feasible to do so, this time well may be prolonged to two weeks or more. Since the mass of brickwork in the hearth is quite large, it is customary to dry this part of the furnace as each course of bottom block is being laid. This is accomplished by placing unit heaters or knock-down salamanders on top of the newly laid course during the off turn. Some operators employ thin electric heating strips between the lower two courses of bottom blocks and dry by electricity. In some cases, steam is turned into the cooling staves to help the drying. Following the drying period, some heat must be kept up until such time as the furnace is put on blast to prevent absorption of moisture from the atmosphere. Small unit heaters at the iron notch and cinder notch have worked quite successfully in accomplishing this objective.

Filling—After the furnace is sufficiently dried, it is allowed to cool slightly, and then the important process of filling is begun. While different individuals pursue slightly different methods, the general scheme is carried out rather uniformly. Briefly stated, it consists of first placing wood and coke, or coke alone, on the bottom to a height somewhat below the level of the tuyeres, about which any easily ignited material, preferably charcoal, is piled; then following this material with a large quantity of coke, mixed with enough limestone to flux its ash, and gradually introducing ore with the proper amount of flux. Good practice requires that this initial volume of coke should be about half the cubical contents of the furnace. Sometimes, to get an easily fusible slag and a good volume of it, blast-furnace slag may be introduced ahead of the ore. These are called **blow-in burdens**, and additions are made until the furnace has been filled to the stock line, when it is ready for lighting.

Lighting—The lighting of the materials in the bottom of the furnace may be done in several ways. If the space in front of the tuyeres has been filled with light kindling wood, oil is poured or sprayed in at the tuyeres until the wood is thoroughly soaked with it. Then, with the furnace isolated from the gas-main system beyond the dustcatcher, mains and bells closed, the bleeder and ex-

plosion doors are opened, a light blast is turned on, and the wood ignited. Often, a wood fire is built in the stove nearest the furnace, and the oil-soaked wood is ignited by blowing sparks over with the blast. Others prefer to light the furnace with hot blast alone or with an oxygen lance. With a light blast on, the wood soon burns away, and the stock begins to settle, after which the blast volume is gradually increased. Some furnacemen start off, after the fires are well caught, with a fairly high blast volume for a few minutes, in order to drive the flames well in toward the center of the furnace and consume the wood quickly, as it is thought that a better initial settling of the stock is thus obtained. As soon as the stock gives signs of settling, the blast volume is reduced to that normally used for the rest of the blowing-in period, which volume is at first about $\frac{1}{4}$ that used when the furnace is in full blast. Up to this point a great deal of gas and smoke escapes from the furnace openings, and great care must be exercised, for the gases contain a high percentage of carbon monoxide. Great care is also required to prevent explosions, because, as pointed out in the chapter on fuels, mixtures of furnace gas and air in a wide range of proportions are explosive. Also, the gas from a newly lighted furnace contains hydrogen above that normally present. Since the interstices of the stock in the furnace are filled with air at first, an explosive mixture may be formed any time soon after the lighting, and if this mixture should be ignited it might cause serious damage. The difficulty is overcome generally by allowing the gas to escape through the bleeders to the atmosphere for several hours, after which time valves to the gas-main system are opened. Outlets at the ends of the gas mains are kept open until all the air has been expelled, which condition is indicated by the color and odor of the escaping gas. Both men and fires are kept away from these openings until the outlets have been closed.

Heating the Bottom—Another feature connected with the lighting of the furnace is heating up the bottom, which is warmed by the drying-out fires to only a slight degree as compared with the temperature required to keep molten the slag and iron that form. In order to have the bottom at the proper temperature when slag begins to form, two methods are employed, both of which involve leaving an opening at the tapping hole so as to draw flame downward from the tuyeres upon the bottom. In the first method, a round tapered wood plug, three or four inches in diameter at the smaller end, is placed in the taphole and the space about it is packed full and tight with clay. With the rise in temperature due to the burning of the wood and coke in the bottom, the clay sets, and this plug is then removed, which permits the flame from within to shoot forth, thus heating up the runner outside as well as the bottom inside the furnace. When slag begins to flow from the taphole, the opening is closed until time for tapping the first iron has arrived. In the second method, an iron pipe, about four inches in diameter, is placed in the furnace before it is filled, so that one end protrudes from the taphole outside the hearth, while the other extends to the center of the furnace. The space about the pipe where it passes through the wall of the hearth is tamped with clay or ball stuff, which is also built up about the part of the pipe within the furnace for a foot or so from the hearth wall. When the furnace is lighted, the gas flame is drawn to the center of the bottom to pour forth from the exterior end of the pipe. This pipe need not be moved until a fairly large flow of slag is attained, when it is drawn from the taphole, which is immediately closed, as in the case of the wooden plug.

Heating the Stoves—The temperature of the hot blast when the furnace is in full operation is about 1000°F (540°C) and it is a great help if the stoves can be heated to provide this temperature at the time of lighting of furnace, for the furnace and filling are cold to the bottom. The required stove temperature usually is obtained by the use of blast furnace gas, but in the case of isolated furnaces where gas is not available before starting up the furnace, the stoves are heated as hot as possible by wood and coal fires, by natural gas, or by special fuel-oil burners before lighting the furnace.

Blast Control—For a good blow-in it is now regarded that the volume of blast blown should be increased fairly rapidly (as opposed to former practice requiring as much as a month to get to full wind). Rapid increase in the blast produces better penetration at the tuyeres; the furnace more quickly starts to work regularly. An empirical figure of about 125 cubic feet of blast per minute per square inch of tuyere area seems to produce the desired penetration. The usual procedure is to light the furnace with at least half of the tuyeres plugged and then to open a tuyere at a time as the blast is increased. In addition, many operators use a tuyere smaller in diameter when blowing-in than those which will be used when the furnace reaches normal operation. The furnace is lighted with both the "snorter" (snort valve) and peep sights open which gives hardly more than a breeze entering the furnace. The peep sights are closed gradually and then the snorter also is closed gradually so that 4 or 5 hours after the furnace has been lighted, the snorter is completely closed. The divergence in opinion as to how fast the blast should be increased is indicated in the following tabulation of two blow-ins.

Per Cent of Full Wind Being Blown	Hours from Time Snort Valve is Closed	
	Operation A	Operation B
25.0		Start
40.0	Start	12.0
50.0	4.0	20.0
60.0	9.0	28.0
70.0	19.0	36.0
80.0	34.0	48.0
90.0	54.0	72.0

In both operations it requires several days to get the last ten per cent of the wind on the furnace. The economic importance of obtaining production from such a costly assembly of equipment is obvious.

Flushing and Casting—About 18 hours after the snorter has been closed, there will be a sufficient accumulation of slag to flush. This is done by removing the bott from the monkey, and pricking through the solid slag with a steel bar. The bleeder is closed several hours after the snorter is closed and the gas now can be used in the stoves and boilers. Up to this time the blow-in blast-furnace gas is higher in carbon monoxide and hydrogen than normal, with generally unsatisfactory combustion characteristics. With a multiple furnace plant, the procedure is to turn the steam off the dustcatcher and open the water seal or goggle valve between the dustcatcher and the gas line, thus permitting the gas from the other furnaces to flow momentarily through the open bleeder, after which the bleeder is closed. In a single furnace plant, a connection such as a boiler burner is left open at the extreme end of the gas main and the gas forces the air ahead of it through the open connection. Extreme care must be exercised at this point that no fires are adjacent to the open connection which might ignite the explosive mixture being forced out of the connection.

It requires about 24 to 30 hours before enough iron accumulates for the first cast. A hole is bored by means of a long auger or drill, electrically or pneumatically operated, into the clay plug of the tapping hole until the hard iron skull is reached. The loose dust is blown out of the hole by a jet of compressed air. The **splasher** having been put in place, the opening of the hole is completed by an oxygen lance, a small quantity of coal having previously been placed into the end of the hole to provide a means of "lighting the lance." The coal itself is ignited from the heat of the red hot skull, and soon the iron skull is melted. The iron then flows down the runner, under the skimmer, and into the ladles provided to receive it. During the flow of the metal, samples of the iron for chemical analysis and fracture tests are taken by collecting small spoonfuls from the main runner. The iron is usually analyzed for silicon, sulphur, manganese, and phosphorus. The slag which follows the iron near the end of the cast is stopped by the skimmer where it is run off through a runner to the slag ladles or granulating pit. When the iron has almost ceased to flow and gases are pouring from the hole, the blower signals the blowing room to reduce the blast and also may open the snort valve, it being the practice with modern guns to stop the hole with about 10 pounds blast pressure on the furnace. The clay gun is then swung into position for stopping the hole and the clay forced into the tap hole.

Care of Runners—After the tapping hole has been closed (from one to three minutes being required), the troughs are emptied, and preparations for the next cast are begun. The runners are cleaned carefully of both metal and slag, and their inside surfaces are carefully brushed with a thick clay or loam slurry, which, when dry, protects the trough, and prevents the iron from sticking to the runner. More recently a preference has been shown for a carbon-lined main trough due to the reduced scrap produced, since the iron does not stick to the carbon brick. For such main runners a steel plate trough is built, lined next to the plate with firebrick, and a carbon-brick facing placed next to the firebrick. The carbon brick is plastered with clay before cast time in order to keep the carbon from oxidizing during cast. Water can be turned directly on the red-hot carbon after a cast without any deleterious effects.

Sampling the Iron—Sampling pig iron is a very important part of every tapping (cast). As the iron is graded by chemical composition, care must be taken to obtain a sample that will be representative of the whole cast. This sample, therefore, is generally made up of a number of equal portions taken from the main runner at the farther side of the skimmer and at periods corresponding to the first, middle, and last thirds of the cast. These samples consist of small castings made by pouring the metal into a mold. Chill tests of all slags are also taken and a daily composite analysis made for each furnace.

Changing Stoves—The temperature of a furnace at the hearth is a matter of great importance, as this is one of the two main factors which control the quality of the iron produced. The other main factor is the composition of the slag. One of the means of regulating hearth temperature is by control of the hot-blast temperature. This may be raised or lowered by use of the mixer valve, and can be kept high by proper manipulation of the stoves. It has been said with some justice that when the slag is of proper composition and temperature, the quality of the iron automatically will take care of itself. As a part of the routine of blast-furnace work, the tending of stoves is of importance. They must be kept clean and be changed regularly and at not too long intervals.

SECTION 6

CHEMISTRY OF THE PROCESS

The reactions of the blast furnace have been thoroughly investigated, not only by deductions from observation and controlled laboratory experiments, but also by experiments conducted with small experimental furnaces and indeed with full size furnaces—the latter being conducted through holes left in the wall when the furnace was built. To summarize the knowledge thus gained in the briefest possible form, the several possible reactions for each element entering the furnace are set down. At one zone or another each may occur, though the final result is the same regardless of the precise stages actually passed through by any particular part of the charge.

Reactions of Oxygen and Carbon—20.8 per cent of the blast by volume or 23.2 per cent by weight is oxygen, which enters the furnace at a high temperature, say 1000°F (540°C) and, coming in contact with hot coke, immediately reacts with carbon, giving off heat, thus:

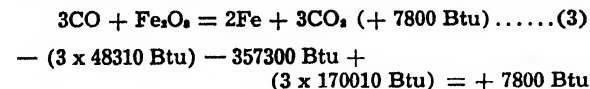


In the presence of an excess of carbon at a high temperature, CO_2 is at once reduced to CO, and 73390 Btu are absorbed per pound-molecule of CO_2 .

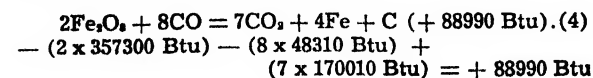


The net heat, then, from (1) and (2) is 170010 - 73390 = 96620 Btu, or 48310 Btu per pound mol. of CO.

At moderately high temperatures the CO gas acts upon ferric oxide as a reducing agent and in so doing will liberate heat at the same time, thus:

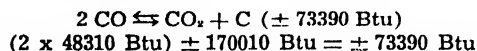


At temperatures ranging from 482°F to 1292°F (250 to 700°C), the latter a dull red heat, the reduction of Fe_2O_3 by CO may take place in three steps, the Fe_2O_3 being successively reduced to Fe_3O_4 , FeO, and finally to Fe. That the reduction of the ore in the blast furnace does take place in this way is suggested by the fact that a large part (60 to 74 per cent) of the flue dust ejected from the top of the furnace is magnetic though only Fe_3O_4 may have been charged. If all three of these reactions proceed to completion, the total heat liberated and the total iron reduced would be the same as that in reaction (3) for equal weights of CO. A very interesting reaction may take place between Fe_3O_4 and CO at low temperature as in Reaction (4), in which carbon is deposited and heat is liberated, thus:



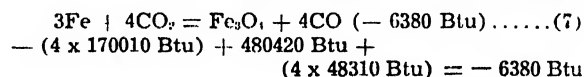
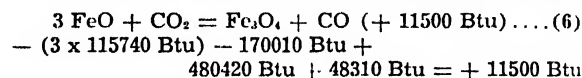
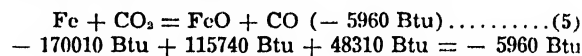
* In the interests of uniformity, all heats of formation used in this book, unless otherwise stated, are expressed in Btu per pound-molecule at 64.4° F (18° C), based on the values given in "The Thermo-chemistry of the Chemical Substances," by F. Russell Bichowski and Frederick D. Rossini, Reinhold Publishing Corp., New York (1936). It is, however, recognized that the heating value of coke carbon is higher than that given for carbon in the graphitic form; for coke carbon the value is 174,600 Btu per pound mol., or 14,550 Btu per pound of coke carbon.

Carbon is not deposited by the action of CO upon FeO or Fe_3O_4 (magnetite), but it may be deposited at temperatures below 1112°F (600°C) by the action of metallic iron upon CO, thus: $Fe + CO = FeO + C$. Also, as the ore is reduced, carbon may be deposited by the reversible reaction,



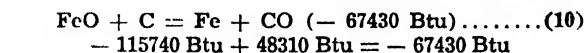
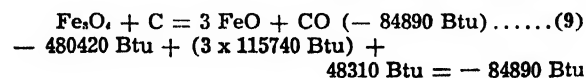
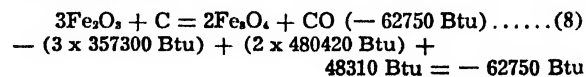
Since this is an equilibrium reaction, the amount of carbon deposited depends upon the ratio of CO to CO_2 and the temperature, reaching a maximum at about 942°F (505°C) with the ratio $CO/CO_2 = 0.4$. This free carbon penetrates a porous ore to some extent, and seemingly becomes a more powerful deoxidizer of the ore than CO at temperatures above 1382°F (750°C).

The CO_2 formed in the preceding reactions and from the decomposition of limestone may act as an oxidizing agent, absorbing or giving off heat, as shown in Reactions (5), (6) and (7).

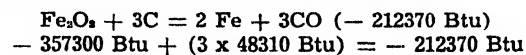


These reactions may take place at temperatures ranging from about 662° to 1562°F (350° to 740°C) and to a degree are governed by the relative amounts of CO_2 and CO in the furnace gas.

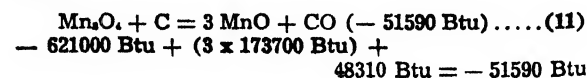
Carbon alone is also a reducing agent toward oxides of iron at temperatures of 942° to 1292°F (505° to 700°C) but the reduction of ore by carbon alone absorbs much heat whether the reduction takes place at these or at higher temperatures, as shown by Reactions (8), (9) and (10). However, the carbon monoxide produced can be utilized in the indirect reduction of more iron oxide and thus liberate heat, minimizing the net heat-absorbing effect.

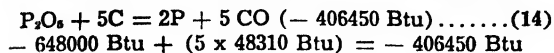
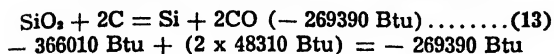
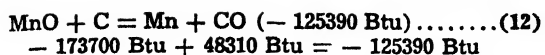


Under proper conditions the reduction of Fe_3O_4 by solid carbon may take place in a direct way, thus:



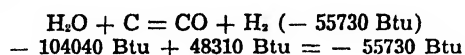
At very high temperatures—say around 2700°F—carbon in large excess may reduce manganese, silicon, and phosphorus oxides, the reactions being represented thus:



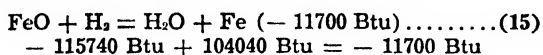


Since it is probable that silica often does not exist as such, but rather in chemical combination with other oxides, and since the silicon is dissolved in iron, equation (13) cannot be used as absolute in determining the heat absorbed.

Some oxygen also enters the furnace as water vapor in the blast, where the following endothermic reaction occurs:



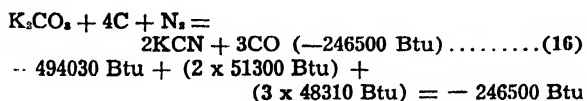
The hydrogen formed may do work temporarily by reacting with iron oxide and reducing it thus:



The water so formed is again decomposed as shown by the presence of hydrogen in blast-furnace gas. Therefore, the net energy result from water vapor is a loss.

In this connection it should be noticed that, since carbon is the only fuel employed, the carbon-oxygen reactions must be relied upon to furnish the heat required in the process, and that only a few of these are heat producing. The net heat from Reactions (1) and (2) and the heat evolved by Reactions (3) and (4) produce most of the heat absorbed by other modes of reduction, also that required to dry the raw materials, to decompose the carbonates and hygroscopic minerals, to melt the iron and slag, and to replace the heat wasted. On this account they are among the most important reactions occurring in the furnace.

Behavior of Nitrogen—Nitrogen and the other inert gases of the air, totaling 79.2 per cent of the blast by volume, pass through the furnace, for the most part, unchanged chemically. Since they equal in weight about six-tenths of all the other materials entering the furnace, they play an important part in heat convection, and make a source of unavoidable heat waste. Some nitrogen, however, possibly may react with alkali carbonates and carbon to form salts of hydrocyanic acid.



This reaction explains the trace of cyanogen (CN), analytically detectable in blast-furnace gases. In the iron blast furnace the cyanide compounds are largely decomposed or oxidized within a short distance above the bosh, and in furnaces making ferroalloys, the gas is likely to contain more cyanogen, perhaps because the zone of cyanide decomposition is nearer the top.

Carbon in Pig Iron—Other than the metal iron itself, carbon is the most important element in pig iron. In iron for steelmaking, since the carbon must be removed largely by oxidation, it may be said to be a fuel. Due to the nature of the process and the chemical properties of the elements involved the amount of carbon in pig iron cannot be controlled wholly in the blast furnace except, indeed, within a relatively narrow range above a certain minimum which varies for different grades. The reasons for this statement may be explained briefly

as follows: in the blast furnace, the ore is reduced largely in the upper part of the furnace, and the reduced iron is melted in the fusion zone above the tuyeres, whence it falls in drops through the slag into the hearth where it lies in contact with the bottom of the central column of coke. In the hearth, the iron gives up sulphur to the slag, the sulphur contents of the metal and slag finally approaching a condition of equilibrium. The amount of carbon absorbed depends not only upon the temperature, but also upon the percentages of manganese, silicon and phosphorus which the iron contains; for example, high temperature and high-manganese content increase the carbon content of the iron.

Briefly, the factors affecting the total carbon in pig iron are as follows:

1. *Hearth temperature.* At 2100°F (1150°C), under favorable conditions, the purest iron will absorb 4.25 per cent carbon, and this carbon content is increased by about 0.15 per cent for each increase of 100°F (55°C). Hearth temperatures, however, are adjusted to other more important factors, such as the control of silicon and sulphur. In considering the effect of temperature upon molten iron, it is necessary to keep in mind that the effective temperatures are inevitably above a certain minimum of about 2085°F (1140°C), marking the freezing point of the iron-carbon eutectic (see discussion of the iron-carbon equilibrium diagram in Chapter 42).
2. *Rate of cooling immediately after tapping.* Control of the cooling rate of the liquid iron gives the only chance of controlling the total carbon. If the cooling prior to solidification is slow, carbon in excess of the eutectic composition separates as graphite, which rises to help form the mixture of graphite and slag known as kish. Such control of carbon is restricted by other factors.
3. *Composition of the iron.* The proportion of carbon in the eutectic alloy and the total amount of carbon the iron will absorb at a given temperature are affected by the presence of other elements, the individual effects of different elements being about as follows:
 - (a) *Silicon* reduces progressively the carbon in the eutectic from 4.25 per cent to about 3.50 per cent with 3.0 per cent silicon present, but as a high temperature (at which carbon solubility is increased) is necessary to obtain the higher silicon, the total carbon may not be much affected, the net result being rather to decrease the combined carbon and increase the graphitic. Since low silicon is obtained at low temperature, lowering the silicon will neither raise nor lower the carbon much from about 4.25 per cent.
 - (b) *Phosphorus* decreases the total carbon the iron will absorb, one per cent causing a drop of about 0.3 per cent in the carbon held at any temperature above 2200°F (1205°C). It probably lowers the melting point of the eutectic, hence increases the loss of carbon as kish when the iron is held a time in the liquid state.
 - (c) *Manganese*, also *chromium*, *titanium*, *vanadium*, *tungsten*, and *columbium* are carbide-forming elements and tend to increase the total carbon and to hold more of it in combined form on cooling. The effect of manganese in increasing the carbon is much less than the

effect of silicon in decreasing the carbon, one per cent manganese increasing the total carbon by about 0.04 per cent, and one per cent silicon decreasing it by about 0.3 per cent. Thus, an increase of 7 to 9 per cent manganese would be required to neutralize the effect of a one per cent increase in the silicon content.

- (d) **Sulphur**, in amounts usually occurring in pig iron (0.03 to 0.05 per cent), even up to 0.2 per cent, has little effect upon the total carbon. It does, however, show a marked effect in opposing the formation of graphite, so that with silicon constant, say at one per cent, a chill test reveals any increase in sulphur by increasing the depth of chill.

Silicon—Silicon enters the furnace as silica (SiO_2), some of which may be combined with bases as silicates. At temperatures of about 2192°F (1200°C), corresponding to the fusion zone in the blast furnace, the greater portion of this silica combines with lime, CaO , and other bases to form silicates. However, at a high temperature, such as exists in the hearth of the furnace and in the presence of carbon, silica is reduced, and the resultant silicon combines with the iron to form iron silicide, as represented by the following two equations:



The ordinary amounts of silicon dissolve in the iron and formation of the silicide is theoretical. Besides the effects of silicon upon the carbon given above, other uses and characteristics are listed as follows:

1. In steelmaking processes, it is oxidized to SiO_2 and enters the slag. This reaction is important in the acid Bessemer process (and to a less degree in other processes) because it is an essential source of heat.
2. In the blast furnace, the amount reduced, though difficult to control closely, depends upon the slag composition and the temperature. Below 2700°F silicon may absorb oxygen from carbon monoxide (CO) and its reduction absorbs much heat; hence, high temperature and the presence of iron with which it can combine instantly are essential. Its reduction, therefore, is possible only below the melting zone.

Manganese—Coming now to the elements that are introduced into the pig iron through reduction processes occurring in the furnace, manganese is the one most closely resembling iron in its chemical conduct. While traces of the element are found in both the limestone and the coke ash, practically all that entering the furnace is present as an ingredient of the ore or in materials such as open-hearth slag or converter slag, in which it occurs as oxide, silicate or phosphate. The manganese compounds are never wholly reduced in the furnace, and the proportion of them that is so reduced depends upon the conditions of the temperature and basicity of the slag. Under conditions ordinarily prevailing in the blast furnace producing pig iron, from 50 to 75 per cent of the total amount of this element charged will be found in the pig iron, the highest proportion being obtained with high hearth temperatures and basic slags.

Manganese alloys with iron in all proportions. An alloy containing 10 to 25 per cent manganese is called *spiegel*, or *spiegeleisen*. Alloys containing 40 to 80 per cent manganese are called *ferromanganese*.

Manganese decreases the effects of sulphur, with

which it combines, forming MnS and replacing iron in the sulphide FeS . If a high-manganese iron is kept molten for some time, some of the sulphur it contains will react with manganese in the iron to form manganese sulphide, which will rise to the surface, like slag, and react with the oxygen of air, liberating sulphur as oxides. With high-manganese ores, this fact can be utilized to reduce the sulphur content of the pig iron, but in order thus to obtain a material reduction of the sulphur content the manganese content of the pig iron should exceed 2 per cent. For these reasons, a high content of manganese is desirable in some irons for the foundry and in all irons in open-hearth steelmaking purposes. Manganese is less desirable in iron to be utilized in the Bessemer process in which it is oxidized and unites with silica to form a slag that fuses at a comparatively low temperature and is very fluid, so that iron containing more than 0.5 per cent manganese gives rise to a condition known as a "sloppy" heat when blowing iron in the Bessemer converter. High manganese is also undesirable in certain grades of foundry iron. Hence, the manufacture of merchant iron requires that manganese, as well as silicon, sulphur, and phosphorus, be controlled.

Phosphorus—Phosphorus is the only element entering the blast furnace over which the skill of the furnaceman has practically no control, except by proper selection of raw materials. Its compounds are completely reduced, so that essentially all the phosphorus in the raw materials is found in the metal. Phosphorus enters the furnace with the charge in the form of phosphates. At very high temperatures and in the presence of carbon, these compounds are reduced, as shown in Reaction (14). Phosphorus may react with iron to form iron phosphide, thus:



This phosphide, however, is soluble in iron and becomes a part of the metallic bath in the blast furnace.

Disposition of Sulphur—Sulphur is carried into the furnace mainly by the coke, though 10 to 20 per cent of the total is often found in the ore and the limestone. The sulphur contained in the coke enters in the form of elementary sulphur and FeS , all of which, when melted, alloy with the iron in the furnace; a smaller portion, in the form of sulphates, such as CaSO_4 , enters as an impurity in the ore, limestone, and coke, and is reduced to sulphide in the presence of carbon. At a very high temperature and in the presence of carbon and a basic slag (wherein the ratio of $\text{CaO} + \text{MgO}$ to $\text{SiO}_2 + \text{Al}_2\text{O}_3$ is greater than 1), a reaction takes place which is usually represented as follows:

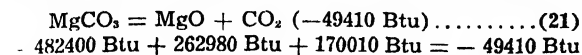
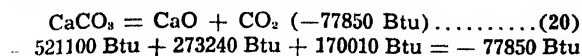


This reaction, being influenced by other conditions as well as those just mentioned, never goes to completion, so a small portion of the sulphur remains in combination with the iron. This iron sulphide is soluble in iron, hence, becomes a part of the metal.

The blast furnace provides the only economically positive means for the control of sulphur. Removal of sulphur in any of the steelmaking processes is either uncertain or expensive, or both. For this reason, pig iron containing less than 0.05 per cent sulphur is preferable for making steel by all the fusion processes.

Action of Calcium and Magnesium—Calcium and magnesium enter the furnace mostly as carbonates. Small portions may be in the form of silicates, in which CaO and MgO are combined with SiO_2 , and may undergo no chemical change in the furnace. The carbonates, however, are slowly decomposed at temperatures above

1470°F (800°C), liberating CO₂, with the reaction becoming rapid at 1740°F (950°C).



At the proper temperature for their formation the caustic lime and magnesia, in intimate contact with SiO₂, both combine with it to form slags.

Action of Aluminum—Aluminum, in the form of alumina (Al₂O₃) and aluminum silicates, are found in ore, flux, and fuel. Neither alumina nor aluminum silicates are reduced under the conditions that prevail in a blast furnace. Alumina may exert a marked influence upon the fluidity and fusibility of the slag, and increases the desulphurizing power of the slag. Alumina in slag is neither definitely acid nor basic. Investigating the question as to whether or not alumina is an acid or a base, runs of one week each were made in which the alumina in the slag was varied from 11 per cent to 21 per cent, approximately, to produce the same grade of pig iron. The data were then collected and compared as shown in the tabulation below. It will be noticed that magnesia has been omitted. This omission was made because the magnesia in the slag was fairly constant in all cases, ranging between 2.1 and 2.4 per cent.

Furnace	1	2	3	4	5	6
	Al ₂ O ₃	SiO ₂	CaO	CaO	CaO + Al ₂ O ₃	CaO
	%	%	%	Al ₂ O ₃ + SiO ₂	SiO ₂	SiO ₂
A	11.5	34.5	51.5	1.20	1.82	1.49
A	18.6	31.2	47.9	0.96	2.13	1.53
G	20.1	31.0	46.5	0.91	2.15	1.50
F	20.8	30.4	46.5	0.91	2.21	1.53

Columns 4, 5, and 6 give the acid-to-base ratios with the alumina considered respectively as an acid, a base, and a neutral or amphoteric component respectively. It is at once noticed that with alumina omitted, the ratios are almost constant as they should be to produce iron of the same kind, grade, and quality. The ratios in columns 4 and 5 show a variation not to be expected under the conditions of the experiment. A comparison of columns 1, 2, and 3 indicates alumina may act as an amphoteric component. Many furnaces are running with higher magnesia (about 5 per cent) in the slag than that indicated above and a ratio of lime-plus-magnesia to silica of from 1.35 to 1.40 is considered as producing an ideal slag.

Action of Less Abundant Elements—Titanium, potassium, sodium, zinc, arsenic, barium, beryllium, boron, copper, chromium, nickel, selenium, tellurium, tin, zirconium, and vanadium are some other elements which are present in very small, but varying, amounts in the raw materials going to the blast furnace.

Titanium enters the furnace as titania, TiO₂, combined with some base, or as ilmenite (FeTiO₃ or FeO·TiO₂). When titania comprises less than about two per cent of the ore burden, 50 to 60 per cent of the titania will appear in the slag as TiO₂, and the balance will be reduced and appear in the iron. When TiO₂ in the slag exceeds

about 1.5 per cent, slags may be very viscous and result in irregular furnace operation. Also, the character of the hot metal may be adversely affected, resulting in excessive skulling of hot-metal ladles. Under the conditions prevailing in the furnace, titanium exhibits a slight tendency to combine with carbon and nitrogen to form titanium cyano-nitride. This substance is sometimes found in the salamander on the hearths of furnaces being repaired. Here, it occurs in the form of small cubes that have the appearance of copper.

Beryllium and **zirconium** are similar to titanium, but much more difficult to reduce, so that they probably pass through unchanged.

The **alkalies**, soda and potash, are found in nearly all blast-furnace slags; when they are present in the raw materials to a considerable extent, they may, however, flux the furnace lining, and in part be volatilized to escape with the gases from which they later may condense to form troublesome accretions (scaffolds) on the furnace lining. They may play a part in the furnace reactions, for it is known that Na₂CO₃ and K₂CO₃ accelerate somewhat the reduction of Fe₂O₃ at 1650°F (900°C).

Zinc is a very troublesome element when present in blast-furnace material. Its compounds may be reduced in the lower regions of the stack; and, if so, the zinc is volatilized, driven upward and oxidizes to zinc oxide, which condenses on the walls of the colder part of the flues and in time closes up small passages to such an extent that the flow of the gases is restricted seriously. Zinc oxide also tends to combine with the alumina in the firebrick lining of the furnace, causing the brick to expand with damaging results.

Arsenic acts very much like phosphorus. Its compounds are reduced, and the resultant elementary arsenic then combines with iron to form iron arsenide, which dissolves in the metal.

Barium is similar to calcium, acting as a nonreducible basic flux; in an ore, it may be present as the carbonate, silicate, or sulphate. BaO is a base similar to CaO, but of higher molecular weight, hence does not increase basicity on a percentage basis, nor the desulphurizing power, as rapidly as CaO by almost one-third. On a molar basis its effect is similar to CaO.

The compounds of **boron** presumably behave as a flux.

Copper compounds are readily reduced yielding metallic copper, which alloys with the iron.

Chromium is reduced from its oxides with difficulty in the blast furnace, an exceedingly high temperature and an acid slag being most favorable for the reduction of its oxides. Small amounts appear to be more nearly completely reduced than large amounts.

Compounds of **nickel** and **vanadium** are almost completely reduced, the resulting nickel and vanadium alloying with the iron.

Tin is rare in raw materials of the United States; if introduced, its compounds are reduced and the tin alloys with the iron.

Selenium and **tellurium**, though somewhat rare, may be present in some raw materials; in their reactions, they are somewhat similar to sulphur but with an even greater tendency to remain with the metal.

Reactions Within the Furnace—With all the foregoing facts concerning the properties of the various ingredients of the raw materials in mind, the changes that take place in the blast furnace are understood more easily. The accompanying chart (Figure 12—25) gives a graphic representation of these changes, showing the relative weights and volumes of materials, the reactions and the temperatures at which the changes take place, and the

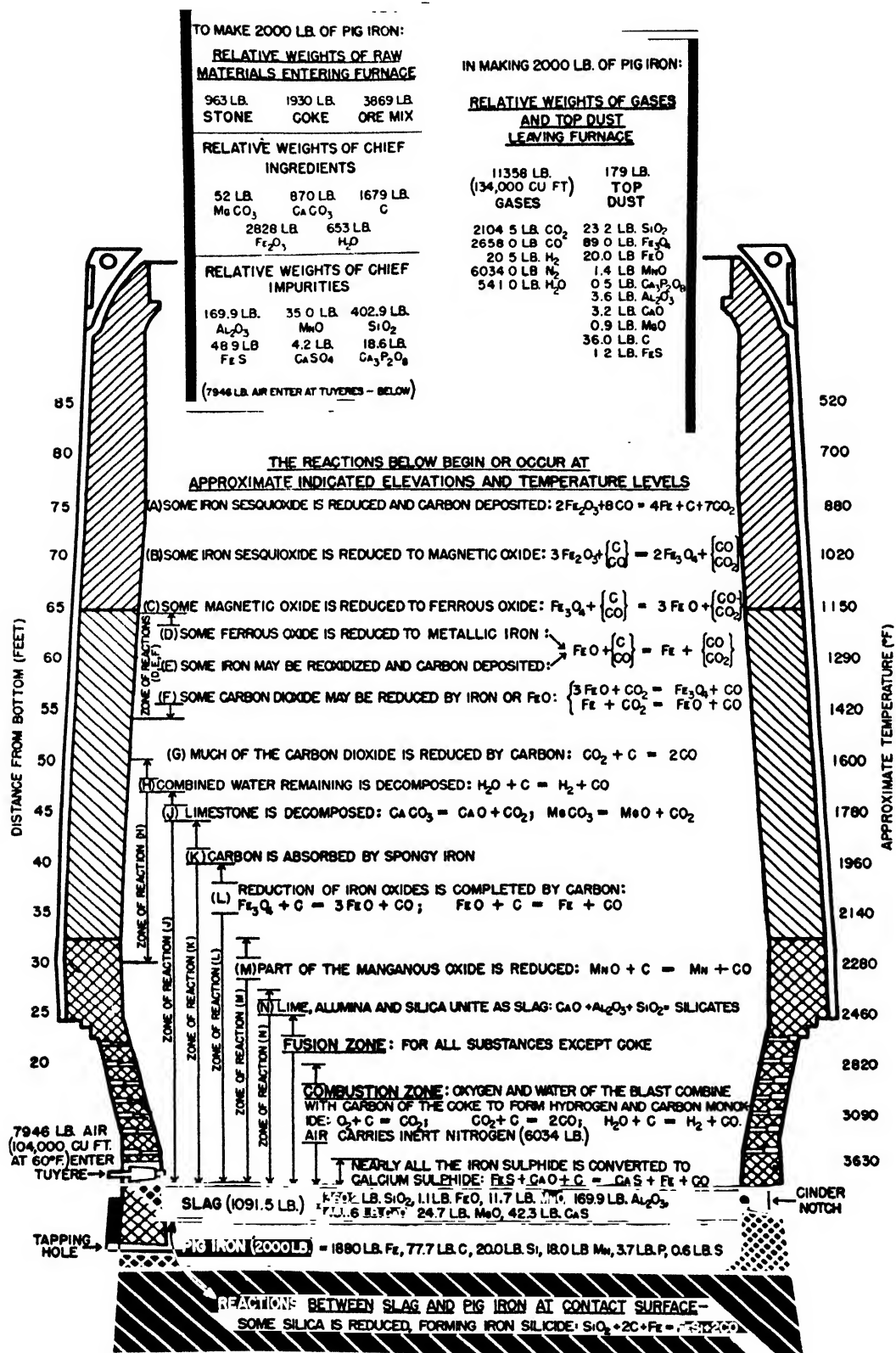


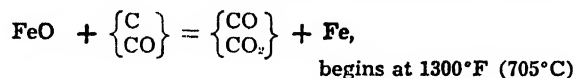
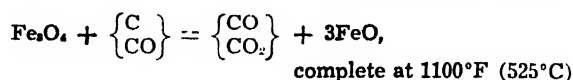
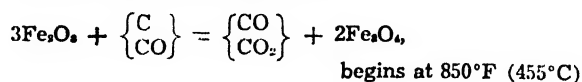
FIG. 12-25. The making of a ton of pig iron, indicating the raw materials used, the products, and their relative weights, and the reactions that take place at various levels in the blast furnace. NOTE: Shape of the furnace has been distorted greatly to accommodate notations.

final disposition of the products. In studying this chart, however, one important fact should be kept in mind. It is this: Owing to the conditions prevailing within the furnace, very few, if any, of the reactions will be complete, that is, use up all the material at hand at the location indicated. Thus, the first reaction, showing the reduction of the ore to metallic iron with the deposition of carbon, affects only a part of the ore and gas. This condition, with but one exception, holds for all these reactions. Even limestone, which will decompose completely into lime and carbon dioxide at 1800°F if given sufficient time, will often reach the tuyeres in some small part as calcium carbonate. The fact that iron and manganese oxides are not completely reduced is established by their presence in the slag. The one exception to this rule is phosphorus. Its compounds, down to small traces, seemingly are reduced completely.

The explanation for these statements is to be found only in a careful study of chemical laws in connection with the conditions prevailing in the furnace. Such a study reveals the fact that the reactions in the upper part of the stack of the furnace are subject to conflicting tendencies. Thus, the gaseous substances, CO and CO₂, are in constant contact with the solid substances, Fe₂O₃, FeO, and C. Of these, Fe₂O₃ and CO₂ are oxidizing agents. FeO and CO may act as either oxidizing or reducing agents, while C and Fe are reducing agents. With these substances in contact at any given temperature and in any given proportions, the reversible reactions would tend to proceed in a given direction until equilibrium should be established, and no further change would occur until either the concentration of one of the reacting substances or the temperature changes. Then the reactions would once more proceed in a direction that would again establish equilibrium. But the slow downward movement of the stock to regions of higher and higher temperatures, the presence of an excess of carbon and the rapid upward flow of the gases, which has the effect of giving a constant surplus of CO and of carrying CO₂ out of the field of action, tend to prevent the establishment of equilibrium, and to force the reactions to proceed in a direction that results in the final reduction of the iron oxides, with the consequent oxidation of either the C or the CO. Since the desired reactions are between solids and gases, the physical condition of the ore, especially its porosity and its particle size, play a part in either accelerating or retarding the reducing reactions. These same conditions, however, which tend to reduce the oxides of iron, prevent the complete oxidation of the CO to CO₂, because the CO, passing so rapidly over the stock—3 to 5 seconds to travel the height of the furnace in full blast—does not have time to become wholly oxidized, and the presence of the reducing agents, Fe, FeO, and C, tends to oppose the formation of CO₂. The escaping top gases, necessarily therefore, always show a large content of CO, much in excess of the CO₂ content. In modern furnaces, operating according to the prevailing practice with respect to coke consumption, the relative volumes of CO and CO₂ are approximately 2 to 1. In the lower part of the stack, the temperature is so high that CO₂ can exist but transiently, if at all, in the presence of carbon, and any oxide reduced in this region results in the gasification of a proportionate amount of carbon. This direct reduction of oxide by carbon is the most thermally inefficient mode of reduction, because it absorbs much heat, as shown by Reactions (8), (9), and (10), and robs the tuyere zones of carbon needed for combustion.

Tracing the Materials Through the Furnace—The ore, limestone, and coke, upon being charged into the top of the furnace, come in contact with an ascending current

of hot gases at a temperature of about 300°F (150°C). The first change that takes place is the physical one of drying, the hygroscopic water being driven off and carried out of the furnace by these gases. (Water of crystallization or chemically combined water is driven off from the various compounds in which it occurs in the range of temperature between 1300° and 1475° F). The stock, with its interstitial spaces filled with an ascending atmosphere containing the reducing gas CO, descends to regions of higher and higher temperatures. At different levels, then, chemical reactions peculiar to the temperatures of these levels occur. At first only the oxides of iron and carbon suffer change, and one of the first reactions to occur is number (4), in which carbon deposition may begin at a temperature as low as 575°F. In the presence of both C and CO, a large part of the remaining iron oxide, of small particle size, is next reduced at successive levels and at temperatures perhaps somewhat as follows:



At about 1457°F (800°C), the free iron is subject to reoxidation by CO₂, as are also the compounds, FeO and Fe₃O₄, though to a less degree, the chief action being represented by Reaction (5). These reactions and temperatures apply only to very fine ore. Lump ore, especially if it is not very porous, will descend to much higher temperatures before it is acted upon, the large, dense lumps may descend even to a level below the top of the bosh.

At 1475°F (800°C), or a little above, the decomposition of limestone takes place, thus: $\text{CaCO}_3 = \text{CaO} + \text{CO}_2$. This reaction is complete at 1830°F (1000°C). At 1650°F (900°C), carbon reduces CO₂ to CO, thus: $\text{C} + \text{CO}_2 = 2\text{CO}$, so that CO₂ does not exist normally below the 60-foot level. From this level the mixture is one of gangue, quicklime, coke, spongy iron, and varying amounts of unreduced ore, all of which descend to the fusion zone with very little change, if the absorption of carbon by the iron and the action of carbon on the unreduced ore be excepted. At this level, which is located at the top of the bosh, the lime combines with some of the gangue and, with a little unreduced iron oxide and manganese oxide, forms a part of the slag. The slag, such as already is formed, and the iron, both now in the liquid state, trickle down through the interstices of the coke to the hearth. In the hearth zone just above the tuyeres, where the metal and slag are in contact with hot coke, Reactions (11), (12), (13), (14), (17), (18) and (19), known as hearth reactions, occur. In the crucible, the slag and metal become separated by gravity, forming these two layers; a lower or metallic layer containing all reduced substances and an upper or slag layer containing all unreduced matter.

Conditions Affecting Silicon and Sulphur in the Metal—Reactions (13) and (19) should receive special attention, because of all the reactions that directly affect the quality of pig iron, these are the only ones the furnaceman can control. Both reactions depend upon two conditions, namely, temperature and basicity of slag. High temperatures favor the reaction, $\text{SiO}_2 + 2\text{C} = \text{Si} + 2\text{CO}$,

while a more highly basic slag retards it. Reaction (19), $\text{FeS} + \text{CaO} + \text{C} = \text{CaS} + \text{Fe} + \text{CO}$, requires lime and a high temperature. Therefore, high hearth temperature tends to raise the silicon content of the iron and lower the sulphur, provided the slag volume is adequate and the high temperature is obtained without the use of an excessive amount of high-sulphur coke. High basicity tends to lessen both the silicon and sulphur in the iron. In both instances, the extent to which the reactions progress under any given set of conditions is governed by time. The longer the time the farther they progress. This fact results in a difference in composition between

the first and the last metal in the same cast. Since the iron on the bottom of the furnace crucible is formed four or five hours before that on the top of the layer at the time of tapping, these reactions will tend to advance farther in the first than in the last iron formed under normal conditions. The first of the cast, therefore, usually will be found to contain a higher percentage of silicon and a lower percentage of sulphur than the last. This first iron is called **hot iron** on this account, because, as will be shown later, the oxidation of the silicon generates heat when the pig iron is purified, as in the Bessemer process, in the making of steel.

SECTION 7

THE BLAST-FURNACE BURDEN

The amounts of ore and stone charged per ton (or other fixed quantity) of fuel are referred to as the **burden**, the fuel or coke being constant in amount. Any increase in ore and stone above the normal is spoken of as a **heavy burden**, while the reverse of this results in a **light burden**. The regulation of the proportions of ore, flux, and coke is called **burdening**. It has two objects; namely, the most efficient operation of the furnace and at the same time the production of metal of the desired composition. The subject is of the greatest importance in the operation of a furnace, and is a

problem that may be solved either by practical experience or by calculations based on theoretical considerations.

Burdening the Furnace—With a furnace well started and on familiar materials, practical knowledge and experience alone may be sufficient to operate successfully. But in dealing with changing and unknown materials, it is necessary to resort to computed or "theoretical" burdening based on chemical composition. As illustrating the problems that confront the blast-furnace operator, the following will supply concrete examples:

Table 12—III. Representative Compositions of Raw Materials Used in the Blast Furnace (Dry Basis)

Elements and Radicals	Formula	Ores				Lime-stone	Coke
		(From Lake Superior District)					
		1	2	3	4		
		(%)	(%)	(%)	(%)		
Silica	SiO ₂	6.48	9.04	12.78	5.86	3.43	4.40
Iron	Fe	56.80	54.84	52.93	55.59	0.30	1.35
Manganese	Mn	1.14	1.09	0.83	0.16	0.08	0.07
Phosphorus	P	0.082	0.096	0.083	0.618	0.006	0.030
Alumina	Al ₂ O ₃	3.22	2.70	3.34	3.63	0.86	2.80
Lime	CaO	0.11	0.18	0.25	1.22	51.45	0.25
Magnesia	MgO	0.14	0.26	0.21	0.87	1.66	0.15
Fixed Carbon	C	90.00
Carbon Dioxide	CO ₂	41.43
Sulphur	S	0.940
Sulphuric Anhydride	SO ₃	0.03	0.04	0.04	0.06	0.060	Undetermined
Alkalies	$\left\{ \begin{array}{l} \text{Na}_2\text{O} \\ \text{K}_2\text{O} \end{array} \right\}$	Trace	Trace	Trace	Trace	Trace	Trace
Titania	TiO ₂	0.018	0.009	0.012	0.019	Trace	Trace
Moisture (Wet Basis)	H ₂ O	14.06	14.50	15.00	12.10	0.50	1.50

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Chapter 13

EARLY PROCESSES FOR CONVERSION OF IRON INTO STEEL

SECTION 1

PROBABLE ANCIENT METHODS

It is probable that most of the steel produced in ancient times consisted of partially case-hardened wrought iron. Wrought iron, when heated and suddenly cooled by quenching in some liquid, will not harden because it does not contain enough carbon. By increasing the carbon content in the manner to be described, the outer portions and edges of a piece of wrought iron could be made to harden by heating and quenching.

The hardenable outer portion or "case" was produced by allowing a wrought-iron object to remain in a forge fire in contact with hot carbon, which completely surrounded it and protected it from oxidation. The carbon absorbed by the surface "layers" of the wrought iron would make it possible to obtain very hard surfaces and edges on a weapon, for example, by quenching the heated metal in a suitable liquid. The interior of the tool or weapon, of course, would remain relatively soft. Undoubtedly some steel was produced, intentionally or otherwise, when an imperfectly decarburized product was obtained from the South Wales, Lancashire, Wallow or similar processes. Another method, used in some of the Eastern countries to produce what very commonly is known as "Damascus steel," consisted of piling alternately pieces of soft iron on pieces of high-carbon iron and then heating, forging, fagoting and re-forging the billets. The layering that resulted from incomplete diffusion of carbon from the high-carbon bars into the low-carbon iron resulted in the surface appearance called "watering," which was characteristic of the so-called Damascus steels.

It is difficult to say how or when the first processes for intentional making of steel were developed. Archeological specimens from as far back as 1000 B.C. exhibit evidences of having been deliberately treated (case-hardened) to produce points and edges that were hardened by rapidly quenching the heated steel. One example is a chisel with a hardened point, found in one of the ancient cities of Ceylon and believed to date back to about 500 B.C. Early writers mention steel razors, surgical instruments, files, chisels and stone-cutting implements as early as several hundred years before the Christian Era.

The steel called *wootz* was produced in India for many centuries. Its method of manufacture has been variously

described. It is generally agreed that the first step consisted of heating pure ore with carbonaceous material such as charcoal or finely-chopped wood in closed crucibles. After heating at a high temperature for several hours, the ore was reduced to metallic iron and absorbed sufficient carbon from the excess of charcoal to have a low enough melting point to become fluid. The crucibles were allowed to cool and, when broken open, a small "button" of high-carbon steel was found at the bottom. Two methods have been recorded for lowering the carbon content of the buttons to give steel having the desired carbon content or "temper." One method consisted of repeatedly heating the buttons while they were covered with a layer of iron-oxide paste. The other recorded method comprised heating the buttons for several hours in a charcoal fire to a temperature not much below their melting point and turning them over in the path of the blast, so that the metal would be partially decarburized. In both cases, the partially decarburized buttons would then be heated to be welded together by hammering to form bars. Ambiguous records of various other processes for making steel by carburizing wrought iron appear in fragmentary literature from very early times.

In summary then, it may be said that prior to the invention of the Bessemer process for steelmaking in 1856 there were only two methods of making steel. One was the process of increasing the carbon content of wrought iron by heating it in contact with hot carbon away from air; this came to be called the **cementation process**. The other method, the **crucible process**, consisted of melting wrought iron in clay crucibles in which carbon had been added for the express purpose of increasing the carbon content of the iron. Both of these processes were certainly known to and practiced by the ancients.

During the Middle Ages both the cementation and crucible processes appear to have been lost to civilization. The cementation process was revived in Belgium about the year 1600 A.D. while the crucible process was rediscovered in England by Benjamin Huntsman in 1742. Both processes were practiced in secret for some time after their revival. Hence, little is known of their early history. The following brief descriptions will, therefore, be confined to practices in later years.

SECTION 2

THE CEMENTATION PROCESS

The cementation process was highly developed and flourished in England during the 18th and 19th centuries and, though it has practically been replaced by other methods, is still practiced to a limited extent. The

process depends upon the fact that when a low-carbon ferrous product, such as wrought iron, is heated to a red heat in contact with charcoal or other carbonaceous material, the metal absorbs carbon, which, up to the satura-

tion point of less than 2.00 per cent (about 1.70 per cent), varies in amount according to the time the metal is in contact with the carbon and the temperature at which the process is conducted. For carrying on the process, a type of muffle-furnace or pot-furnace is used, and the iron and charcoal are packed in the pots in alternate layers.

The iron used was usually in the form of bars 2½ inches to 3 inches wide, ⅝ inch to ¾ inch thick, and 6 to 12 feet long. For the best grades of steel, only the best wrought iron was supposed to be used, though low-carbon steel made by the open-hearth process was later substituted. Charcoal, which had been passed over about a ¼-inch screen to remove fines, represented the favorite carburizing agent, though various other substances and mixtures had been tried and used. In charging the pots, their bottoms were first covered with a layer of charcoal 2 or 3 inches thick, then, in alternation, layers of bars and charcoal were added to each until the pots were full. The bars were laid flatwise and about ½ inch apart, between edges, so that each bar was completely surrounded by charcoal. After applying the final layer of charcoal the charge was covered with wheel swarf (refuse from the grindstones) and all openings to the pots were closed and made nearly as airtight as possible. A fire was next lighted in the furnace, and the charge, for the next 3 or 4 days, gradually heated to a full red heat—the actual temperature varied from 1470° F to 2010° F (800° C to 1100° C) and this temperature was then maintained for 7 to 12 days, depending upon the size of the bars used, the carbon content desired, and the temperatures attained and maintained. The degree of carburization, or the temper of the bars, was determined by fracture tests on test bars that were withdrawn from the pots through small openings or holes provided for the purpose. When the bars had reached the desired temper, i.e., had absorbed the desired amount of carbon, the fire was banked and the furnace allowed to cool slowly. When the contents of the furnace were cool enough to handle, the bars were removed from the pots.

If the original bars were of wrought iron, their surfaces

were found to be covered with irregular elevations, known as blisters or beads, which resulted from the expansive force of the carbon monoxide formed by carbon reacting with the oxides of the incorporated slag. Hence, these bars were known as blister steel. If the original bars were of mild steel or remelted wrought iron, these blisters were absent. Both products were frequently referred to as converted or cement steel. If air had gained access to the bars during the process, their surfaces were covered with scale and almost decarburized. Such bars were known as aired bars. After sorting and reheating, the cement bars were hammered or rolled into what in England was called spring plate or bar steel, which formerly was used for springs, but later was used as raw material for crucible steel or for the production of shear steel. If for the latter, the bars were broken or sheared into short lengths, fagoted, sprinkled with a little borax, covered with clay, reheated to a welding temperature, and hammered into a bar, known as single shear. For purposes requiring a high grade, uniform steel, the single shear bars were again broken at their centers, the two halves of each laid together, and the fagot thus formed was reheated and hammered down to the required size. These bars were known as double shear steel. This steel was formerly used exclusively for the manufacture of cutlery, hence the name, shear steel.

This working was necessary to obtain the highly desirable homogeneity of the steel, and, prior to the revival of the crucible process, was the only method available to attain that end. Formerly, about seven grades or tempers of shear steel were produced. These varied in average carbon content from 0.50 per cent to 1.50 per cent. But as the carbon was absorbed from the surfaces of the bars, the carbon content of the blister steel bars progressively decreased to the center. In the softer grades the center portions of each bar remained unaltered, and this core was known as sap. In the harder grades, the outside of the bar might show a carbon content of 1.50 to 2.00 per cent with a center of 0.85 to 1.10 per cent. It was this characteristic of cement steel that led Huntsman, who was a clockmaker, to seek a method for making a more uniform steel for his springs.

SECTION 3

THE CRUCIBLE PROCESS

Realizing that the shortcomings of cement steel for springs lay in lack of homogeneity, Huntsman conceived the idea of melting it in crucibles, which melting he thought should make the steel perfectly homogeneous. Briefly, his method was as follows: First, cement bars were carefully selected that would give the exact temper, or carbon content, desired in the finished steel. These were cut or broken into small pieces. Then this steel was charged into large clay crucibles, which, after covering and luting the lids, were placed in a coke fire and heated until the contents were thoroughly fused. At this point the crucibles were withdrawn from the fire, and upon removing the cover and skimming off the small amount of slag formed on top, the molten metal was poured into a cast-iron mold, where it remained until solid. Finally, by the usual method of reheating and hammering then in use, the ingot was worked into the form desired. This method gave a steel that was not only homogeneous throughout, but was free from occluded slag and dirt, hence was so much superior to cement steel for many purposes that the method at once became the leader for the production of the finest steels. This position the process held for almost two hundred years, but today, at least in this country, it has been

superseded for the making of special alloy steels and carbon tool steels by the electric-furnace process, including the high-frequency induction furnace. The electric-furnace process is cheaper, is capable of giving as good steel, and possesses many metallurgical advantages over the crucible method.

Although the principles and the general method of procedure remained the same, the crucible process was the subject of considerable experimentation after the time of Huntsman, and some changes in the material of the charge, in the manufacture of the crucibles, and in the furnace for melting the crucible charges were introduced, so that the details of standard practice for this now practically extinct process in different countries and localities varied somewhat.

Manufacture of the Crucibles—The crucibles, or pots, were a very important part of the equipment required in the manufacture of crucible steel. As they were costly to make and had a comparatively short life, they were a large item of expense. In England, they were frequently made of clay by the steel manufacturer himself, but in this country so-called graphite crucibles were almost universally used. These crucibles, with a capacity of 80 to 124 pounds, varied somewhat in form and thick-

ness of wall. In general, they were barrel-like in shape, and varied in size from 13 to 18 inches in height and from 8 to 12½ inches in outside diameter at the bilge. As to wall thickness, they were somewhat thicker at the bottom than at the top, usually about 1½ inches at the bottom and about ¾ inch at the top.

The Crucible Melting Furnace—In this country, the crucible furnace was generally of the gas-fired regenerative type (Figure 13-1). It consisted of a number

CRUCIBLE

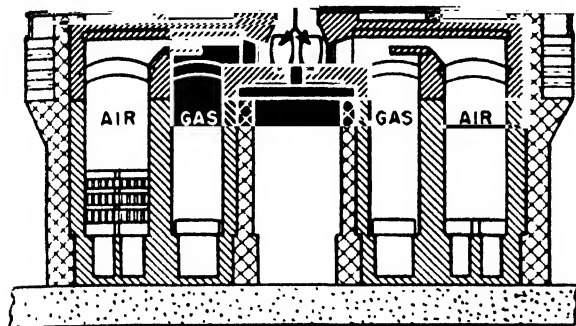


FIG. 13-1. Vertical section through checker chambers and one of the melting holes of a regenerative crucible furnace

of combustion chambers or melting holes built in a row between two sets of checker chambers. Each melting hole, about three feet in depth and otherwise large enough to hold 4 or 6 crucibles, was connected with the two sets of checkers by short flues leading upward from its bottom. If producer gas was used as fuel, the gas checkers were placed next to the line of melting holes with the air checkers extending along the outer walls of the furnace. This plan of construction readily permitted introducing the gas into the ports below the air, which was deflected downward by the roof, thus holding the flame near the bottom of the crucibles as it swept across the melting hole. In case natural gas, fuel oil, or powdered coal was used for fuel, both checkers would be used for air, or the furnace would be constructed with but one checker chamber on each side. The shop floor was on a level with the top of the melting hole, so that only the covers were visible from above. After moving these covers aside, the crucibles were lowered into and lifted out of the hole by tongs with broad jaws curved to fit the crucibles.

The bottoms of the melting holes were built of first-quality firebrick laid on plates, which were supported by beams, or cross bars, spanning the space between the two sets of checkers, thus forming a continuous cellar, or vault, that extended the full length of the furnace beneath the melting holes. A hole, some 4 or 5 inches in diameter, was provided in the center of the bottom of each melting hole, so that any refuse on the bottom of the melting hole could be poked into the vault below for cleaning. To protect the bottom in case of breakouts, it was covered to a depth of 6 to 8 inches with coke dust or crushed coke. This coke also gave some protection to the crucibles, as it helped to consume free oxygen not needed for combustion and to maintain a reducing atmosphere in the hole.

Charging the Crucibles—From what has already been said, it will be evident that, aside from the possible elimination of gases from the steel and the separation of slag and other non-metallic inclusions, the crucible process was not a purification process. Consequently,

any elements capable of alloying with iron charged into the crucible were present also in the steel. On this account the greatest of care in the selection of the raw materials had to be exercised. And it was characteristic of the history of the process that, once a charge that produced the kind or grade of steel desired had been hit upon by any shop, that charge was strictly adhered to and could not be varied. For many years after the introduction of the process by Huntsman, cement steel or blister steel was the only raw material used. Eventually, attempts were made to eliminate the cementation step by the use of wrought iron with charcoal; and, provided Swedish iron, which was the purest wrought iron then made, was used, the steel produced was as good as that made from cement bars. In 1801 David Mushet introduced the use of manganese, by adding oxide of manganese with the charge, which practice was superseded some years later by the use of *spiegel*. It is interesting to note that the use of *spiegel* in Bessemer steel was patented in 1857 by a namesake, Robert Mushet (this method of adding manganese was an important factor in the early development of Bessemer steel). Then followed a great deal of unsuccessful experimental work, involving so many different substances and mixtures that the stock house came to be known as the medicine house. Later, with the introduction of the Bessemer and acid open-hearth processes, efforts were made to reduce the cost by the substitution of soft-steel scrap for the Swedish wrought iron, but owing, perhaps, to the fact that miscellaneous scrap was used, which carried a higher content of manganese, phosphorus, and sulphur and was more variable in composition than the Swedish wrought iron, the steel was found to be of inferior quality. More recently, washed metal, that is semi-purified pig iron, freed from silicon, manganese, and phosphorus, had been used with excellent results to bring up the carbon of the wrought iron. Still more recently, owing to the introduction of the cheaper electric process and the difficulty of securing good wrought iron, recourse was had to the use of steel scrap, in part at least. With good basic or acid open-hearth mill scrap, the composition of which is more uniform than miscellaneous scrap and could be more readily controlled, steel of good quality could be made, provided proper care was taken to adjust the composition with the best wrought iron and washed metal.

All the practices just described had to do with carbon steels. Since the process had always been used in the production of the finest steels, especially tool steel, the introduction of alloy steels created the greatest diversity of practices. In making alloy steels, the old rule-of-thumb methods of charging had to be abandoned for other plans in which the composition of the charge could be accurately determined by chemical analysis. This applied particularly to high-speed tool steels, which, prior to the introduction of the electric process, could be made successfully only by the crucible process. At first, these steels were made from high-grade wrought iron or muck bar by the addition of the desired elements in the form of iron alloys, and the necessary recarburizer in the form of charcoal or washed metal. Later, tool scrap was used to make up a part of the charge, and much inspecting and many analyses were required to select the scrap and determine the proportions of the charge.

Stages of the Crucible Process—The materials for the charge, after they had been inspected and analyzed, were sheared or broken into small pieces, then carefully weighed out in proper proportions and amounts in the mixing house. These were carefully placed in the crucibles with the wrought iron or steel scrap on the bottom.

The crucibles were then tightly covered and taken to the melting hole, into which they were lowered vertically by tongs in the hands of a workman. The gas and air were then regulated to melt the charge as rapidly as possible without injuring the crucibles. The time of melting varied from 2½ to 4 hours, according to the composition of the charge and the heating conditions of the furnace. Low-carbon heats required more time than high-carbon, and high-speed steel required the most time of all grades. As the charge melted, and for some time after melting, the metal evolved gases, which finally collected in fairly large bubbles and broke slowly on the surface of the metal, producing the spectacle known as *cat's eyes*. The melter now watched each pot closely, and when the steel appeared to be dead, that is, free from gases and in a tranquil state, he signaled the puller-out, who grasped the crucible with the long broad tongs, lifted it out of the hole, and set it on the floor. The killing of the heat usually required 30 to 40 minutes, sometimes longer, and was an important part of the process, because the practice was necessary to give sound ingots. On the other hand, if the steel was kept in the furnace too long after it had reached the dead-melt stage, it would be damaged and might be ruined by absorption of too much carbon and silicon from the crucible. The temperature was also important; if the heat were pulled too cold, the steel would begin to solidify before it could be cast, and if it were cast too hot, the resulting ingot would not be sound. In some cases the pots were pulled before the steel was completely killed, and a little aluminum (not over 0.05 per cent) was added to the pot just before the steel was poured into the ingot mold. These methods of finishing affected the hardening properties of many of the steels made by this process.

Casting—The cover was then removed from the crucible and most of the slag was mopped or swabbed up by means of a ball of slag on one end of an iron rod. The crucible then was grasped horizontally with a pair of tongs, and, by tilting the crucible, the steel was poured into a cast-iron mold. This pouring required strength and skill, for it had to be continuous, and the stream of metal could not be permitted to impinge upon the walls of the mold. Any slag that might remain on the surface of the liquid metal was held back, by a small iron bar, and prevented from flowing into the mold. The molds were closed at the bottom and to facilitate the removal of the ingots after casting, split molds were used. These molds were cast in two pieces or halves, which fitted together by rings and wedges, formed a mold of square section with the joint passing longitudinally through diagonally opposite corners. Usually, the capacity of the molds was but a single potful of steel, and their cross section was but 3 or 4 inches square; but occasionally larger ingots were required, in which case two pots might be poured at once by the use of a funnel or spout made from a worn-out crucible, or several pots might be poured into a steel ladle, and the steel teemed into the molds from it. The last plan was followed in the production of fine steel castings of a size requiring more than two pots of metal. As soon as the steel had been poured, the crucible was cleaned and carefully inspected for cracks, when, if found sound and in good condition, it was charged, as before, for another heat.

Stripping and Inspecting the Ingots—As soon as the steel had solidified in the molds, the wedges or keys were loosened, and the rings holding the two parts of the mold together were removed. In the case of lower carbon steels, the mold might be removed at once and the ingot cooled in air, but ingots of high-carbon and alloy steels had to be cooled slowly to prevent the forma-

tion of clinks or tiny cracks, due to non-uniform distribution of contraction and expansion forces. Such ingots were allowed to stand in their molds until solid, then removed and placed in dry lime or covered with ashes or hot, dry sand until cold. The ingots were then topped, that is, pieces of their tops were broken off until the fracture showed perfectly sound steel. From the fractures thus exposed, the temper or carbon content (except in the cases of certain alloy steels) was determined. Then they were inspected for surface defects, and carefully and slowly reheated to a forging temperature. They were next forged, or clogged, under a hammer and any cracks or defects that appeared during the forging were ground or chipped out. After being forged to the required section, usually 1 to 2½ inches square, the billets were inspected again. Any surface defects, such as rough spots, seams, or tiny cracks were ground out with emery wheels, and these billets were reheated and rolled or forged into the sections desired. All this work was done with the greatest care, and was so meticulous as compared with the attention given ordinary steel that it was termed *crucible practice*.

Without doubt, crucible practice played as important a part in the high quality of crucible steel as crucible melting.

Chemistry of the Crucible Process—From a chemical standpoint, at least, the crucible process was the simplest of all for making steel. In the beginning, the charge carried a small quantity of iron oxides in the form of rust and scale and an almost negligible quantity of free oxygen in the air trapped in the crucible. As the charge melted, these oxides at first formed an oxidizing and very basic slag, which was soon reduced to the ferrous condition by the carbon in the charge or the crucible. Some ferrous oxide dissolved in the metal, and the remainder reacted with clay of the crucible to form ferrous silicates and ferrous aluminum silicates, which, after the charge had all melted and the temperature had been raised, absorbed more and more silica until the slag became very acid. In the meantime, the metal had been absorbing carbon, either from the crucible or the charcoal charged, and the conditions within the crucible now became strongly reducing. Any oxides dissolved or otherwise incorporated with the metal now reacted with carbon, forming carbon monoxide gas; and since the slag was very acid, silica was readily split off from the slag molecules and was at once reduced, forming silicon, which alloyed with the iron and entered the metal. The absorption of carbon and the reduction and absorption of silicon both progressed with time and temperature, and were more rapid in new crucibles than in crucibles that had been used. The presence of silicon in the metal had the effect of cleansing it of oxides, and, without doubt, it was a large factor in "killing" the steel; but after a content of 0.50 per cent was exceeded, it began to have a noticeable embrittling effect upon most steels. Silicon also affected the welding and certain other properties adversely, hence was positively undesirable in some steels, a difficulty overcome by partial substitution of aluminum. This change in the deoxidizer used sometimes was made for the purpose of controlling the hardening properties of the steel. Hence, the time of killing as well as the temperature had to be closely watched, but it is evident that close control of either of these factors was impossible, with the result that both the carbon and the silicon contents might vary over relatively wide limits. To produce a large quantity of steel of uniform composition, say up to one ton, it was necessary to pour a number of these 100-lb. crucible melts into a ladle in such a manner that the metal would be thoroughly mixed.

SECTION 4

MODERN STEELMAKING PROCESSES

All of the processes discussed in the earlier parts of this chapter were destined eventually to be supplanted by entirely new methods of steelmaking. The first of these new methods was the **pneumatic or Bessemer process**, which first found commercial application in 1860. By this process the excess amounts of carbon, manganese and silicon present in pig iron are removed by blowing ordinary air through a shallow bath of liquid pig iron. The unwanted elements are literally "burned" out of the bath by combining with oxygen in the air blast. In a typical 25-ton **converter**, as the vessel in which this process is carried out is called, this amount of liquid pig iron can be converted into steel in 12 to 15 minutes of actual blowing with air. The great advantages of the process over earlier methods are: (1) its speed, (2) the fact that the refined steel is obtained in liquid form, and (3) the amount of liquid steel produced per operation is very much greater than that produced by any earlier method. The complete history of development of this process is discussed in Chapter 14

Closely following the invention of the Bessemer process was the development of the **regenerative-type furnace**. Originally designed by Karl Wilhelm Siemens for work in steam generation, glass melting, and in heating furnaces, the regenerative furnace was capable of attaining such high temperatures that it naturally suggested itself for use in steelmaking. In this service, it became known generally as the **open-hearth furnace**. Eventually, the open-hearth furnace was destined to become the chief means for producing steel throughout the world. Its historical development is described in Chapter 15.

The **electric furnace** is relatively a newcomer to the field of steelmaking and is gradually finding more and more applications in the quantity production of quality steels. There are various ways in which the electric current can be made to generate the heat required for steelmaking operations. The development of furnaces that employ these several methods, and the design and operation of modern electric steelmaking furnaces are discussed in Chapter 16.

Chapter 14

THE PNEUMATIC STEELMAKING PROCESSES

SECTION 1

INTRODUCTORY

General Principles of Steelmaking—Pig iron consists of the element iron combined with numerous other chemical elements, the most common of which are carbon, manganese, phosphorus, sulphur and silicon. Depending upon the chemical composition of the raw materials used in the blast furnace—principally iron ore, coke and limestone—and the manner in which the furnace is operated, pig iron may contain 3.0 to 4.5 per cent of carbon, 0.15 to 2.5 per cent or more of manganese, as much as 0.2 per cent sulphur, 0.025 to 2.5 per cent of phosphorus, and 0.5 to 4.0 per cent of silicon. In refining pig iron to convert it into steel, all five of these elements must either be removed almost entirely or at least reduced drastically in amount.

Modern steelmaking processes, including the pneumatic processes, are divided into two general classes from the chemical standpoint; acid processes and basic processes. Carbon, manganese, and silicon can be removed with relative ease by any of the processes, either acid or basic. The removal of phosphorus and sulphur requires special conditions that can be met only by the basic processes wherein lime is added to the chemical system to form a basic slag which is capable of forming compounds with phosphorus and sulphur during refining operations, thereby removing them from the metal. Because of the chemical nature of the slags, each of the processes must be carried out in equipment lined with proper refractories. Otherwise, the slags would react with and be neutralized by the lining material while, at the same time, rapidly destroying the lining.

In general, the chemical principle of oxidation is employed to convert pig iron into steel. Each of the steelmaking processes has been devised primarily to provide some means whereby controlled amounts of oxygen can be supplied to the molten metal undergoing refining. The oxygen combines with the unwanted elements (with the exception of sulphur) and, unavoidably, with some of the iron, to form oxides, which either leave the bath as gases or enter the slag. The mechanism by which sulphur is removed does not involve direct reaction with oxygen but depends instead on whether the slag is sufficiently basic and high enough temperatures are attained. As purification of the pig iron proceeds, the melting point of the bath is raised, and sufficient heat must be supplied from some source to keep the bath molten.

Principles and Types of Pneumatic Processes—In common with other steelmaking methods, there are two chemical types of pneumatic processes—acid and basic. In both types, air, high-purity oxygen or combinations of these and other oxidizing gases are blown, under pressure, through, or over the surface of, molten pig iron to produce steel. If air alone is used for blowing, its nitrogen content serves little useful purpose and ac-

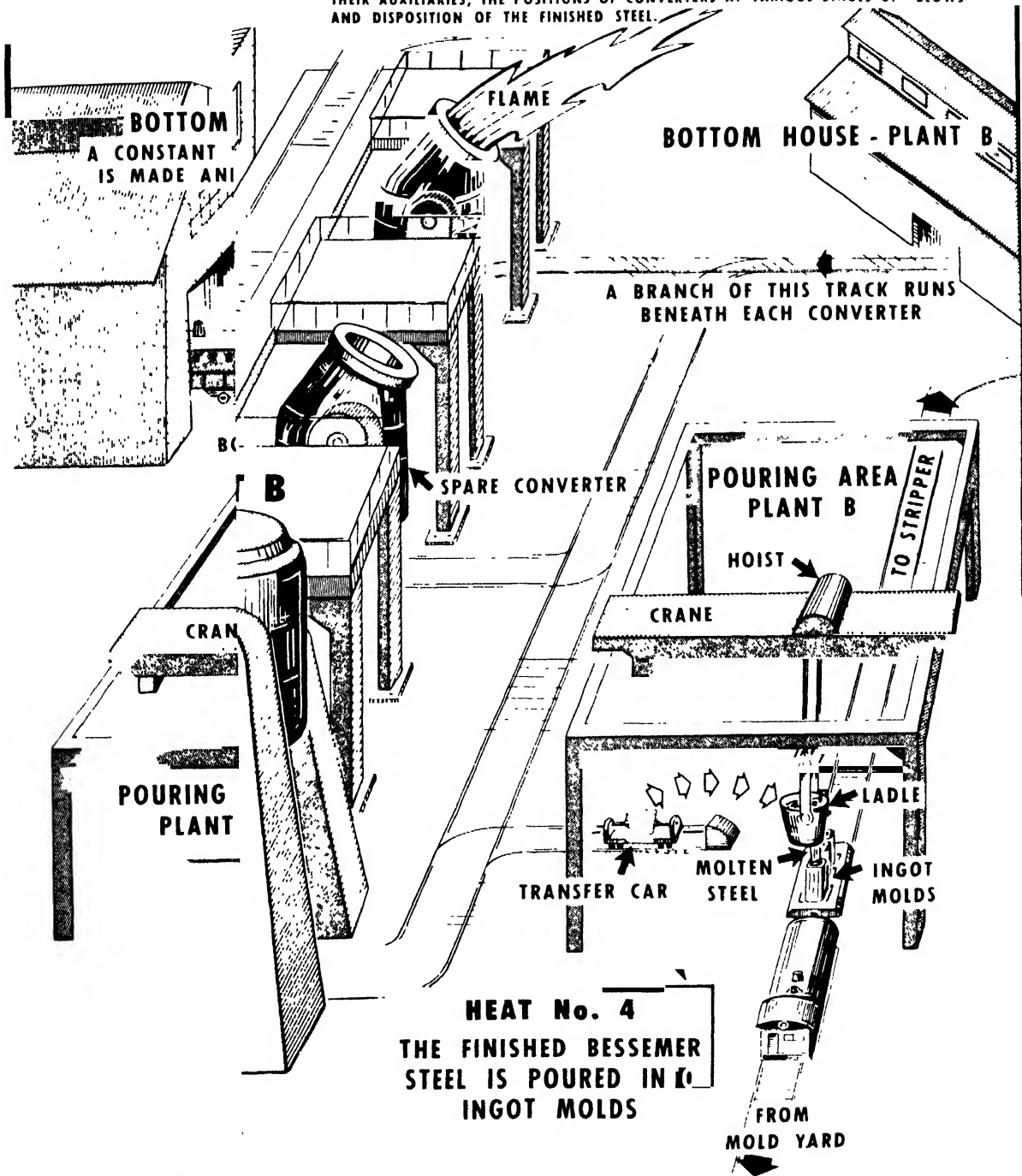
tually removes useful heat from the system. In some cases, nitrogen absorbed during blowing is considered as an undesirable impurity in the finished steel. There are three ways in which the oxidizing gas can be supplied to a pneumatic converter; these are shown schematically in Figure 14—2. The manner in which the oxidizing gas is introduced affects both the relative rates of the refining reactions and the product. The **bottom-blown converter** has been the principal type used in both the acid and basic pneumatic processes for the production of steel ingots (see Sections 2 and 3), the blast traveling the full depth of the bath, thus representing the extreme of submerged blowing practices. The **side-blown converter** can be arranged for either submerged or surface blowing, and significantly different performance characteristics obtain for the two arrangements. The submerged-side-blown converter presents a number of difficult maintenance problems and the design has not been popular, although in recent years a few Thomas converters have been modified to introduce the blast through the side of the vessel. A discussion of the aims and results of the Thomas modification will be found in Section 3. At present the only side-blown converters in extensive commercial use are acid-lined vessels arranged for surface blowing, and these are used chiefly in foundries to produce steel for castings (see Section 4). While basic-lined versions of these foundry converters have never been successful, considerable experimentation has been underway for some time in the United States and several foreign countries to develop a commercially useful side-blown pneumatic process, using a basic-lined converter, for the production of ingots; none of these has yet attained commercial status, so they will not be discussed further in this book. The last variety of surface-blown converter can be termed **top-blown**, the oxidizing gas being introduced by a pipe or lance through the mouth of the vessel. Bessemer and other early investigators employed this or a similar arrangement in some of their designs, and these practices are being used at the present time, chiefly by surface blowing with a high-velocity jet of high-purity oxygen (see Section 5).

Historical Development—The original pneumatic process, developed independently by William Kelly of Eddyville, Kentucky, and Henry Bessemer of England, involved blowing air *through* a bath of molten pig iron contained in a bottom-blown vessel lined with acid (siliceous) refractories. The process was the first to provide a large-scale direct method whereby pig iron could rapidly and cheaply be refined and converted into liquid steel. Bessemer's American patent was issued in 1856; although Kelly did not apply for a patent until 1857, he was able to prove that he had worked on the idea as early as 1847. Thus, both men held rights to the process

THE BESSEMER CONVERTER

(THIS DRAWING IS ENTIRELY SCHEMATIC AND NOT TO SCALE)

THIS IS NOT AN ENGINEERING DRAWING. THE VARIOUS PIECES OF EQUIPMENT ARE NOT DRAWN TO THE SAME SCALE. THE SKETCH IS INTENDED ONLY TO SHOW THE MAJOR EXTERNAL FEATURES OF TWO SEPARATE BESSEMER STEELMAKING PLANTS AND THEIR AUXILIARIES, THE POSITIONS OF CONVERTERS AT VARIOUS STAGES OF "BLOWS" AND DISPOSITION OF THE FINISHED STEEL.



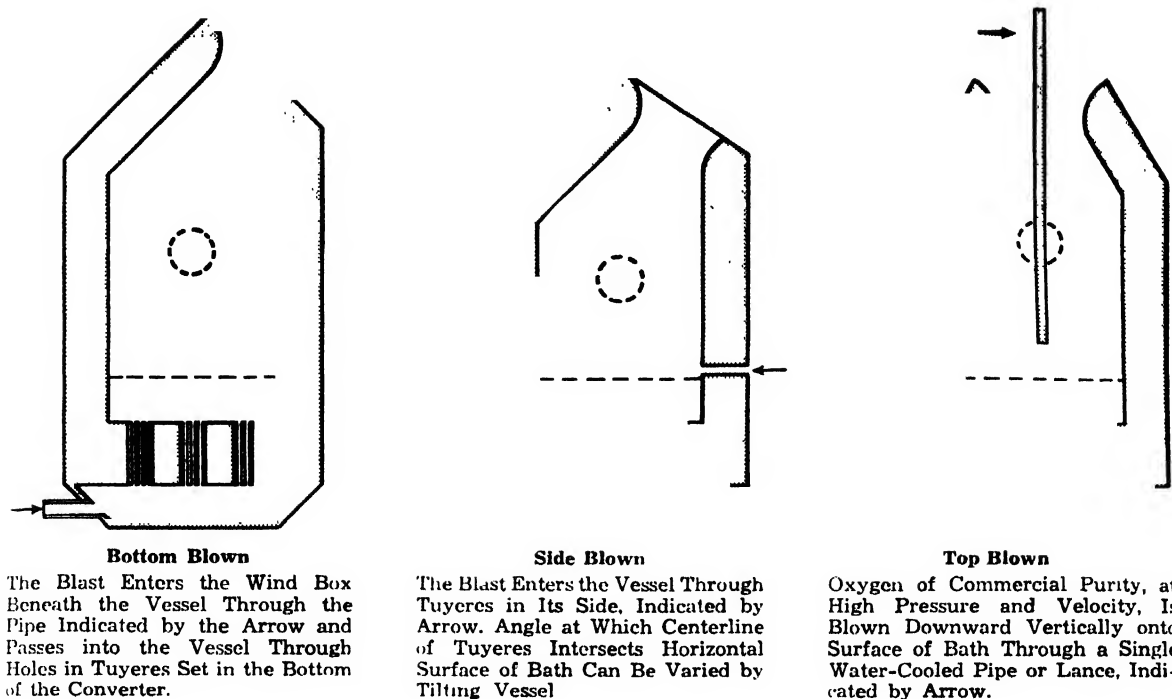


FIG. 14-2. Entirely schematic representation, not to scale, of the three ways in which air and/or oxygen can be supplied to pneumatic converters. Horizontal dotted lines represent approximate level of bath. Dotted circles indicate approximate location of trunnions on which vessels can be rotated.

in this country, this led to considerable litigation and delay, as discussed later. Lacking financial means, Kelly was unable to perfect his invention and Bessemer, in the face of great difficulties and many failures, developed the process to a high degree of perfection and it came to be known as the acid Bessemer process, described in its modern form in Section 2.

The fundamental principle proposed by Bessemer and Kelly was that the oxidation of the major impurities in liquid blast-furnace iron (silicon, manganese and carbon) was preferential and occurred before the major oxidation of iron; the actual mechanism differs from this simple explanation, as outlined under "Chemistry of the Acid-Bessemer Blow" in Section 2. Further, they discovered that sufficient heat was generated in the vessel by the chemical oxidation of the above elements in most types of pig iron to permit the simple blowing of cold air through molten pig iron to produce liquid steel without the need for an external source of heat. Because the process converted pig iron to steel, the vessel in which the operation was carried out came to be known as a converter.

At first, Bessemer produced satisfactory steel in a converter lined with siliceous (acid) refractories by refining pig iron that, smelted from Swedish ores, was low in phosphorus, high in manganese, and contained enough silicon to meet the thermal needs of the process. But, when applied to irons which were higher in phosphorus and low in silicon and manganese, the process did not produce satisfactory steel. In order to save his process in the face of opposition among steelmakers, Bessemer built a steel works at Sheffield, England, and began to operate in 1860. Even when low-phosphorus Swedish pig iron was employed, the steels first produced there contained much more than the admissible amounts of oxygen, which made the steel "wild" in the

molds. Difficulty also was experienced with sulphur which, introduced from the coke used as fuel in melting the iron in cupolas, contributed to "hot shortness" of the steel. These objections finally were overcome by the addition of manganese in the form of spiegeleisen to the steel after blowing was completed. The beneficial effects of manganese were disclosed in a patent by R. Mushet in 1856. The carbon and manganese in the spiegeleisen served the purpose of partially deoxidizing the steel, while part of the manganese combined chemically with some of the sulphur to form compounds that either floated out of the metal into the slag, or were comparatively harmless if they remained in the steel.

Because of trouble with tuyeres in the bottom, some early converters had tuyeres located in the side of the vessel but below the metal-bath surface. Many of the early converters were stationary and had to be tapped in a manner similar to the cupola or the open-hearth furnace; such converters were used for a considerable period in Sweden and Germany. However, the rotating or tilting type was favored in England and in the United States.

As stated earlier, Bessemer had obtained patents in England and in this country previous to Kelly's application; therefore, both men held rights to the process in the United States.

The Kelly Pneumatic Process Company had been formed in 1863 in an arrangement with William Kelly for the commercial production of steel by the new process. This association included the Cambria Iron Company; E. B. Ward; Park Brothers and Company; Lyon, Shord and Company; Z. S. Durfee and, later, Chouteau, Harrison and Vale. This company, in 1864, built the first commercial Bessemer plant in this country, consisting of a 2½ ton acid-lined vessel erected at the Wyandotte Iron Works, Wyandotte, Michigan, owned by Captain

E. B. Ward. It may be mentioned that a Kelly converter was used experimentally at the Cambria Works, Johnstown, Pennsylvania as early as 1861.

As a result of the dual rights to the process a second group consisting of Messrs. John A. Griswold and John F. Winslow of Troy, New York and A. L. Holley formed another company under an arrangement with Bessemer in 1864. This group erected an experimental $2\frac{1}{2}$ -ton vessel at Troy, New York which commenced operations on February 16, 1865. The rival organizations, after much litigation had failed to gain for either sole control of the patents for the pneumatic process in America, decided to combine their respective interests early in 1866. This larger organization was then able to combine the best features covered by the Kelly and Bessemer patents, and the application of the process advanced rapidly.

By 1871, annual Bessemer steel production in this country had increased to 45,000 net tons. This comprised about 55 per cent of the total steel production of the country, and was produced by seven Bessemer plants.

A. L. Holley's contributions to the early development of the process were exceedingly important. He acted as consulting engineer on most of the plants erected in the first ten to fifteen years of the development of the process in the United States. Holley also greatly improved the design of the detachable bottom, originally developed by Bessemer in 1863; this permitted increased tonnage output because it permitted replacement of the bottom when repairs were necessary without excessive loss of production.

A process involving the use of a basic lining and a basic flux in the converter, which made it possible to use the pneumatic method for refining pig irons smelted from the high-phosphorus ores common to many sections of Europe, was patented in England in 1879. The process, known as the Thomas, Thomas-Gilchrist or basic Bessemer process, is described in Section 3.

Another notable contribution to development of the pneumatic processes was made at a later date (1889) by William R. Jones of the Edgar Thomson Works of the Carnegie Steel Co., now a part of United States Steel Corporation, who conceived the idea of the hot-metal mixer. The purpose of the mixer and its construction are described in Section 2.

Characteristics of Pneumatic Steels—While the pneumatic steelmaking methods possess the advantages of speed and simplicity, there are certain chemical disadvantages common to both the acid and basic processes.

For example, bottom-blown acid Bessemer steels are generally higher in phosphorus, sulphur and nitrogen contents than basic open-hearth steels. Steels produced by the bottom-blown basic pneumatic processes may approach basic open-hearth steels with respect to phosphorus and sulphur contents, but it is more difficult to produce steel of low nitrogen content by this process. When their phosphorus, sulphur and nitrogen contents are high in comparison with basic open-hearth steels, steels produced by the pneumatic processes possess higher yield and tensile strengths and lower ductility than the open-hearth steels. Further, when the nitrogen content is high, pneumatic steels are subject to some loss of ductility due to aging. When air alone is used for blowing, bottom-blown vessels produce steel with the highest nitrogen content, side-blown vessels produce steel possessing intermediate nitrogen content, and top-blown vessels produce steel with the lowest nitrogen content. Another disadvantage of most pneumatic steels is the inability to control carbon and oxygen contents of the finished steels as closely as can be accomplished in the basic open-hearth process. Because of the nature of pneumatic processes, it is presently impossible to closely control the final carbon content of pneumatic steels by stopping the blow at the exact time when the carbon content has fallen to the desired level, except at very low levels of carbon content. This generally limits the variety of steels made to those containing 0.30 per cent and less carbon (although commercial steels containing up to 0.50 per cent carbon are produced). The higher-carbon steels are made by blowing the metal to a consistently low carbon content and then adding the required amount of carbon in the form of recarburizers. In addition to the control of carbon, some difficulty is also experienced in controlling the final temperature and state of oxidation of pneumatic steels.

In general, steels which have similar chemical compositions have similar physical and mechanical properties. Steels made by pneumatic steelmaking processes which have the same chemical composition as those made in the basic open-hearth furnace, particularly with regard to phosphorus, sulphur, nitrogen and other contents, will have properties quite similar to the basic open-hearth steels and may be used in the same general applications. There are some applications where pneumatic steels, particularly those produced by the acid Bessemer process or variations of it, are superior to steel produced by any other method because of the desirable mechanical and physical properties resulting from the chemical composition of the pneumatic steels.

SECTION 2

THE BOTTOM-BLOWN ACID PROCESS

As has already been stated, the bottom-blown acid process now known generally as the acid Bessemer process was the original pneumatic steelmaking process. Many millions of tons of steel have been produced by this method. From 1870 to 1910, the acid Bessemer process produced the majority of the world's supply of steel.

The success of acid Bessemer steelmaking is dependent upon the quality of pig iron available which, in turn, demands a reliable supply of relatively high purity ore and good metallurgical coke. At the time of the invention of the process, large quantities of this type of ore were available, both abroad and in the United States. With the gradual depletion of supplies of high-quality ore (particularly low-phosphorus ores) abroad and the rapid expansion of the bottom-blown basic pneumatic and basic open-hearth processes of steelmaking, acid

Bessemer steel production declined to a very low level percentage-wise, particularly abroad in the early twentieth century. Sweden and Great Britain are the only European countries with adequate reserves of ores suitable for the production of steels by the acid Bessemer process.

In the United States, the Mesabi Range has provided a source of relatively high grade ore for making iron for the acid Bessemer process for many years. In spite of the availability of good ore in this country, the acid Bessemer process has still declined from a major to a minor steelmaking method in the United States. The reasons are partly metallurgical and partly economic.

The early use of acid Bessemer steel in this country involved production of a considerable quantity of rail steel, and for many years this process was the principal

method used for the production of steel, as shown by Table 14—I. At the present time, the acid Bessemer process in the United States is used principally in the production of steel for butt welded pipe, seamless pipe, free-machining bars, flat-rolled products, wire, steel castings, and blown metal for the duplex process. The rated converter capacity of the nation is approximately 12,000,000 tons of steel. Over half of this capacity represents blown metal for use in the open-hearth or duplex process and the remainder Bessemer ingots. Most of the steel in this country is manufactured by the open-hearth process, and during the past few years few new converters have been installed, although extensive experimental work on converter steel is in progress. The inherent advantages of the process involve economic considerations in times of peace and national security in times of war. Fully-killed acid Bessemer steel was used

for the first time commercially by National Tube Division of United States Steel Corporation in the production of seamless pipe. In addition, dephosphorized acid Bessemer steel has been used extensively in the production of welded pipe and galvanized sheets.

The modern shop for using the acid Bessemer process usually is built in conjunction with basic open-hearth furnaces to take advantage of the so-called "duplex" process (see Chapter 17). Since the production of duplex steels is covered in a later chapter, the present discussion is confined to that part of the Bessemer converter tonnage manufactured as acid Bessemer steel in ingot form.

PRINCIPAL FACILITIES

Plant Layout—Newer installations usually consist of three bottom-blown converters, also called vessels, of

Table 14—I. Annual Production of Steel for Ingots and Castings (Net Tons)^(a)

Year	Bessemer	Open Hearth		Electric Furnace	Crucible	Steel for Castings	Total
		Acid	Basic				
1870	42,000		1,500	None	(b)	?	77,000
1880	1,203,173		112,953	None	(b)	?	1,397,015
1890	4,131,536		574,820	None	79,716	?	4,790,320
1900	7,486,942		3,805,911	(b)	112,629	?	11,410,928
1910	10,542,305	876,742	16,641,355	56,919	136,979	?	29,226,309
1911	8,901,596	681,131	16,149,623	29,494	109,371	?	26,517,238
1920	9,949,057	1,451,712	35,140,810	566,370	80,937	?	47,188,886
1921	4,497,851	325,640	16,648,360	190,897	8,527	?	22,157,853
1929	7,977,210	1,254,926	52,900,309	1,065,603	7,442	?	63,205,490
1930	5,639,714	874,559	38,380,514	676,111	2,523	?	45,583,421
1931	3,386,259	424,668	24,786,046	460,255	1,733	?	29,058,961
1932	1,715,925	184,406	13,151,804	270,044	722	?	15,322,901
1933	2,720,246	363,469	22,464,004	471,747	763	?	26,020,229
1934†	2,421,840	307,651	26,047,187	404,651	595	119,331	29,181,924
1935	3,175,235	396,695	34,004,585	606,471	719	170,407	38,183,705
1936	3,873,472	471,858	48,288,605	865,150	914	285,653	53,499,999
1937	3,863,918	559,768	51,265,211	947,002	1,046	280,616	56,635,945
1938	2,106,340	305,017	28,774,999	565,627	7	155,848	31,751,990
1939	3,358,916	581,100	47,828,700	1,029,067	931	261,275	52,798,714
1940	3,708,573	690,243	60,882,840	1,700,006	1,024	332,822	66,982,686
1941	5,578,071	1,076,768	73,312,851	2,869,256	2,313	404,892	82,839,259
1942	5,553,424	1,318,892	75,183,065	3,974,540	2,010	485,755	86,031,931
1943	5,625,492	1,413,934	77,207,870	4,589,070	146	451,510	88,836,512
1944	5,039,923	1,195,659	79,168,294	4,237,699	25	399,585	89,641,600
1945	4,305,318	869,726	71,069,876	3,456,704	24	310,831	79,701,648
1946	3,327,737	599,663	60,112,300	2,563,024	(c)	267,624	66,602,724
1947	4,232,543	664,525	76,209,268	3,787,735	(c)	298,133	84,894,071
1948	4,243,172	625,305	78,714,852	5,057,141	(c)	286,595	88,640,470
1949	3,946,656	506,693	69,742,110	3,782,717	(c)	248,135	77,978,176
1950	4,534,558	600,858	85,661,651	6,039,008	(c)	296,461	96,836,075
1951	4,890,946	779,071	92,387,447	7,142,384	(c)	346,187	105,199,848
1952	3,523,677	703,039	82,143,400	6,797,923	(c)	323,756	93,168,039
1953	3,855,705	646,094	99,827,729	7,280,191	(c)	283,555	111,609,719
1954	2,548,104	307,866	80,019,628	5,436,054	(c)	191,099	88,311,652

^(a)From Annual Statistical Reports, American Iron and Steel Institute.

^(b)Figures not available.

^(c)Included with electric steel.

†The figures for 1934 and subsequent years include only that portion of the steel casting production which was produced in foundries operated by companies producing steel ingots.

from 25 to 30 tons capacity, one of which is maintained as a spare. The shop is on three levels with the pouring floor at ground level. Two sets of standard-gage track service the vessels. The first set runs along the back of the shop with spurs running under each vessel. On these spurs are placed the slag pots and, when it is necessary to change bottoms, the bottom-hoist car. The other tracks run in front of the vessels and are used to position the ladles. A teeming floor, with the necessary cranes for handling the ladles from the cars, is located close to the converter shop.

On the second level are located the vessels, vessel-repair car and the iron-transfer ladle car to carry the molten metal from the mixer to the vessels (Figure 14-3). To the side of the shop are located the mixers and the bins for ferromanganese and other ladle additions. On the same level, but located across the shop from the vessels, is the control pulpit. On the third level are the tracks for ladle cars used to fill the mixers, the mixer controls and scales for the iron-transfer ladle.

The Air Blast—Adjacent to the converter building is located the blowing room, wherein the air blast is generated by turboblowers, although in older shops, steam-driven, vertical, reciprocating-type blowing engines are still in use. The blast is carried into the converter building through large mains which lead to the hollow trunnion of each vessel. Each line is fitted with control valves which are regulated from the pulpit by the blower. Also

connected to each air-blast line is a steam line fitted with valves for injection of steam into the blast for temperature-control purposes. These valves also are regulated from the blower's pulpit. The pressure in the main is regulated by an automatic blow-off valve, which functions when the vessel is off blast during charging or pouring. Blast pressures normally range from twenty to twenty-five pounds per square inch. Pressures up to thirty-five pounds per square inch are used on larger vessels. Although pressures as low as eighteen pounds per square inch are sufficient to prevent the molten iron from entering and blocking the tuyere holes, the use of this low pressure would prolong the blowing time unnecessarily.

Pressures over the indicated maximums are not used because of the large amount of metal which would be ejected from the converter during blowing.

Bottom House—The bottom of the converter is the part subject to the most wear and requires frequent replacement. Adequate facilities must be provided to insure a constant supply of rebuilt bottoms. These facilities include adequate storage bins or hoppers for the refractories used, equipment such as pug mills for the grinding and mixing of the refractories, pneumatic tools for ramming the bottom material in place, a storage area for tuyeres, and ovens for drying the completed bottoms. These ovens may be either of the in-and-out batch type or tunnel ovens of the continuous type.



FIG. 14-3. General view of a recent installation of Bessemer converters, showing one of the three vessels blowing. Slag pots and the bottom-hoist car operate on standard-gage tracks on the level beneath the vessels.

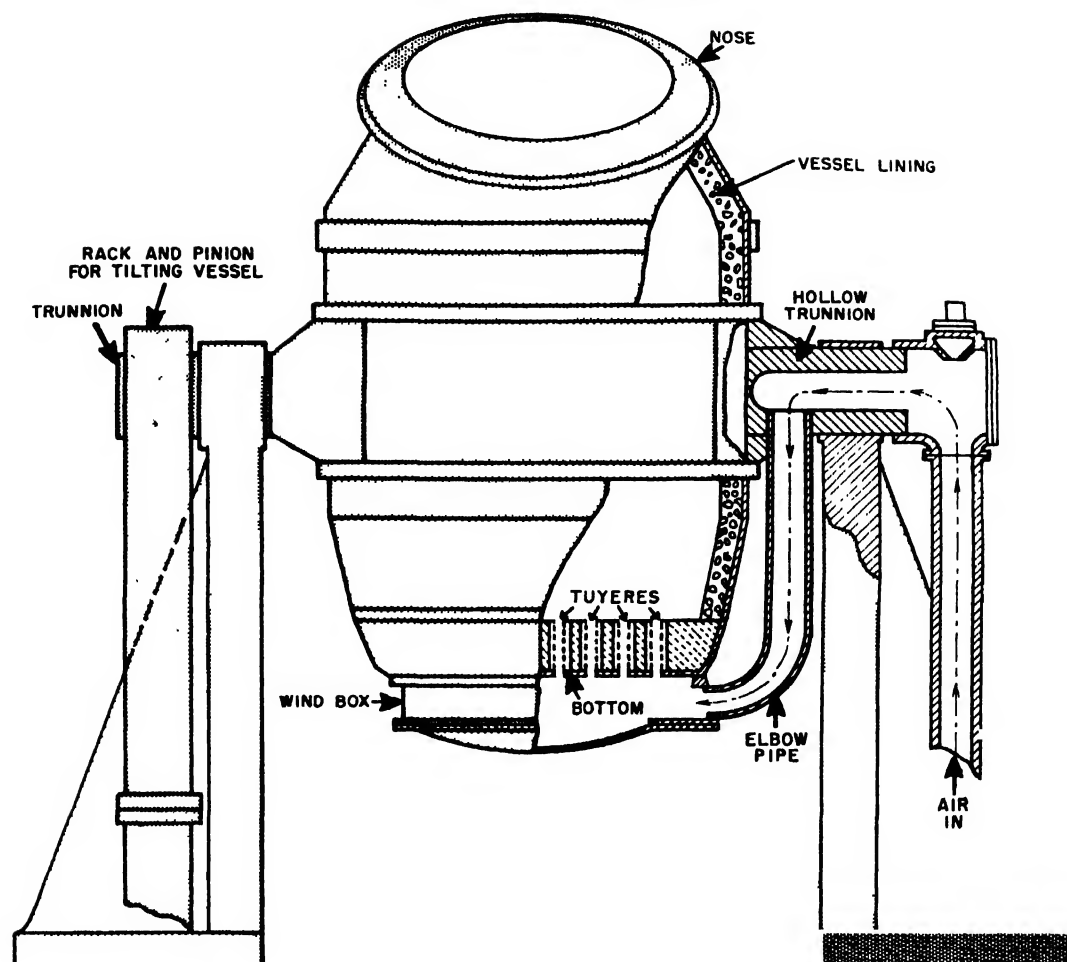


FIG. 14-4A. (Above) Diagram illustrating the details of construction that permit air to enter the converter through a hollow trunnion, so that the vessel can be turned down without interrupting the flow of air. This schematic diagram is not intended to show actual details of vessel construction, many of which are visible in Figure 14-5, showing an actual converter in operation. (See also Figure 14-4B).

THE CONVERTER OR VESSEL

The cylindrically shaped vessel consists of a steel shell of riveted or welded construction, supported on two horizontal trunnions upon which it can be made to rotate in a vertical plane (Figures 14-4 and 14-5). One of these trunnions is hollow, through which the blast is passed by a pipe called the gooseneck or elbow pipe, to the windbox at the bottom of the vessel. A pinion is fastened to the other trunnion and engages an electrically operated rack to tip the converter. The bottom, which is detachable, is pierced with twenty-eight tuyeres—larger vessels have as many as thirty-five—through which the air blast passes from the wind-box into the metal bath.

Refractory Lining—With the vessel in the inverted position, the steel shell is lined with siliceous stone. Cut and dressed sandstone (firestone) or micaceous schist is used, most operators preferring a combination of the two. The micaceous schist usually is placed in those areas of the lining subject to the most mechanical wear. This stone has a laminated structure composed of tiny plates of mica, and, when laid in such a manner that the edges of the laminations are exposed as the wearing surface, it is very durable. The stone is laid up tightly

with a thin layer of refractory fireclay and keyed into position to insure against movement when the vessel is tilted. Micaceous schist contains about 90 per cent silica, 3 to 7 per cent alumina and 2 to 4 per cent iron oxide; while sandstone is composed of approximately 93 per cent silica and about 4 per cent alumina, the remaining 2 to 3 per cent being iron oxide, lime and magnesia. Linings of this type have a life of approximately 25,000 tons of steel ingots. The linings are dried before using by a flame inside the vessel, supplied by a "lance" burning gas or oil that is inserted through the mouth of the converter.

Bottom Design—The tuyeres which admit the air blast from the wind-box into the vessel proper are cylindrical in shape, ranging in length from 26 to 36 inches depending on the size of the vessel. They are made of a good-quality fireclay, medium burned. Soft tuyeres deteriorate rapidly during blowing and decrease bottom life, while hard-burned tuyeres are too friable. The most commonly used tuyere is one containing seven $\frac{5}{8}$ -inch diameter holes which run the length of the tuyere block. The tuyeres are slightly mushroomed on one end, so that, when inserted through the tuyere plate, they form a tight fit. The tuyeres are arranged in the bottom usually in two or more concentric circles with one tu-

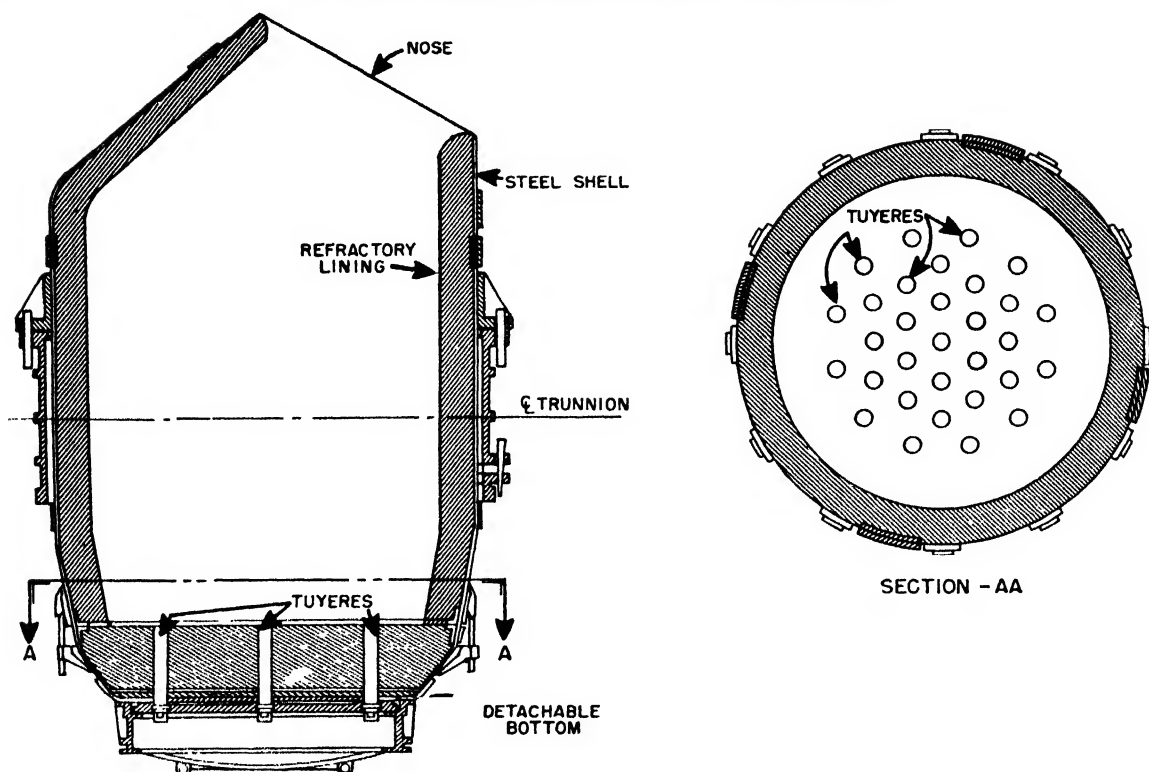


FIG. 14-4B. Plan of bottom and section through a 25-ton eccentric Bessemer converter, so-called because of the location of the opening in the nose. Concentric converters have the nose opening parallel to and concentric with the bottom.

yere in the center. On making a bottom, the tuyeres are set in position protruding through the cast tuyere plate. The large ring-shaped castings comprising the structural part of the bottom are used repeatedly over long periods of time. Extra bottom castings are, of course, available so that new bottoms can be in the process of being built while the converting mill is operating.

After positioning the tuyeres, long rectangular tile brick are positioned vertically throughout the bottom. The number used may vary anywhere from as few as six to as many as forty per bottom. It is felt that these tile brick act as stiffeners and prolong the bottom life. The number used must be balanced against the expected increase in bottom life, since their use increases the cost of the bottom.

The spaces between tuyeres and tiles are then filled with a refractory material suitably ground and moistened in a pug mill. The bottom material, consisting of siliceous stone such as ganister, brickbats and clay together with reclaimed material from spent bottoms, all thoroughly ground and mixed together, is shoveled into place and rammed solidly, particularly around the tuyeres, to form a compact mass. This constitutes the so-called **dry bottom**, which is then thoroughly dried and baked in the ovens for a minimum of forty-eight hours at temperatures ranging from 300 to 400° F to insure complete drying. Some operators prefer a **wet bottom**, in which materials are moistened to the consistency of a concrete and poured much in the same manner. Longer drying cycles and higher temperatures are required for this type of bottom.

Life of Bottoms—The bottom of the converter, as already stated, is subject to severe wear. On the average, bottom life will range only from twenty-five to thirty-

five "blows" for a plant whose total output is mixed steel ingots and duplex metal. The vesselman examines the bottom periodically while the vessel is inverted. If he notices that a particular tuyere is wearing more rapidly than the other tuyeres, which is referred to as **boring**, and thus endangering the life of the bottom, he will direct the removal of the plate from the bottom of the wind-box. The bad tuyere will then be plugged with a blanking plate or clay to prevent the blast from passing through it. As many as ten or twelve tuyeres may be so blanked before the bottom is replaced.

Once a tuyere begins boring, deterioration will proceed at an exceedingly fast rate, sometimes as much as one inch per minute. An explanation for boring that has been advanced is the possibility that the holes in the tuyere were not spaced symmetrically. As a result of two tuyere holes being abnormally close together a cavity may form in the tuyere end. This cavity would be filled with a small pool of metal because of the localized reduction of effective blast pressure. The small pool would be set in rapid motion by the blast and the metal would become highly oxidized and super-heated. In this condition it would be highly corrosive and chemical attack on the tuyere material would result. Once this reaction had started it would proceed at an increasing rate, thus causing rapid deterioration of the tuyere.

The bottom is changed when the tuyeres are worn down to a length of about eight inches. The change requires approximately thirty minutes. The bottom plate of the wind-box is removed. A small car is set in position under the vessel, and the vessel is turned to the vertical position. A hydraulic jack raises the car until it contacts the bottom. The keys holding the bottom in

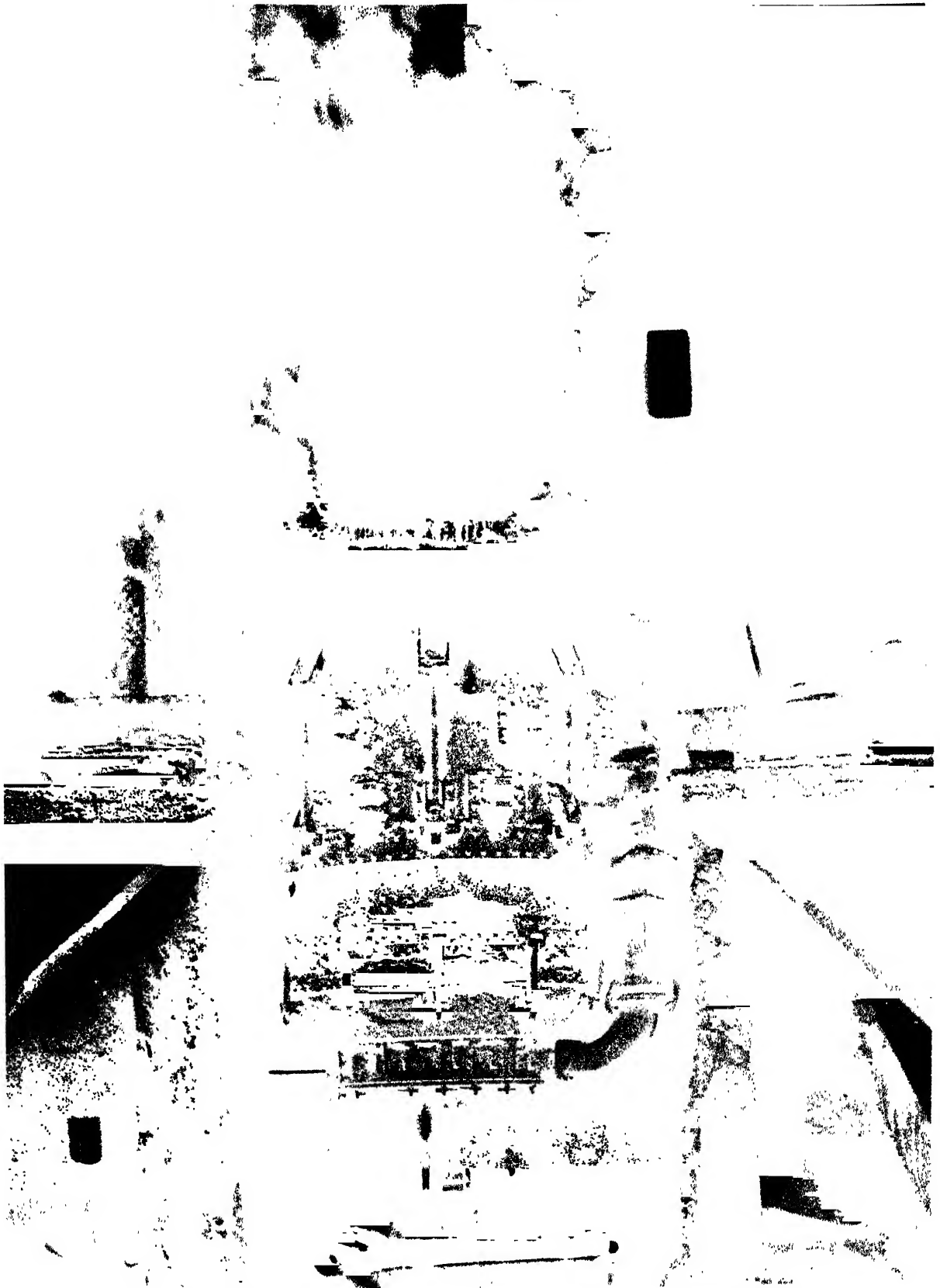


FIG. 14-5. Close-up view of a Bessemer converter during the progress of a "blow."

place are knocked out, the connecting links disengaged, and the bottom allowed to settle on the car. The car is then lowered to ground level and pulled out of position by a standard-gage locomotive. The new bottom, mounted on a similar car, is placed in position, and the operation reversed.

Cupolas—Many of the earlier Bessemer plants were small, isolated units often located some distance from the source of pig iron. To supply the liquid-iron charge required by the process, solid pigs of iron were melted in reverberatory furnaces, superseded in a short time by cupolas. The cupola is a cylindrical stack-type furnace into which successive layers of coke, pig iron and limestone are charged at the top. A cold-air blast produced by fans, under a pressure of six to ten ounces, is introduced into the bottom of the furnace above the hearth through tuyeres located symmetrically around the circumference. The cupola consists of a steel shell, lined with firebrick in its upper part and firestone in the lower part which is subjected to higher temperatures. The temperature generated at the tuyeres by combustion of the coke is sufficient to melt the iron and also superheat it. The limestone, having calcined in the stack, also fuses, combining with the ash in the coke to form a slag. The molten iron accumulates in the hearth from which it is drawn periodically, depending on the requirements of the converter, by opening the tap hole and allowing the iron to flow by way of a spout into the transfer ladle. Spiegel, a high-manganese pig iron, containing about 5 per cent carbon, formerly was melted in a similar manner and was used to recarburize the blown metal.

The use of the cupola had certain inherent disadvantages which from the standpoint of production and quality control were most undesirable. Much of the sulphur and phosphorus in the coke was absorbed by the iron. These elements being undesirable above certain limits in the steel, and the acid process being incapable of removing them, it was necessary to use the best type of coke available. For this reason, as well as for economy, the amount of coke charged was held to a minimum. These factors, combined with fluctuations in iron composition, caused erratic swings in iron temperatures, resulting in difficulties in controlling the blowing of the converters. The desirability of using molten iron from blast furnaces became apparent at an early date, but the intermittent casting of the blast furnace, coupled with the large amount of molten metal taken from the furnace at one time, made such practice impracticable until the hot-metal mixer was invented. The advantages of the hot-metal mixer described below were quickly recognized in the industry and, at those plants where sufficient blast-furnace facilities were available, it replaced quickly those cupolas used for melting iron for blowing in the converters.

Hot-Metal Mixer—The fundamental purpose of the hot-metal mixer is to store molten iron from the blast furnaces until it can be used in the Bessemer converters or in the open-hearth furnaces. A modern mixer is illustrated in Chapter 15.

The modern plant usually has two or three mixers of from 200 or 300 to 1500 tons capacity. The mixer is a large, cylindrical vessel mounted horizontally upon two sets of races and rollers, which in turn are set upon firm foundations of concrete. The vessel is rotated about its central axis by an electric motor connected through reduction gears to a large screw, which actuates the vessel through a connecting rod attached to its bottom. The vessel is constructed of steel plates joined together to form a shell, which is lined with a good grade of fire-clay brick or micaceous schist. The lining is usually

about two feet in thickness. It is not uncommon for a mixer to be in continuous operation for a year before it is necessary for it to be drained in order to repair the lining. Complete relining is necessary about every three to four years. Molten iron is received from the blast furnace in ladles mounted on cars. These ladles are heavily insulated to conserve heat and are filled and emptied through a hole on the top. Molten iron is poured out of this hole into an opening on top of the mixer by revolving the ladle on its trunnions. On the front of the mixer is a second opening provided with a spout which permits hot metal to be drawn off as required by turning the vessel. The mixer, acting as a reservoir for molten iron, serves these useful purposes:

1. It conserves the heat in the molten pig iron, and is capable of maintaining the iron in the molten condition for long periods of time. Many mixers are equipped with gas- or oil-fired burners inserted in the ends, whereby additional heat can be made available if necessary.
2. It promotes uniformity of iron by mixing the product of several blast furnaces, thus equalizing the irregularities of individual casts. This permits delivery of iron of more uniform composition and temperature to the converters, which enables the blower to control steel-making variables more closely.
3. Because of its large capacity, it can permit independent operations between the blast furnaces and the converter shop to a large extent, since any ordinary delay in the operations of either unit will not affect the other seriously.

OPERATION OF THE PLANT

Sequence of Operations—After completion of the previous blow, the vessel is turned on its trunnions until it assumes an almost horizontal position and scrap, scale, or ore is dumped into the vessel as desired. The molten pig iron is then poured in from the transfer ladle, as shown schematically in Figure 14-6. In this horizontal position the metal is contained in the belly of the converter and does not come in contact with the tuyeres. The blast is started and the vessel is turned to a vertical position and remains in this position throughout the balance of the blowing period, unless "side blowing" is resorted to for increasing temperature, as discussed below.

The Bessemer blow is usually thought of as being divided into three parts, the first period, the second period and the after blow.

The first period, or the **silicon blow**, as it is commonly called, begins as the blast is turned on and the vessel turned up. During this period of the blow, a short, transparent flame extends from the mouth of the vessel. As the blowing continues the flame starts to lengthen after about four minutes and the second period or **carbon blow** begins.

The long, brilliant flame which is characteristic of the carbon blow continues until the elimination of carbon approaches completion, whereupon there is a definite change in the appearance of the flame. The flame gradually shortens in length and seems to fan out. As viewed through the blower's colored glasses, streaks of red appear in the flame at the mouth of the converter, and then almost instantaneously the whole flame changes from the usual golden yellow to a reddish-appearing flame. This change, which always occurs at almost the same carbon content from blow to blow, has been designated as the **end point**. As such it has been established as a control reference point by the blower. On some

POSITIONS OF BESSEMER CONVERTER AND CONTENTS DURING VARIOUS OPERATIONS

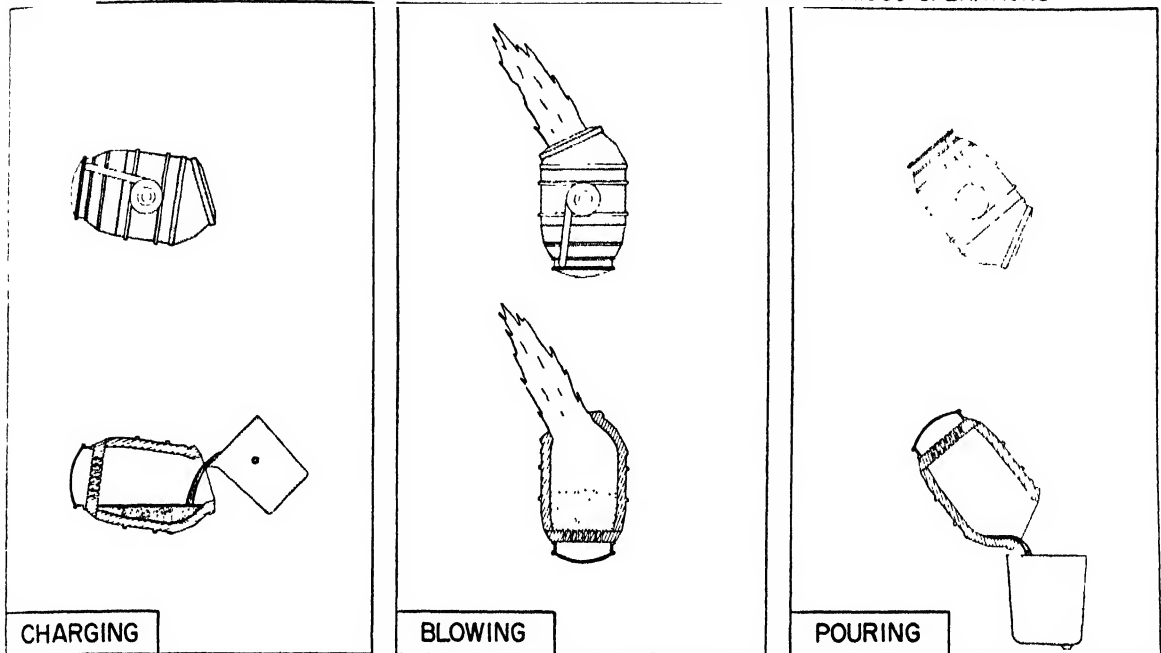


FIG. 14-6. Positions assumed by the Bessemer converter when [left] pouring molten pig iron into the vessel, [center] blowing compressed air through the molten metal, and [right] pouring the refined metal [steel] into the ladle whence it is poured into ingot molds. The bottom left sketch shows how tilting the vessel raises the tuyeres above the level of the molten pig iron charged into the converter. The bottom center illustration is drawn to indicate the highly turbulent condition of the molten metal due to the passage of gases during the blow. Note that some slag, not indicated on the drawings, accompanies the molten metal.

grades of steel the blow will be terminated promptly on reaching the end point. These blows are then said to be **young blown** heats. In other cases, it is necessary to prolong the blowing beyond this point as long as fifteen or twenty seconds. The time interval from end point to turn down is referred to as the **after blow**, and heats handled in this manner are said to be **full blown**.

It is during the second period that the flame attains its full brilliance and length, extending as much as thirty feet beyond the mouth of the converter. This flame results from the evolved carbon monoxide burn-

ing to carbon dioxide as it comes in contact with the air at the mouth of the converter. It is at this time that the blower carefully studies the flame to determine if his judgment regarding temperature control, and in estimating the amount of scrap needed, initially was correct. If the flame shows brilliant white patches and is extremely feathery around the edges, the temperature in all probability is too high. By introducing steam into the blast line the temperature can be lowered because of the heat absorbed through the dissociation of water vapor. A dull flame streaked with dark patches is indicative of low temperature. The temperature can be increased by resorting to **side blowing**. This involves tipping the converter over on its back, thereby exposing several of the tuyeres above the metal bath. By so doing, oxidation of carbon monoxide to carbon dioxide is promoted within the vessel itself, thereby increasing the temperature of the bath by the radiation of heat.

Darken and Gurry have calculated the comparative effect of various coolants used in the process to control temperatures (Table 14-II).

Table 14-II. Effect of Coolants in Bessemer Blowing

Addition of 100 lbs.	Cooling Effect Produced
Pig iron	42,000 Btu
Steel scrap	59,000 Btu
Dry ore	161,000 Btu
Steam or water vapor	415,000 Btu

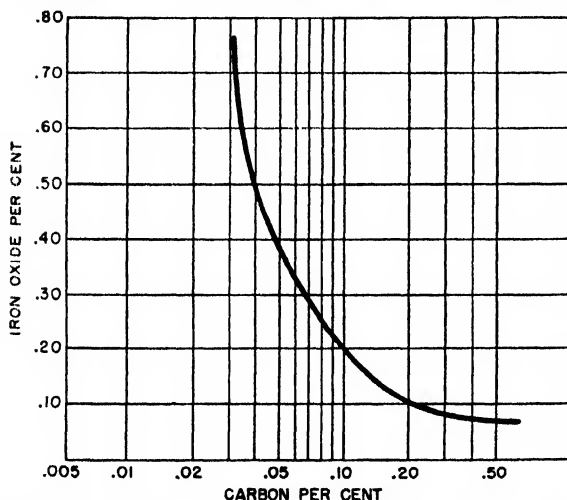


FIG. 14-7. Curves showing relationship between the carbon content and iron-oxide content of the metal in a converter.

Studies have shown that there is a definite relationship between the carbon and iron oxide in the converter. This relationship is exhibited in Figure 14-7. Uniformity of product is influenced by control of iron-oxide content. After the carbon content has been lowered to

0.05 per cent, any further lowering of the carbon is accompanied by an increase in iron-oxide content. It is, therefore, apparent how important the determination of the end point becomes in insuring uniformity of product from blow to blow.

The quality of the pig iron furnished by the blast furnaces to an acid Bessemer shop will determine to a great extent the quality of the steel which can be produced. Iron of the correct chemical composition and temperature is required. For the best blowing, both from the standpoint of production and of quality, iron within the following ranges in composition is desired:

Element	Per Cent
Silicon	1.10 to 1.50
Manganese	0.40 to 0.70
Phosphorus	0.090 max.
Sulphur	0.030 max.

It is felt that for best blowing conditions a ratio of silicon to manganese of two or two and one-half to one should be maintained. The carbon content of the iron will range from 4.00 to 4.50 per cent, but chemical determination of this element in the iron generally is not made.

A typical charge for a 25-ton acid Bessemer converter is as follows:

Hot Metal (Bessemer).....	55,000 lbs.
Scrap	7,000 lbs.

With the above charge the blowing time will vary from eleven to fifteen minutes, depending upon the chemical composition of the iron and the condition of the bottom in respect to the number of blanked tuyeres. For the smaller, 10-ton converters, the time is shorter, averaging nine to eleven minutes.

Roll-scale additions are sometimes made in order to decrease blowing time. The roll scale, which is largely iron oxide, begins immediately to react with the silicon and manganese in the iron, and thus hastens their elimination. The scrap is added as a coolant. The amount required is estimated by the blower from his observations as to whether or not a satisfactory final temperature was obtained on the previous blow. The amount necessary to use will vary with the actual or physical temperature of the iron, as well as the potential chemical heat available in the iron, which in turn depends on its composition.

After the blower has made the decision that the blow should be terminated, the vessel is turned down and the blast turned off. Molten iron is added to the vessel at this time if it is desired to produce killed Bessemer steel, as described later under "Carbon Deoxidation." The ladle is positioned under the mouth of the converter to permit the blown metal to be poured into it. This operation is done slowly and carefully in order to retain as much slag in the vessel as practical. While the metal is pouring into the ladle, the recarburizers and de-oxidizers are added. When the vessel has been drained, the ladle is withdrawn and the vessel turned further from the vertical to permit the remaining slag to drop out into a slag pot mounted on a car below. The ladle is carried to the teeming platform, and the steel is teemed into the ingot molds. In the practice being described, each ladle is capable of containing all of the steel produced in two blowing operations; the content of a full ladle is referred to as a "heat."

STEEL-DISPOSAL EQUIPMENT

Ladles—Cup-shaped ladles constructed of steel plates are used to transfer the molten steel from the converter to a location where it can be teemed into the ingot molds. These ladles are generally similar to those de-

scribed and illustrated in Chapter 15 in connection with open-hearth practice. The ladles are lined with ladle brick and sealed with a thin clay slurry. To prevent the slag, which always floats on the steel in the ladle, from entering the mold and contaminating the ingot, the steel is teemed through a small hole, 2 inches in diameter, in the bottom of the ladle. For opening and closing this hole when moving from mold to mold, the ladle is equipped with a stopper which can be operated by the steel pourer on the teeming platform. This stopper consists of a steel rod, protected with fireclay sleeves, to the lower end of which is attached the stopperhead made of graphite, which fits snugly into the nozzle in the bottom of the ladle. The upper end of the stopper rod is fastened to a gooseneck which is keyed to a vertical sliding bar attached to the outside of the ladle. This bar is provided with a lever by which it may be raised or lowered, causing a like movement of the stopper. After the teeming of each heat the ladle is upended by an auxiliary hoist to dump the slag into the slag pot. At this time a wad of clay is inserted into the nozzle to guard against a leaking or running stopper when the ladle is refilled. The stopper rod is replaced after teeming so as to be ready for the next heat. The ladle is patched after teeming about eighteen heats. At this time the rim of the ladle is cleaned of adhering slag or steel, patched with clay and a new nozzle inserted in the bottom. Ladles require complete relining only after having teemed 50 to 60 heats or 100 to 120 blows.

Ingot Molds—The ingot molds into which the finished steel is teemed are made of cast iron. The standard mold used at the plant being discussed here as an example has a 24-inch x 27-inch average internal cross-section and is 81 inches high. The mold corners are rounded, the mold wall is corrugated, and the sides are tapered to facilitate stripping of the mold from the ingot. The molds are open at both ends and, when ready for teeming, rest big-end-down on heavy cast-iron plates called stools. A stool large enough to accommodate three molds is mounted on a small narrow-gage car or buggy. Buggies are constructed so that their sides form aprons which protect both the track on which the cars run and their own running gear from splattering by molten steel during teeming. The care of the molds is highly important, since dirty or defective molds will result in defects on the surface of the ingots, which in turn will be reflected in inferior surface of the rolled product. Good teeming practice resulting in clean, scab-free ingots, pays dividends in subsequent operations, since the cost involved in removing surface defects from the semifinished rolled product is thereby minimized.

The Stripper—As the name implies, this piece of equipment is designed to remove or strip the molds from the ingot after a sufficient time interval has elapsed after teeming to permit a substantial shell of solidified steel to have formed on all six surfaces of the ingot. It consists of a powerful overhead crane, from which is suspended a vertical arm with two jaws that engage lugs cast on the outside of the mold on either side near the top. Operating between the jaws is a ram or plunger, capable of exerting pressure on the top of the ingot. In stripping the ingot, the jaws engage the lugs and exert a powerful upward pull on the mold, while the ram, having been inserted through the open top of the mold, holds the ingot on the stool until the mold is loosened. The mold is then raised high enough to clear the ingot, and placed upon an empty car standing on an adjoining track. Illustrated descriptions of strippers are included in Chapter 15.

The older stripper installations are operated hydraulically, while the newer are powered electrically. The

electric strippers, since they are of the traveling type, possess a decided advantage over the older hydraulically operated machines, which are stationary. The most favorable location for the stripper is as close to the soaking-pit building as space requirements will permit. This is important, in that minimizing the time that elapses between stripping the molds from the ingots to completion of charging the ingots into the blooming-mill soaking pits, other things being equal, will result in better surface quality of the rolled product and reduce the amount of fuel required for reheating in the soaking pits.

CHEMISTRY OF THE ACID-BESSEMER BLOW

Some knowledge of the laws controlling chemical reactions is necessary for a clear understanding of the various reactions occurring in the acid Bessemer converter. The law of mass action states, in effect, that the rate or speed of a chemical reaction may be increased by increasing the active masses, or amounts, of one of the reacting substances; another law says that when chemical reactions take place without the addition of heat from an external source, those substances which have the greatest heats of formation, that is, those that give off the most energy during formation, will tend to form.

As previously stated the Bessemer blow usually is thought of as being divided into three parts, called the silicon blow, the carbon blow and the afterblow. Changes in chemical composition of the bath during progress of a 25-ton blow are shown diagrammatically in Figure 14-8.

The silicon blow commences as soon as the blast is let into the converter. The oxygen in the air unites directly with the iron to form iron oxide, which in turn combines with the silicon and manganese. These reactions and the net heat evolved can be shown by the following equations (unless otherwise stated, all heats of

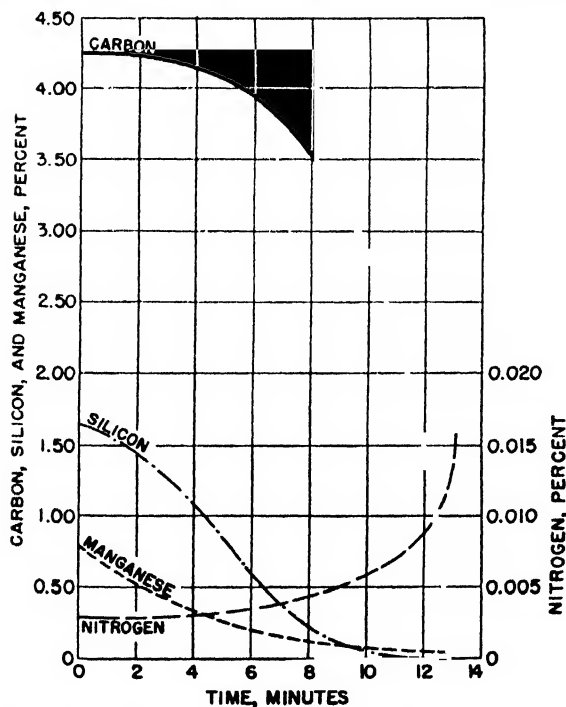
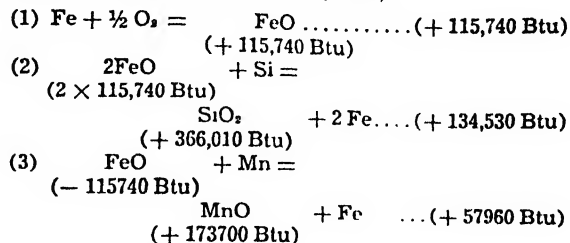
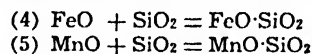


FIG. 14-8. Changes in chemical composition of the metal in the bath of a 25-ton acid Bessemer converter during the progress of a blow.

reactions have been taken from "The Thermochemistry of the Chemical Substances," by F. Russell Bichowski and Frederick D. Rossini, published by Reinhold Publishing Corporation, New York, 1936):

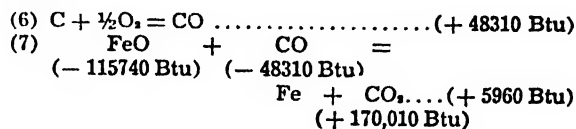


With the oxidation of silicon and manganese, a slag is formed immediately by the combination of silica with the excess iron oxide and the manganese oxide, the reactions being illustrated generally as follows:

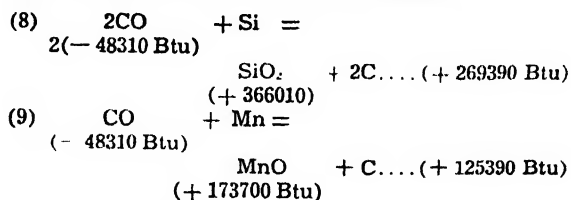


During this period of the blow, which lasts usually about four to five minutes, a short, stubby, transparent flame extends from the mouth of the converter. Pellets of molten iron and slag can be seen ejected by the blast. The converter gases evolved during this period are mainly nitrogen, with some carbon dioxide and traces of oxygen and hydrogen. The hydrogen results from the dissociation of water vapor in the blast.

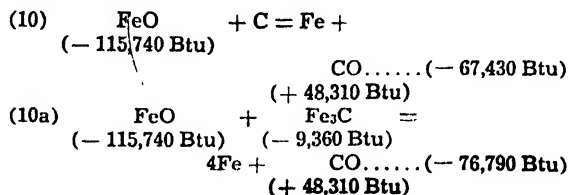
The slag formed is intimately mixed with the metal as a result of the violent agitation of the bath, and, being a mixture of silicates, can either take on or give off some iron oxide. Thus, it also may help to oxidize the metalloids as long as they are present, or decoxidize the metal when the metalloids are absent. Toward the end of the period carbon may be oxidized either directly by the oxygen of the blast or by the dissolved iron oxide in the metal.



Such reactions account for the presence of both CO_2 and CO in converter gas during the first part of the blow. At the beginning of the blow CO is subject to reduction by both silicon and manganese, especially if the iron contains a high concentration of these elements.



As the temperature rises, the second period or carbon blow commences. The elapsed time for completion of this period is six to seven minutes. The reactions occurring now are as follows:



These reactions, in conjunction with reaction (6), rapidly burn out the remaining carbon. According to the law involving heats of formation, these reactions should not take place, but since one of the products is a gas and the net heat of reactions (1) and (10) is equal to the heat of formation of CO, they progress rapidly almost to completion under the conditions of blowing. Again, endothermic reactions may be reversed at high temperatures. This fact suggests the possibility that with "hot iron," that is, iron high in silicon, which is also initially at a very high temperature, it might be possible even to eliminate the carbon before the silicon and manganese could be oxidized, and the testimony of the older and more experienced operators of converters is to the effect that just such a result as this has often occurred when the conditions noted were present. Furthermore, in the elimination of the carbon, the law of mass action becomes prominent again, for with the elimination of the silicon and manganese, the active mass of the ferrous oxide rapidly increases until equilibrium is established with carbon. Comparatively little additional heat is generated in the bath during this period. The net heat generated is the difference between the heat of formation of FeO (115740 Btu) and that absorbed in reaction (10), (67430 Btu), or 48310 Btu which is the heat of formation of CO. The carbon reaction occurs concurrently with the phenomenon commonly spoken of as the boil. There is, during this period, very little, if any, iron oxidized above that required to eliminate the carbon, so each volume or molecule of oxygen in the blast will produce two volumes or molecules of CO. The converter gases, therefore, show a high content of CO, very little CO₂, and a marked decrease in N₂. But as the carbon is eliminated, the amount of oxygen in the metal increases, until the content of the former reaches a point between 0.01 and 0.02 per cent, when the oxygen content may increase rapidly without further oxidation of the carbon.

Phosphorus and Sulphur Control—In the acid process, the phosphorus and sulphur suffer no oxidation from the action of iron oxide. Assuming that the loss in weight in the bath from elimination of the metalloids, mechanical ejection by the blast and oxidation of iron and retention as iron oxide in the slag was 10 per cent, if an iron contained 0.030 per cent sulphur and 0.075 per cent phosphorus, the metal at the termination of the blow would contain approximately 0.033 per cent sulphur and 0.083 per cent phosphorus.

Iron ores with a low phosphorus content (less than 0.045 per cent) are required in the production of blast-furnace iron that can be converted into low-phosphorus steel by the acid Bessemer process without employing a separate dephosphorizing process.

Dephosphorization in the ladle has been used successfully for a number of years in the production of acid Bessemer steel with less than 0.050 per cent phosphorus. This is accomplished by adding a mixture of lime, iron oxides and flux, all in the solid state, to the blown metal as it flows into the ladle. Another method for phosphorus removal consists of the well-known duplex practice in which a tilting basic open-hearth furnace is employed. The technical possibilities of dephosphorization in the Bessemer process is a field for further investigation.

Sulphur in coke is also an important factor in Bessemer steel production as this element is not removed in the acid Bessemer process. If higher sulphur coke is used in Bessemer iron production, it is necessary either to remove sufficient sulphur in the blast furnace or resort to other treatments. In the blast furnace, limestone in the charge is adjusted with the burden for sulphur removal. Ladle treatments have been developed

in which fused sodium carbonate or sodium hydroxide has been used successfully, particularly with cupola iron. In the duplex process, sulphur may be removed to some extent in the basic open-hearth furnace. In Europe, a certain amount of sulphur is removed in the basic Bessemer process.

PROCESS AND QUALITY CONTROL

End Point Evaluation—Although visual characteristics continue to be the most widely used means of controlling the Bessemer blow and determining the end-point characteristics, several instruments have been investigated in an attempt to obtain better control. The spectroscope, one of the first instruments investigated, has been used with reasonable success. The amount of natural light and other conditions in the Bessemer plant may interfere with and limit its applications. It is necessary, depending on the grade of steel being produced, to select a time interval after changes in the spectrum are observed in order to turn down the vessel at the proper carbon content.

Automatic recording of the radiation characteristics of the flame as measured by a photocell also has been used to aid in control of the Bessemer blow. As the flame drops at the end of a blow, the curve drawn by the recorder indicates an arrest which is taken as the end point. Each type of steel requires a specific afterblow, measured in seconds, which is determined by statistical studies of variations in yield and quality with time of afterblow, beyond this end point.

One of the most important features of end-point control is the elimination of overblowing which results in the absorption of excessive amounts of oxygen and nitrogen. This not only affects the efficiency of deoxidation, but also influences behavior of the product during processing in the rolling mills.

Temperature Control—The production of steel by the converter process requires careful temperature control in order to insure satisfactory practice. If the finished steel is too low in temperature, ladle skulling and pouring nozzle difficulties will be encountered which are undesirable from the viewpoint of steel quality. Steel with a high temperature not only contains excessive quantities of nitrogen and oxygen, but also is unsatisfactory in other respects. The Bessemer blower is, therefore, confronted with the problem of producing steel at a satisfactory temperature which can be determined conveniently at the present time only with an optical pyrometer as the blown metal is poured into the ladle or during teeming. Many attempts have been made to develop a satisfactory device to measure temperature of the molten charge during the Bessemer blow, but a satisfactory technique has not been developed. The blower still must continue to depend upon the characteristics of the flame.

A photronic cell located in the wind-box has been investigated and the possibilities of making accurate temperature determinations by this method are promising, but additional experimental work will be required to develop a satisfactory technique.

Chemical composition of the iron is a fundamental consideration with respect to temperature control, particularly with regard to silicon. The oxidation of silicon is an exothermic chemical reaction and data showing the actual silicon content in the iron should be available to the blower. If the silicon content is high, cold metal (scrap) or pig iron may be added to the vessel to eliminate the necessity for steaming. If the silicon content is too low, it may be necessary to side blow in order to increase the temperature of the metal. During the blow, if the temperature is excessive, all of the silicon may

not be removed before carbon oxidation predominates.

A cold ladle or vessel and the amount of ferroalloy addition at the end of the blow are among the factors which must be considered by the blower in producing steel with a satisfactory temperature. The blower, therefore, must take into consideration the many variables which affect temperature and, by judging character of the flame, produce a satisfactory quality steel. The accurate determination of temperature during the blow with a radiation instrument, consequently, would be of considerable assistance in the economic production of Bessemer steel.

Oxygen-Enriched Blast—Extensive investigations of the use of an oxygen-enriched blast have been carried out with the following results:

- (1) Shorter blowing time resulted.
- (2) Greater utilization of cold iron and scrap was achieved.
- (3) With a blast containing about 30 per cent oxygen, the bottom and vessel linings were not abnormally worn in a limited number of heats.
- (4) Steel quality was similar to regular Bessemer heats.
- (5) Flame characteristics were readily distinguishable by blowers and satisfactory teeming temperatures were obtained.

Blowing Time and Quality—One of the important economic factors in the production of Bessemer steel is that a blow usually lasts for a period of only 10 to 15 minutes. However, the time required to make a certain quantity of steel depends on size of the vessel, and converters in this country are relatively small. Due to the fact that the time required to make an acid Bessemer blow is relatively short, the problem of controlling quality of the product differs somewhat from open-hearth practice. Speed of the operation is so rapid that the necessity for certain control measures differs from those required by the open-hearth melter. The Bessemer process, to a marked degree, is automatic in its operation.

The composition of the steel cannot be regulated satisfactorily by stopping the blow at an intermediate point due to the speed of reactions and other conditions. It is possible to blow a Bessemer heat to any selected carbon level, but a product more uniform in chemical composition is obtained by blowing to the drop of the flame and adding carbon and manganese as required. High-carbon blown metal is sometimes produced in the duplex proc-

ess as further refining and adjustment in chemical composition are made in the open-hearth furnace. At the first indication of a drop of the flame and other changes in appearance of the flame, the converter may be turned down "young," and the carbon content will be approximately 0.10 per cent. However, most of the acid Bessemer steel is blown "full" to insure uniform composition. The vessel is turned down as height of the flame continues to decrease and certain changes in appearance are observed. Full-blown heats contain approximately 0.04 per cent carbon. If the manganese content of the metal at the beginning of the blow is too high, a fluid slag which contributes to "slopping" is produced. Therefore, the ratio of silicon to manganese must be controlled. Silicon already has been described as a source of heat and therefore an important factor in temperature control during the blow.

DEOXIDATION OF ACID BESSEMER STEEL

General—In the Bessemer process, characteristics of the flame at the end of the blow and the deoxidation practice employed are fundamental factors which control quality of the product. The additions required to produce various grades of acid Bessemer steel are incorporated in the deoxidation practice and may be added to the vessel, ladle, or mold.

In the deoxidation of acid Bessemer steel, the oxygen content of the blown metal is one of the significant factors involved. The iron oxide content is also related to the nitrogen content in that both are increased by higher temperatures and overblowing. In the production of killed acid Bessemer steels, the deoxidation practice also must take into consideration the fixation of nitrogen. Therefore, for proper deoxidation, it is essential to control carefully the temperature and end point of the Bessemer blow.

Rimmed and capped acid Bessemer steels usually are blown fully and finished with a low carbon content. These steels are not completely deoxidized so that a strong evolution of gas occurs during solidification in the mold. Sufficient deoxidizer may be added, either in the ladle or mold, to control action in the mold (Table 14—III). In capped steels, the period of gas evolution in the mold is further controlled by using a steel cap which chills and solidifies the top of the ingot. Steel of this type usually contains about 0.07 per cent carbon, 0.45 per cent manganese, 0.08 per cent phosphorus, 0.025 per cent sulphur and 0.005 per cent silicon. In the manufacture of killed acid Bessemer steel, strong deoxidizers, includ-

Table 14—III. Typical Compositions of Deoxidizers (Per Cent)

	Fe	C	Mn	S	P	Si	Ash
Ferromanganese	11.95	6.50	80.40	Trace	0.160	1.00
Ferrosilicon (50%)....	48.4	0.15	0.31	0.008	0.035	50.0
Ferrosilicon (85%)....	85.60	...
Pig or Molten Iron....	92.78	4.25	0.56	0.032	0.078	1.33
Anthracite Coal	85.50	4.50
	Fe	Al	Cu	Ni	Si
Aluminum	2.24	90.19	4.14	0.66	2.28		

ing carbon, ferrosilicon and aluminum are added to the molten steel at the end of the blow. Ingots with a dense, homogenous structure, relatively free from porosity or blowholes, are produced from killed steel. Steel of this type may contain 0.10 per cent or more carbon, 0.35–1.25 per cent manganese, 0.10–0.30 per cent silicon, 0.08 per cent phosphorus and 0.025 per cent sulphur. Alloys also may be added if desired. The factors responsible for the superior quality of deoxidized as compared with rimmed or capped acid Bessemer steel do not depend entirely on the fact that a killed steel is involved, but more precisely, are related to the method of killing the steel.

Carbon Deoxidation—The oxygen content of rimmed or capped acid Bessemer steels is high and must, to a certain degree, be evaluated by behavior of metal in the mold. These steels are generally low in carbon, and carbon deoxidation is not an important factor in the production of "open" steels.

The carbon content of killed acid Bessemer steel varies from 0.10 per cent to any higher level desired, depending upon the grade of steel. A sufficient amount of deoxidizer is added to produce a steel which has no action in the mold. In order to meet carbon requirements and properly deoxidize the steel, molten blast-furnace iron containing about 4 per cent carbon is added to the vessel at the end of the blow. After the addition of hot metal, a reaction takes place for several minutes in the vessel which results in the formation of carbon monoxide gas. This is similar in some respects to "blocking" an open-hearth heat (described in Chapter 15), except that the mixing of blown metal and hot metal is almost instantaneous. The carbon monoxide burns at the mouth of the converter. Removal of oxygen as a gas not only deoxidizes the steel but removes oxygen immediately instead of forming a solid metallic inclusion which must rise to the surface of the steel. Steel made by this process is distinctly different from regular acid Bessemer steel. The steel contains fewer non-metallic inclusions than if silicon or aluminum had been added before the carbon reaction.

In the use of hot metal for deoxidation of acid Bessemer steel, provisions must be made for accurate weighing of the addition. Other means may be used for carbon deoxidation in which solid materials are added, but the simplest and most thorough method at the time this is written is the use of hot metal.

Manganese and Silicon Additions—In the manufacture of killed acid Bessemer steels manganese and silicon may be added after carbon deoxidation either to the vessel or ladle in sufficient quantity to provide the amount desired in the finished steel. Additions to the

vessel are preferred and are accomplished by the use of a chute, as shown in Figure 14–9. Usually, 75 per cent ferromanganese and 50 per cent ferrosilicon are employed. Special types of deoxidizers containing silicon and/or manganese with aluminum have been used successfully. In the manufacture of rimmed or capped acid Bessemer steels, silicon is seldom used, and manganese is added to the ladle for fixation of sulphur and to improve rolling characteristics. The efficiencies of the various additions are similar to open hearth practice at the same carbon level.

Although aluminum is added to steel primarily for deoxidation purposes, it also is used for nitrogen fixation in killed acid Bessemer steels. The fixation of nitrogen with aluminum or other elements imparts superior toughness and improved resistance to notch sensitivity. The proper use of aluminum or its equivalent is essential in the production of a satisfactory killed acid Bessemer steel.

ECONOMIC CONSIDERATIONS

Investment Costs—The cost of building an acid Bessemer plant is appreciably less than an open-hearth plant of equivalent capacity and the difference in cost depends, to some extent, upon the size of individual units. However, in the operation of an acid Bessemer plant, greater blast-furnace and by-product coke capacity is required as only a small amount of scrap is used in the process. Therefore, the investment in blast-furnace and by-product coke facilities for the acid Bessemer plant would be greater than for an open-hearth plant and must be considered as part of the investment costs. This probably was a factor which, to some extent, was responsible for the shift from acid Bessemer to basic open-hearth steel production during the years between about 1910 and the present.

With proper facilities in an integrated plant, consisting of acid Bessemer converters and tilting or stationary basic open-hearth furnaces, the cost of producing Bessemer ingots over a period of years should be less than open-hearth ingots. When a shortage of scrap exists and the price is high, acid Bessemer facilities not only have an economic advantage, but are in a better position to meet market demands for steel products. This latter feature is of considerable importance in time of a national emergency.

Scrap Market Conditions—The ratio of acid Bessemer to basic open-hearth steel production is influenced by the availability of steel scrap. The Bessemer process uses about 10 per cent scrap. The duplex open-hearth process utilizes very little scrap while the stationary open-hearth process ordinarily employs 35 per cent to 60 per cent. Although the quantity of scrap used in the Bessemer process is small, this quantity may be increased appreciably in the future to approximately 25 per cent by oxygen-enrichment of the blast. Further, the use of scrap in the blast furnace may be considered in the Bessemer situation.

The Bessemer process produces more scrap than it consumes. The crop ends in the blooming mill alone produce about 15 per cent, and additional scrap is produced in further processing of the material. The stationary open-hearth process produces a similar amount of scrap, but constantly uses more than it produces. The deficit is supplied by scrap originating from the Bessemer process and from sources outside the plant.

The Bessemer process has certain economic advantages when scrap is scarce and costly, but there are periods in our economy when scrap is plentiful and cheap. Therefore, the economic pressure which exists over a period of years will be a controlling factor which

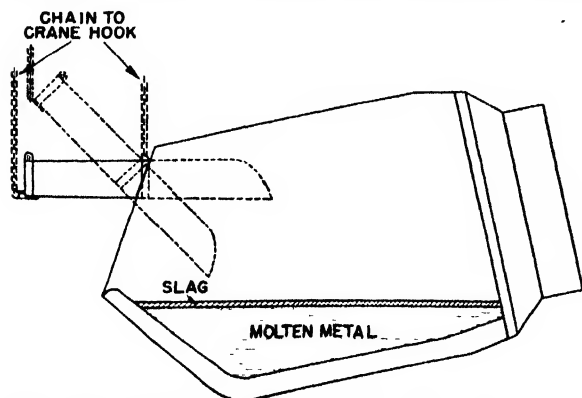


FIG. 14–9. Diagram illustrating the use of a crane-operated chute for making additions of solid materials to a Bessemer converter.

influences the acid Bessemer steelmaking production of the nation.

Metal Losses—Metallic yield in the acid Bessemer process is comparable with stationary open-hearth practices and is influenced by metalloid losses, oxidation of iron, slopping and other operation characteristics. Depending upon facilities and operating conditions, the performance of acid Bessemer plants in terms of metal yield has been similar to that of open-hearth plants.

Possible Future Trends—The future design of converters probably will be directed to the construction of vessels with a larger capacity in order to reduce the cost of acid Bessemer steel. A number of large bottom-blown (basic) converters with capacities exceeding 50 tons were in operation in Germany during World War II. At present, the size of a heat may be increased by pouring several acid Bessemer blows in one ladle and this practice has some of the advantages which are achieved in open-hearth practice with larger furnace

capacities. In order for rimmed or capped converter steel to have properties similar to open-hearth steel, it is not only necessary to reduce the nitrogen content, but also the phosphorus content of the finished steel. In the manufacture of seamless pipe, the phosphorus content of killed converter steel imparts to the steel strength properties which are achieved in open-hearth steel only at higher levels of carbon, or by deliberate increase of phosphorus and/or nitrogen content.

Changes in converter design require thorough evaluation with respect to operating characteristics. The problems encountered in open-hearth design are of an entirely different nature, and if a similar effort had been directed in this country to converter design and practice, considerably more progress would have resulted. Present trends in the use of oxygen and compressed air in the bath of open-hearth furnaces indicate clearly the future possibilities of a more widespread use of the fundamental principles of the pneumatic processes.

SECTION 3

THE BASIC BESSEMER OR THOMAS PROCESS

Historical—The bottom-blown basic pneumatic process known by the several names of the **Thomas, Thomas-Gilchrist** or **basic Bessemer process**, was patented in 1879 by Sidney G. Thomas in England. The process (never adopted in the United States) developed much more rapidly on the Continent than in Great Britain and, in 1890, Continental production was over 2 million tons as compared with 400,000 tons made in Great Britain.

The simultaneous development of the basic open-hearth process resulted in a decline of production of steel by the bottom-blown basic pneumatic process in Europe and, by 1904, production of basic open-hearth steel there exceeded that of basic pneumatic steel. From 1910 on, the bottom-blown basic pneumatic process declined continuously until World War II.

Composition of Pig Iron for Basic Process—The bottom-blown basic pneumatic process requires the use of a basic lining in the converter, and the use of lime or limestone as a slag-making material. The basic slag makes the process capable of removing phosphorus (and sulphur) from the blast-furnace iron. Thus, iron can be produced from iron ores of lower grade, using lower-grade coke as fuel, than would be permissible in production of iron for the acid Bessemer process which cannot remove phosphorus or sulphur and requires irons with a low content of these two elements.

A typical blast-furnace iron for the bottom-blown basic pneumatic process contains from 1.4 to 2.0 per cent of phosphorus and approximately 0.5 per cent of silicon. Iron ore and other raw materials of the type required to produce iron of this composition in the blast furnace, or to give a molten product of this composition when pig iron is melted in the cupola, have been and still are available abroad in rather large quantities.

Since some phosphorus and sulphur are removed in the bottom-blown basic pneumatic process, the chemical composition and properties of steels produced by this method more closely approach the composition and properties of basic open-hearth steels of similar grade than do comparable steels made by the acid Bessemer process. However, the oxygen content of the bottom-blown basic pneumatic steels is generally higher and the nitrogen content is definitely higher than that of basic open-hearth steel. For these reasons, the properties of steels made by the basic Bessemer (Thomas)

process, while more similar to basic open-hearth steels than are acid Bessemer steels, are still inadequate for certain applications because of their higher strength, lower ductility and susceptibility to strain aging.

PRINCIPAL FACILITIES

The equipment for the bottom-blown basic pneumatic process is very similar to that described for the acid Bessemer process in Section 2. Consequently, the following description will mention largely differences between the types of equipment used for the two processes. A typical Thomas steel plant has six 25-ton vessels which can produce about 120 tons of steel per hour or 3000 tons per day.

Mixer—Hot-metal mixers for the Thomas plants are of the same general size and type as those already described in connection with acid Bessemer steel plants. Some so-called "active" mixers with regenerators are used to keep the iron hotter, and through the addition of lime some refining also is carried out prior to charging the metal in the converters. Many mixers are lined with magnesite brick rather than with siliceous refractories.

Blowing Equipment—There are no marked differences in blowing equipment for the bottom-blown basic pneumatic process from that already described for the acid Bessemer process. The normal air-blast pressure in the basic process is 35 pounds per square inch, which is somewhat higher than that employed for the acid Bessemer process in the United States.

THE CONVERTER OR VESSEL

The size of bottom-blown basic converters varies from 25 tons to 60 tons capacity. The design of vessels is similar to that of the acid Bessemer converters used in America. A large number of basic vessels are concentric rather than eccentric, in relation to the position of the opening in the nose. To conserve heat, the nose opening in some basic vessels is quite small. Further, the bottom design and the lining, of course, are different. Almost all of the bottom-blown basic pneumatic converters are lined at present with a mixture of tar and burned dolomite rammed into place in the vessel.

Bottom Design—The bottom of the basic converters used in most plants contains about 200 holes, each $\frac{1}{2}$ inch in diameter. Separate tuyere brick are not used in

these bottoms; instead the holes are formed by ramming the refractory bottom material around wooden plugs. The bottom is then heated in an intermittent-type oven for a period of six days, after which the wooden plugs, which have been converted to soft charcoal, are drilled out. Bottoms with tangentially inclined holes have been tried over a long period but have not been proven to have any appreciable effect in improving operations.

Lining and Bottom Life—The lower part of the lining of a bottom-blown basic converter requires replacement after about 200 blows, on the average. The remainder of the lining, with the exception of the bottom, withstands about 400 blows. Bottom life averages about 40 blows. Dolomite consumption has been reported as about 22 pounds per ton of steel produced.

SEQUENCE OF OPERATIONS

The steps in making a heat of steel in a bottom-blown basic converter vary largely because of recent innovations and changes introduced in an effort to consistently produce steels of low nitrogen content. The general sequence in a typical British plant is as follows: the blast-furnace iron is partially desulphurized between the blast furnace and the mixer or between the mixer and the converter by additions of soda ash. Slag produced by the desulphurization operation is carefully skimmed off before charging the iron into the converter. With the vessel in the horizontal position, the necessary burnt lime (averaging 300 pounds per ton of iron) is added by an overhead chute. The vessel is then turned down past the horizontal position and a weighed molten iron charge is added. Blowing is started with a blast pressure of 30 pounds per square inch. The time of addition of scrap, required to control the temperature of the blow, varies with the type of scrap available. With bloom-end scrap the addition is made two or three minutes before the drop of the flame but, with heavy ingot scrap, the addition is made early in the blow to allow for the complete melting of the scrap.

The total duration of the blow depends upon time of additions and other operating factors. The end point is determined by visual inspection of the flame, confirmed by the blower's judgment of a fractured sample of steel taken from the vessel. After blowing is completed, most of the slag is removed by tilting the vessel, following which the necessary amount of ferromanganese is added to the steel in hot lump form. The residual slag is retained in the vessel by the formation of a lime dam in the mouth of the vessel while the steel is poured into the ladle held underneath the vessel by the casting car. For a 40-ton converter, the total duration of blow is 15 to 16 minutes, including an after blow of 3 to 5 minutes. It should be noted that the after blow in the basic process is relatively long as compared with the acid Bessemer process; this is necessary to accomplish removal of phosphorus from the bath. The end point of a basic Bessemer blow is very similar to that of the acid Bessemer process, but occurs somewhat earlier in the blow because of the lower silicon content of the iron used in the basic process which results in a shorter silicon blow. A 40-ton vessel has been reported to use about 12,000 cubic feet of air per ton of steel produced.

CHEMISTRY OF THE THOMAS PROCESS

Oxidation Reactions—The major chemical differences between the bottom-blown acid and basic pneumatic processes lies in the chemical reactions involving removal of phosphorus (and to some extent, sulphur) from the iron. The effective removal of these elements depends almost entirely on slag-metal reactions which are almost absent in the acid Bessemer process. Oxida-

tion of carbon, manganese and silicon occurs in essentially the same manner as was described for the acid Bessemer process.

The removal of phosphorus from the metal involves first its oxidation to phosphoric acid anhydride (P_2O_5) which then combines with lime (CaO) in the slag to form the chemical compound ($4CaO \cdot P_2O_5$). The removal of sulphur depends upon the migration of ferrous sulphide (FeS) into the slag where it reacts with lime (CaO) to form calcium sulphide (CaS).

The removal of both phosphorus and sulphur in the basic pneumatic process depends upon the concentration of lime in the slag; however, removal of phosphorus is favored by a high content of iron oxide in the slag and oxygen in the metal, while sulphur removal is favored by the reverse conditions. Thus, only a small amount of sulphur is removed in the basic pneumatic process unless slags with a high lime content are formed. Further, because of the nature of the dephosphorization reaction, it occurs largely after the removal of the majority of the carbon, in the after-blow period. Figure 14-10 illustrates the course of the oxidizing reactions

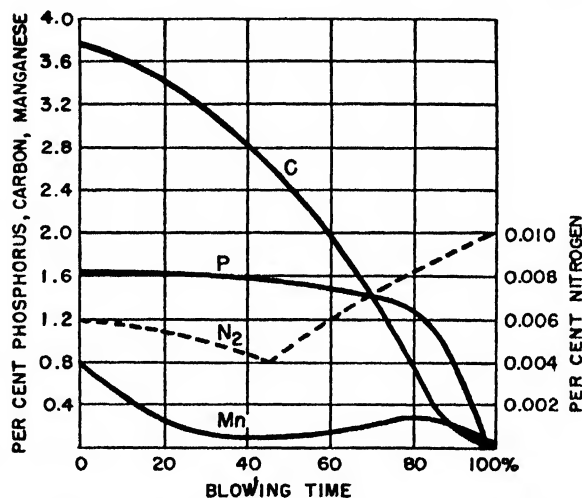


FIG. 14-10. Changes in chemical composition of the metal in the bath of a bottom-blown basic-lined pneumatic converter during a typical Thomas blow (after Carney).

during a typical Thomas blow. As in most basic steel-making processes, an undesired reversion of manganese and phosphorus from slag to metal is more probable if high finishing temperatures are attained.

Thermal Requirements—In the bottom-blown basic pneumatic process, lime is added with the hot metal. This added cold material, as well as the greater slag volume produced, requires additional heat as compared with the acid process. In the basic pneumatic process, iron of higher silicon content cannot be used to obtain this additional heat, since the more silicon the iron contains, the more lime will be required to flux the oxidized silicon (silica, SiO_2) both to prevent the silica from attacking the lining and to keep the slag basic enough for effective phosphorus and sulphur removal. The higher the silicon content of the iron, the greater will be the slag volume produced. Fortunately, in the bottom-blown basic converter, a portion of the additional heat required is obtained from the oxidation of phosphorus in the iron. For this reason, a proper thermal balance in the bottom-blown basic converter makes the relative percentages of silicon and phosphorus in the iron critical. A typical blast-furnace iron for use in the Thomas

process at present contains 0.2 to 0.4 per cent silicon, 0.6 to 1.0 per cent manganese, 1.5 to 2.0 per cent phosphorus and 0.03 to 0.05 per cent sulphur. The phosphoric acid anhydride (P_2O_5) content of the final slag produced in bottom-blowing iron of the above composition in a basic converter, using 300 pounds of lime per ton of steel, is about 16 to 18 per cent, with a slag volume of about 20 to 22 per cent. Slags of this composition make desirable fertilizers and they are processed and sold for this purpose, thus aiding the economics of the process.

Temperature Control—During a bottom-blown basic converter blow, the temperature normally rises during the oxidation of silicon, remains fairly constant during the carbon blow, and rises rapidly during the after blow. In some respects control of temperature in the bottom-blown basic converter is more important than in the acid Bessemer vessel because of the need for effective control of phosphorus content of the finished steel. The methods used for temperature control during a blow are very similar for both the basic and acid processes. The most important factors relating to temperatures in the vessel are the as-charged temperature and composition of the iron being blown and the blowing time. Common methods for controlling temperature during blowing are the use of side blowing to increase temperature and the use of steam and scrap to decrease temperature. The photocell and immersion thermocouples have been used with some success in quite a few plants for controlling the end point and for accurately recording the final bath temperature. The main reason for this closer control of temperature is the concern over residual nitrogen and the desire to produce low-nitrogen steels. Residual nitrogen contents are markedly affected by the bath temperature.

Deoxidation Reactions—There are no major differences in the deoxidation reactions between the bottom-blown acid and basic pneumatic processes. A fair number of heats produced in the bottom-blown basic converter in Europe are fully killed. In the United States considerable tonnages of killed acid Bessemer steel are produced. Deoxidation with aluminum is practiced more generally abroad than in the United States. Much of the deoxidation with aluminum in Europe is accomplished by ladle additions compared with mold additions in this country.

Nitrogen Control—Since bottom-blown basic pneumatic steels have sulphur and phosphorus contents similar to or slightly higher than those of basic open-hearth steels, many attempts have been made to develop practices that would make it possible to reduce the nitrogen content of these steels to levels approaching that normally attained in basic open-hearth steels (0.003 to 0.005 per cent nitrogen). Some of the factors which influence the nitrogen content of the pneumatic steels were covered in Section 2 where the acid Bessemer process was discussed. In general, the following items have been found to be important in the commercial production of low-nitrogen bottom-blown pneumatic steels:

- (a) The blow must be worked at a low temperature.
- (b) A shallow metal bath is desirable to limit time of contact between gases and metal.
- (c) High nitrogen content of the blowing gases, especially during the latter part of the blow, should be lowered when possible by use of oxygen-enriched air, mixtures of oxygen and steam, mixtures of oxygen and CO_2 , or other means.

(d) Pressure of the blowing gases should be kept as low as possible, compatible with reasonable blowing time.

(e) Irregularity in blast-furnace operations is believed to promote higher nitrogen content of pig iron, which complicates the production of low-nitrogen steels in the basic bottom-blown converter.

The most frequently used methods for producing low-nitrogen steels in the basic bottom-blown converter have been based upon shortening the after blow and also upon cooling the bath at this point. In one method, this is accomplished by adding ore or roll scale (up to 2.5 per cent) a little before the carbon flame drops to effect oxidation without the introduction of nitrogen while simultaneously cooling the bath. This method may contribute to difficulties arising from low finishing temperatures that cause ladle-skull formation and pouring difficulties due to cold steel.

A second method obviates some of the pouring difficulties while speeding up the process. Best suited to 50 to 60-ton converters, this method consists of double-blowing the metal. Half of the total iron and all of the lime is charged first and blown down to about 0.1 per cent of phosphorus. The shallow bath permits high-pressure blowing without excessive spitting. The balance of the charge is then added and blowing continued, again at high pressures made possible by the diluting effect of the first-blown metal. The total blowing time can be reduced from 16 to 17 minutes to 11 to 12 minutes by this method which is said to increase average size of charge, increase yield and still produce steel lower in phosphorus and nitrogen content than steels produced by ordinary blowing.

A third means for reducing the nitrogen content of Thomas steel involves a modification of the converter so that the blast is introduced through the side of the vessel at a point intermediate between the bottom of the vessel and the level of the bath. The aim of this modification is to reduce the length of travel of the blast through the bath and thus minimize the time for absorption of nitrogen without sacrificing the other characteristics of bottom blowing. It has been observed that nitrogen absorption in acid surface blowing practices is quite low. Consequently, it has been hypothesized that the degree of nitrogen absorption, other things being equal, would be related to the length of travel or time of contact of the air blast with the metal. Modification of the Thomas converter in this manner does not influence the remaining operations and the typical course of the refining remains the same as that shown in Figure 14—10, with the exception that lower nitrogen values are achieved.

Considerable quantities of low-nitrogen steel have been produced by the basic bottom-blown pneumatic process for wire, ship plates, deep-drawing steels, high-carbon high-tensile steels, and killed steels for various purposes. These products require less deoxidation than the material produced by ordinary blowing, and their properties are quite similar to basic open-hearth steels of comparable grades.

Control of End Point—The discussion of the control of end point in Section 2 dealing with the acid Bessemer process applies equally well to the basic bottom-blown process. The spectroscope is used very little in Europe, the most common methods of control being visual observation of the flame by the blower, the use of fractured samples, and the use of the photocell.

SECTION 4

THE SURFACE-SIDE-BLOWN ACID PROCESS

The side-blown acid pneumatic converter is sometimes referred to as a surface-blown converter, or as the Tropenas converter. The chemical reactions which occur in the side-blown acid converter are similar to those occurring in the bottom-blown acid converter; the major difference is the mechanism by which these reactions take place. There have been numerous variations of design of converters employing the side-blowing technique, but all are characterized by having all of the tuyeres above the liquid level of the bath and entering through the side of the vessel.

In bottom-blown converters, the exposure of some of the tuyeres above the liquid level by tilting the converter while blowing increases the temperature inside the vessel, as discussed in Section 2. With all of its tuyeres above liquid level in normal operating position of the vessel, the side-blown acid converter can produce much hotter steel from a similar iron charge than can the bottom-blown acid converter. Temperatures in excess of 3200 °F (1760 °C) can be attained if desired, blowing cupola-melted iron in lots as small as 1000 pounds. Its ability to process small heats at high temperatures and at frequent intervals that can keep up with pouring-floor schedules makes the side-blown acid converter useful to foundries making steel castings.

The quality of ore, coke and other raw materials required to produce iron for the side-blown acid process is similar to that required for the bottom-blown acid process; however, because additional heat is obtained from the oxidation of carbon to carbon monoxide inside the vessel, the silicon content of the iron can be somewhat lower.

The Converter and Its Auxiliaries—Since most side-blown vessels are used in foundries at present, hot-metal mixers are seldom available to provide a source of liquid iron for side-blown acid converters. Mixers, when used, are quite small. Generally, the cupola is employed to melt solid pigs of iron of the proper composition; the cupola can be tapped frequently to supply the side-blown converter with liquid iron as needed.

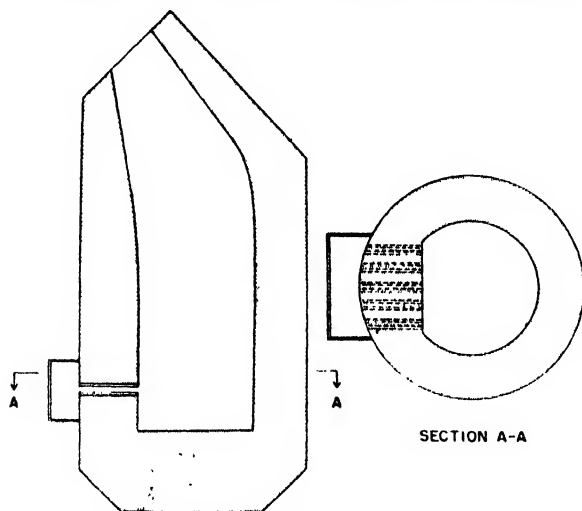


FIG. 14—11. Sectional drawing of a typical side-blown acid converter. While the cross-section of the interior of the type shown is D shaped, many side-blown acid converters are circular in interior cross-section.

Side blowing requires lower blast pressures and smaller volumes of air than bottom blowing; blast pressures between 4 and 10 pounds per square inch are common.

The majority of side-blown acid converters are of one-half to two-ton capacity, although a few larger vessels have been built. Figure 14—11 is a sectional drawing of a typical side-blown acid converter. The refractories used for lining are similar to those used in bottom-blown converters.

Sequence of Operations—While the chemical reactions of the side-blown acid process proceed at a slightly different rate and in a different manner than those of the bottom-blown process, the physical steps in making heats by both processes are quite similar. The end point is observed from the drop of the flame. The length of the flame is shorter than that for the bottom-blown vessel. In the acid processes, the addition of controlled amounts of ferrosilicon to the vessel during blowing to make available more potential heat units is practiced much more frequently in side blowing than in bottom blowing.

Oxidation Reactions—While the chemical reactions which occur in the side-blown acid pneumatic process are similar to those in the other pneumatic processes, the mechanisms by which these reactions take place are somewhat different. The elimination of carbon, manganese and silicon in the side-blown acid process may be considered as taking place in three stages:

(1) The blast of air impinges on the surface of the bath of molten iron and immediately forms a layer of iron oxide. Some silicon and manganese oxides are formed during this stage, but the initial slag consists mainly of iron oxide.

(2) From the time when the metal is completely covered with slag, the oxidation of manganese and silicon is accelerated; the reactions are considered to be between the slag and the metal and not between air and metal as in the bottom-blown converter. During this period, there is a rapid rise in temperature of the bath because of the exothermic reactions involved. Ferrosilicon can be, and is, added before and during the blow, these additions being governed by vessel temperature, metal composition, and metal temperature. Larger additions of ferrosilicon are made when low-carbon metal (2 to 3 per cent carbon) is being blown. The higher silicon content of the bath provided by ferrosilicon additions functions as a kindling agent, and during the oxidation of silicon the temperature of the bath increases to about 2640 °F; at this temperature the silicon content of the bath is about 0.6 to 0.7 per cent.

(3) After the silicon content has been lowered to the above figure, the oxidation of carbon increases to a very rapid rate and the temperature of the bath increases considerably. The oxidation of carbon in the metal by FeO or other oxides in the slag produces carbon monoxide, which may burn to carbon dioxide in the presence of excess air within the vessel. This represents an essential difference between the side-blown and bottom-blown processes, insofar as considerably more heat per unit of carbon in the bath is generated in side-blown practice.

Toward the end of the carbon-oxidation stage, when the bath is attaining its final temperature of 3000 to 3100 °F, other reactions take place which result in the

reduction of SiO_2 and MnO by carbon. It has been established that, under the above conditions of slag and metal, the silicon content of the final metal after the additions of deoxidizing agents and alloys is always greater than can be accounted for by the silicon content of the additions, thus indicating some reduction of silica in the slag to silicon which enters the metal.

Temperature Control—The methods of temperature control used in side blowing are quite similar to those already discussed for bottom blowing. Ferrosilicon is frequently added to the cupola iron during blowing to raise the temperature. Accurate temperature control is vital to the production of quality steel, but apparently is not so critical in regard to the final nitrogen content of the steel as it is with bottom blowing. Little trouble with excess nitrogen absorption in the side-blowing process has been reported, even with steel temperatures up to 3100°F .

Deoxidation Reactions—Deoxidation reactions of side-blown acid pneumatic steels are similar to those in finishing bottom-blown acid heats. However, since the majority of the side-blown steel is used for castings, these steels are fully killed, using as much as four pounds of aluminum per ton of steel.

Nitrogen Control—A major part of the side-blown acid steels are produced with a relatively low nitrogen content (0.003 to 0.008 per cent). Because of the short time of contact of gases and metal, nitrogen control is not as difficult in side blowing as it is in bottom blowing. The use of low finishing temperatures or ore additions is not practiced to any great extent. However, some work has been done on the use of oxygen and oxygen-enriched air for blowing to reduce the nitrogen content to even lower levels (0.002 to 0.003 per cent). These latter practices are not common at present.

Control of the End Point—End-point control in the side-blown acid pneumatic process is very similar to the control exercised in the bottom-blown processes, even though the size of the flame is quite different. The end point is usually determined visually by the drop of the flame.

Oxygen and Oxygen-Enriched Air for Blast—Recent experiments using oxygen-enriched air and 98 to 100 per cent oxygen for blowing in the side-blown acid process showed that blowing time was shortened and the final bath temperature was increased. Blowing losses increased after the proportion of oxygen in the blast exceeded 30 to 35 per cent.

SECTION 5

THE OXYGEN-BLOWN BASIC STEELMAKING PROCESSES

Variations of the pneumatic steelmaking processes which employ oxygen of high purity blown onto the surface of the bath in a basic-lined vessel may become of increasing importance in the future. Oxygen at high or relatively low pressures has been employed, blown through the mouth of the vessel vertically or at an angle to the surface of the bath.

One variation is the so-called Linz-Donawitz or L-D process, in which oxygen of high purity is blown at high velocity onto the surface of the bath in a basic-lined vessel by a vertical pipe or lance inserted through the mouth of a vessel. The name for this top-blown basic process is derived from the towns in Austria where these practices have been adopted on a commercial scale. At the present time, no commercial production of steel by top-blowing in an acid vessel is being conducted although the principle has been employed for desiliconizing pig iron to be used in the basic open-hearth process. The top-blown basic steelmaking process was developed in Austria to employ pig iron produced from local ores that are high in manganese and low in phosphorus; such iron is not entirely suitable for either the acid or basic bottom-blown pneumatic processes. Lack of a sufficient supply of scrap in Austria made it desirable to employ a steelmaking process requiring large percentages of hot metal as the prime source of metallics. Also, the majority of the steel produced was intended for applications requiring steels low in sulphur, phosphorus and nitrogen.

Mixers and Blowing Equipment—Conventional mixers are used in this process. At one plant, two 500-ton mixers service three 30-ton vessels; at another, one 1000-ton mixer services two 30-ton vessels. Since oxygen rather than air is used in the top-blown basic process, equipment capable of supplying high-purity oxygen under pressure is used in place of the normal air-blowing equipment. Oxygen is normally stored in tanks under a pressure of 375 pounds per square inch to allow for the intermittent operation of the vessels.

The Vessel—The design of the top-blown basic vessel is the simplest of all steelmaking furnaces. There are no tuyeres. Oxygen is supplied by a single water-cooled lance with a copper nozzle, suspended vertically so that

it can be raised and lowered through the mouth of the vessel to within the desired distance from the metal bath. The vessel itself, shown schematically in Figure 14—12, is refractory-lined and ladle-shaped with either a concentric or eccentric nose. The bottom of the vessel may or may not be detachable; both types are currently being used. The lining consists of a single course of magnesite brick covered with a rammed layer of tar and magnesite. Large bricks made from tar and dolomite have also been used. The lining must be almost completely replaced after 200 or more heats. Refractory consumption has been reported as between 20 and 30 pounds per ton of steel ingots. The vessels are supported on conventional trunnions that permit tilting for charge-

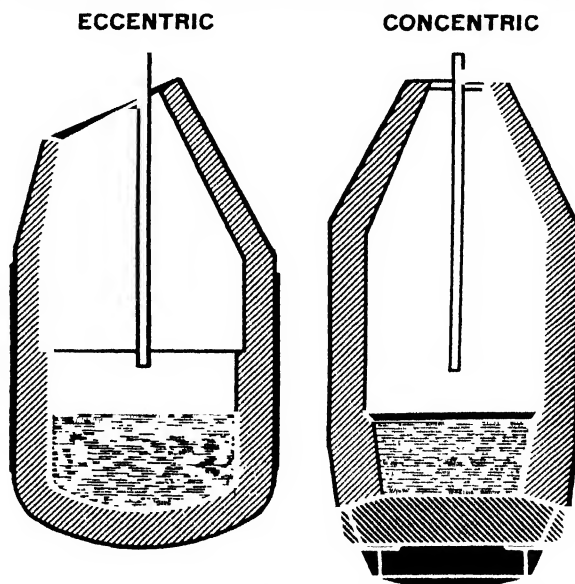


FIG. 14—12. Types of vessel used when blowing oxygen vertically downwards on to the metal through a water-cooled lance.

ing and pouring. The tops of the vessels have hoods for dust collection.

Sequence of Operations—A normal practice is to first charge scrap and hot metal through the nose of the tilted vessel. The vessel is then righted to vertical position and the oxygen lance is lowered from above into the nose of the vessel and to within 25 to 40 inches of the bath. The oxygen is turned on and limestone is charged from an overhead chute. Reactions occur immediately. During the process, lime and scrap are added as required for slag making and temperature control. The end point is indicated by the disappearance of the flame playing at the mouth of the vessel; the oxygen is then turned off as the lance is being withdrawn. Following this, the vessel is turned down to decant most of the slag. More slag is then pulled from the vessel by raking, after which a lime dam is formed at the mouth of the vessel to hold back the remaining slag when the steel is poured into a ladle. Blowing time in the vessels is 20 to 22 minutes with an oxygen input of 2500 cubic feet per minute at 130 pounds per square inch of pressure. Oxygen consumption is 2000 cubic feet per net ton of ingots. The time from charge to charge is between 30 to 35 minutes, including time for charging, changing pipes, steel sampling and testing. Figure 14—13

The end point is controlled visually by the drop of the flame at the mouth of the vessel at the end of the carbon blow.

The quality of the steels produced by the top-blown basic process is claimed to be equivalent to basic open-hearth steels of the same chemical composition.

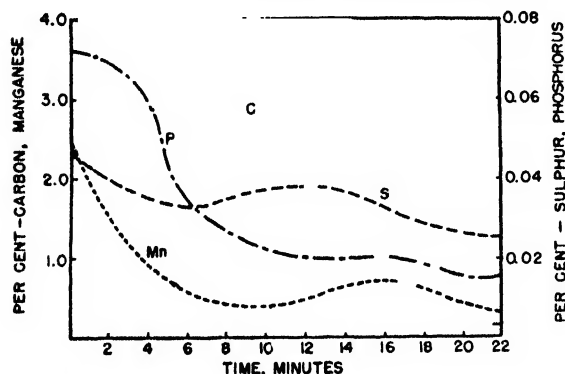


FIG. 14—13. Progress of refining in a top-blown, basic-lined vessel (after Roesner).

shows the progress of refining in a 5-ton experimental top-blown vessel.

Temperature Control—A major problem in the top-blown basic process is one of decreasing rather than increasing the temperature. This probably is associated with the use of oxygen rather than air for blowing. The principal method used for temperature control is regulation of the quantity and timing of scrap additions. Air or steam has not been used to any extent for cooling of the bath.

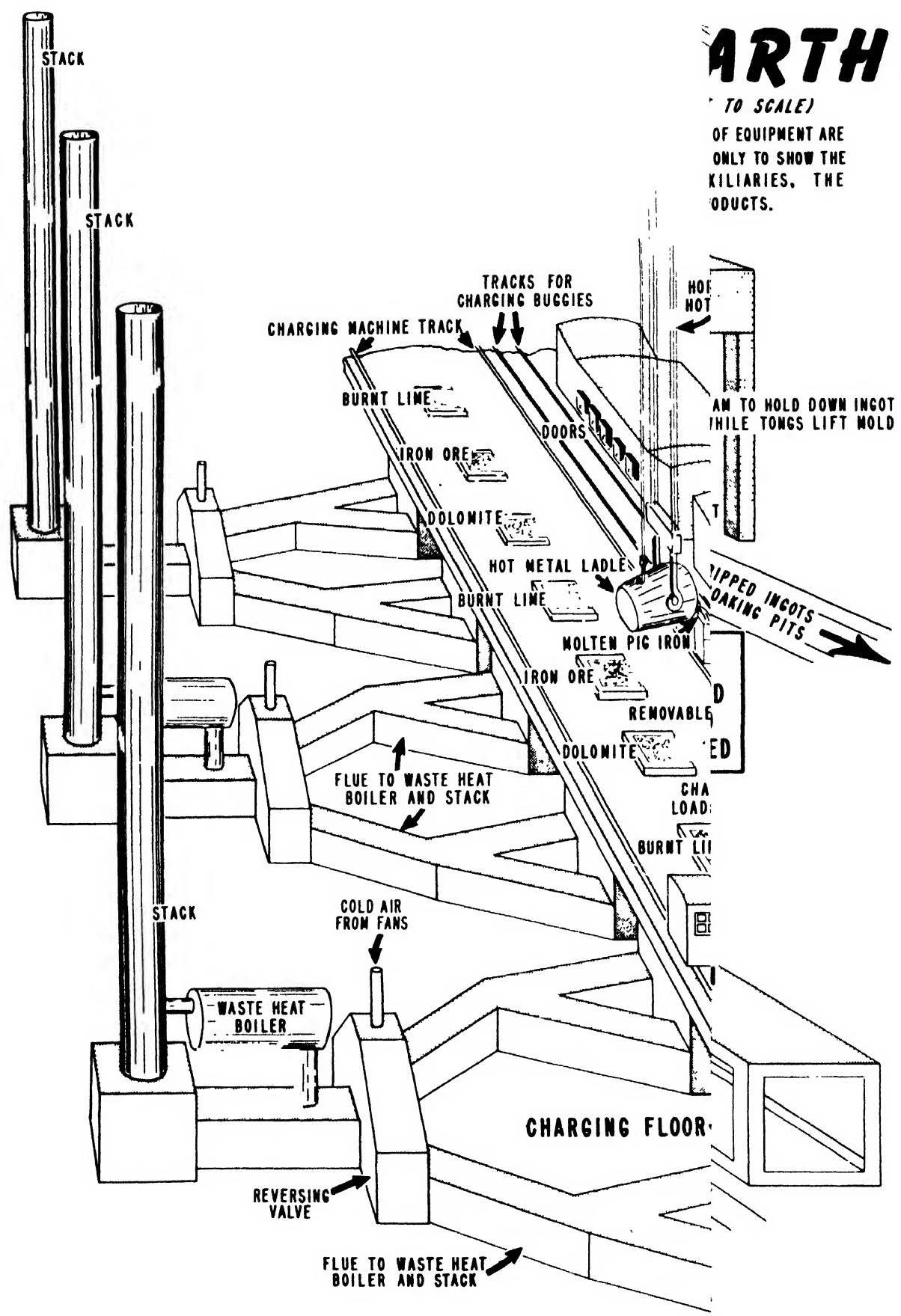
Deoxidation Reactions—It has been mentioned previously that, for most pneumatic steels produced, ferro-manganese is commonly used for deoxidation. Since Austrian iron blown in the top-blown basic process is very high in manganese content (1.6 to 2.5 per cent), the use of manganese additions to the ladle is frequently unnecessary. Further, it has generally been observed that the iron and iron oxide content of the basic slag from the top-blown vessel is lower than that obtained in other pneumatic processes. In these cases, the major control of deoxidation may be exercised by control of the final bath temperature.

Nitrogen and End-Point Control—No special means of controlling the nitrogen content of top-blown basic steels are necessary since the final steels produced contain from 0.002 to 0.004 per cent of nitrogen.

ARTH

(NOT TO SCALE)

THE EQUIPMENT ARE ONLY TO SHOW THE FUNDAMENTALS, THE PRINCIPLES.



Chapter 15

THE OPEN-HEARTH PROCESSES

SECTION 1

SOME GENERAL FEATURES OF THE SIEMENS PROCESS

Early History of the Process—The phenomenal success of the Bessemer process, coupled with the ever-increasing demand for steel, attracted many other inventors to the study of new and improved methods of steelmaking. Of all the methods which received attention, the only process which was destined to become a rival of the pneumatic process was developed through the invention of the regenerative principle by Karl Wilhelm Siemens who, although German-born, was a naturalized British citizen.

Siemens' early work with the regenerative principle as applied to steam engines showed that a very great saving of fuel and very high temperatures could be obtained by its use and, at the suggestion of his brother, Frederick, he turned his attention to the application of the principle for producing high temperatures in furnaces. The first experimental furnace was built in 1858, when it was discovered that, especially with large furnaces using solid fuel, many difficulties were to be overcome if the full efficiency which the principle promised was to be realized.

After two years or more of experimentation, Siemens fell upon the plan of gasifying the fuel prior to burning it in the furnace, whereby he found that most of the difficulties could be overcome. Thus, a correlative development of the regenerative furnace was the **gas producer**. Siemens' patent covering the successful furnace design specifically mentioned the possibility of using the furnace for the production of steel. An early design of a Siemens steelmaking furnace, with gas producer, is shown schematically in Figure 15-2.

In the ferrous industry, some of the early uses Siemens made of the furnace were for puddling, reheating iron and steel for forging and rolling, and for melting crucible steel. Siemens next turned his attention to the manufacture of steel in his furnace and, though many trials were made at different works, he met with only indifferent success. Finally, like Bessemer, he found it necessary to erect a steel works of his own. These works were located in Birmingham, England, and were employed at first in a remelting process by which steel of the best quality was obtained from processing such scrap as old rails, plates, and so on.

In the meantime, Siemens was busy developing the idea that steel could be made from pig iron by oxidizing the carbon content of the latter with iron ore and, by the year 1868, proved that this method, which came to be known as the **pig and ore process**, could be employed successfully. He next turned his attention to evolving a method whereby steel could be produced directly from the ore, thus dispensing with the blast furnace. In this feat he actually succeeded, but the cost of production was many times that of producing steel from pig iron. Subsequent events have proved that it is more practical

and economical to employ the blast furnace for the primary reduction of the iron ore to pig iron, and from this to make steel in the furnace which has been evolved from Siemens' original designs.

Principles of Siemens' Pig and Ore Process—Briefly, the method of Siemens was as follows: A rectangular covered hearth was used to contain the charge of pig iron or pig iron and scrap (see Figure 15-2). Most of the heat required to promote the chemical reactions necessary for purification of the charge was provided by passing burning fuel gas over the top of the materials. The fuel gas, with a quantity of air more than sufficient to burn it, was introduced through ports at each end of the furnace, alternately at one end and then the other. The products of combustion passed out of the port temporarily not used for entrance of gas and air, and entered chambers partly filled with brick checkerwork. This checkerwork, commonly called checkers, provided a multitude of passageways for the exit of the gases to the stack. During their passage through the checkers, the gases gave up a large part of their heat to the brickwork. After a short time, the gas and air were shut off at the one end and introduced into the furnace through the preheated checkers, absorbing some of the heat stored in these checkers. The gas and air were thus preheated to a somewhat elevated temperature, and consequently developed a higher temperature in combustion than could be obtained without preheating. In about twenty minutes, the flow of the gas and air was again reversed so that they entered the furnace through the checkers and port used first; and a series of such reversals, occurring every fifteen or twenty minutes, was continued until the heat was finished. The elements in the bath which were oxidized both by the oxygen of the air in the furnace atmosphere and that contained in the iron ore fed to the bath, were carbon, silicon and manganese, all three of which could be reduced to as low a limit as was possible in the Bessemer process.

Thus, as in all other processes for purifying pig iron, the basic principle of the Siemens process was that of oxidation. However, in other respects, it was unlike any other process. True, it resembled the puddling process in both the method and the agencies employed, but the high temperatures attainable in the Siemens furnace made it possible to keep the final product molten and free of entrapped slag. The same primary result was obtained as in the Bessemer process, but by a different method and through different agencies, both of which imparted to steel made by the new process properties somewhat different from Bessemer steel, and gave the process itself certain metallurgical advantages over the older pneumatic process. These advantages will be discussed later in this section.

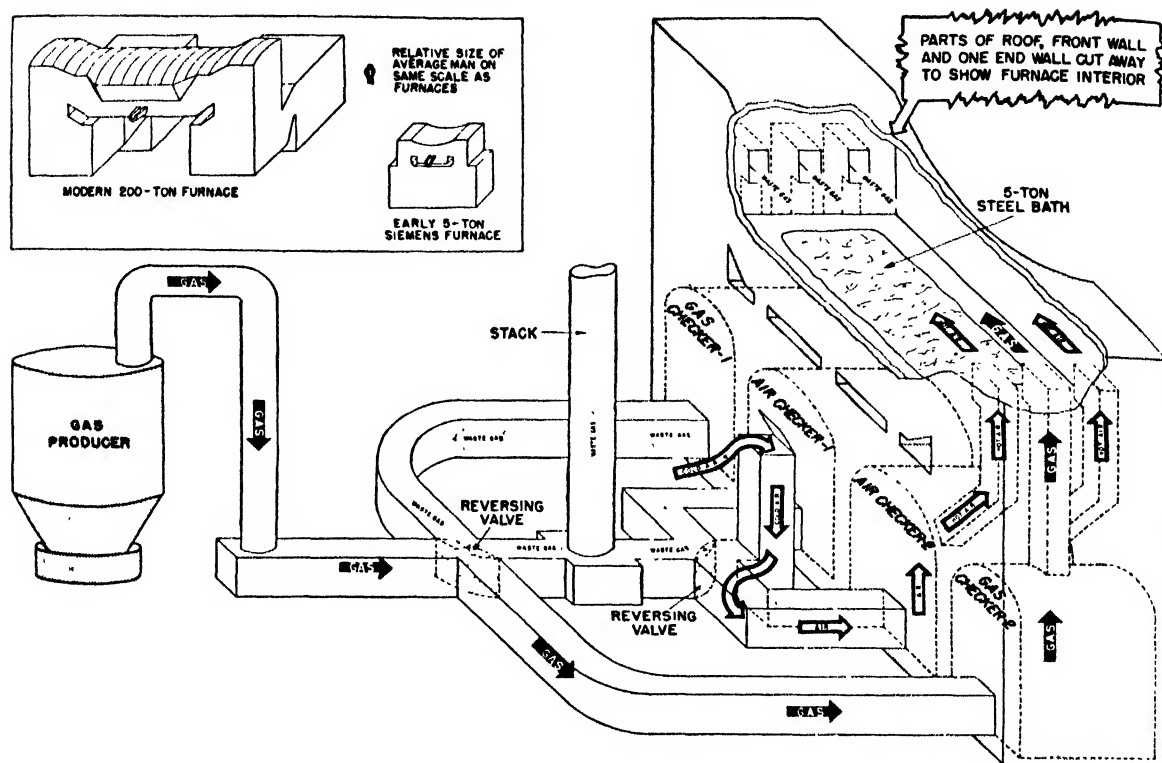


FIG. 15-2. Schematic arrangement of an early type of Siemens furnace with about a 5-ton capacity. The roof of this design (which was soon abandoned) dipped from the ends toward the center of the furnace to force the flame downward on the bath. Various different arrangements of gas and air ports were used in later furnaces. Note that in this design, the furnace proper was supported on the regenerator arches. Flow of gas, air and waste gases were reversed by reversing the position of the two reversing valves. The inset at the upper left compares the sizes of one of these early furnaces with a modern 200-ton open hearth.

Mechanical Changes and Improvements in the Siemens Process—As would be expected, many variations of the process, both mechanical and metallurgical, have been worked out since its original conception. Along mechanical lines, various improvements in the design, the size, and the arrangement of the parts of the furnace have been made. Originally, the furnaces had a capacity of only four or five tons, but now the size ranges generally from 100 to 150 tons, though in isolated cases as low as 40 tons or as high as 600 tons.

The early development of the Siemens steelmaking process was retarded by the lack of refractory materials capable of withstanding the high temperatures and the chemical reactions of the process. Even in today's advanced stage of development of the process, it is recognized that additional production and improved steel quality may be possible of attainment if new or improved refractories eventually make feasible a sustained operation at higher temperatures.

The Siemens process now is known more generally, at least in this country, as the open-hearth process. The name "open hearth" was derived, probably, from the fact that the steel, while melted on a hearth under a roof, was accessible through the furnace doors for inspection, sampling, and testing.

Metallurgical Improvements—The hearth of Siemens' furnace was of acid brick construction, on top of which the bottom was made up of sand—essentially as in the acid process of today. Later, to permit the charging of limestone and use of a basic slag for removal of phosphorus, the hearth was constructed with a lining of

magnesite brick, covered with a layer of burned dolomite or magnesite, replacing the siliceous bottom of the acid furnace. These furnaces, therefore, were designated as basic furnaces, and the process carried out in them was called the basic process.

The pig and scrap process was originated by the Martin brothers, in France, who, by substituting scrap for the ore in Siemens' pig and ore process, found it possible to dilute the charge with steel scrap to such an extent that less oxidation was necessary. Since the time of the Martins, these processes have undergone various modifications, chief of which are those known as the Talbot, the Campbell, the Bertrand-Thiel, and the Monell processes. These latter processes have become obsolete, and they did little to determine modern practice, except that Talbot and Campbell did exert an influence which still is reflected in tilting-furnace practice, as discussed in Chapter 17.

Advantages of the Siemens Process—The advantages offered by the Siemens process may be summarized briefly as follows: (1) By the use of iron ore as an oxidizing agent and by the external application of heat, the temperature of the bath is made independent of the purifying reactions, and the elimination of impurities can be made to take place gradually, so that both the temperature and composition of the bath are under much better control than in the Bessemer process. (2) For the same reasons, a greater variety of raw materials can be used (particularly scrap, not greatly consumable in the Bessemer converter) and a greater variety of products may be made by the open-

hearth process than by the Bessemer process. (3) A very important advantage is the increased yield of finished steel from a given quantity of pig iron as compared to the Bessemer process, because of lower inherent sources of iron loss in the former, as well as because of recovery of the iron content of the ore used for oxidation in the open hearth. (4) Finally, with the development of the basic open-hearth process, the greatest advantage of the Siemens over the acid Bessemer process was made apparent, since the basic open-hearth process is capable of eliminating phosphorus from the bath. While this element can be removed also in the basic Bessemer (Thomas-Gilchrist) process, it is to be noted that, due to the different temperature conditions, phosphorus is eliminated before the carbon in the basic open-hearth process, whereas the major proportion of the phosphorus is not oxidized in the basic Bessemer process until after the carbon, in the period termed the afterblow. Hence, while the basic Bessemer process requires a pig iron with a phosphorus content of 2.00 per cent or more in order to maintain the temperature high enough for the afterblow, the basic open-hearth process permits the economical use of iron of any phosphorus content up to 1.00 per cent. In the United States, this fact is of importance since it makes available immense iron ore deposits which could not be utilized otherwise because of their phosphorus content, which is too high to permit their use in the acid Bessemer or acid open-hearth processes and too low for use in the basic Bessemer process. For this reason, the basic open-hearth process has become the leading steelmaking method in this country.

The Open-Hearth Process in the United States—As early as 1868, a small open-hearth furnace was built at Trenton, New Jersey, but satisfactory steel at a reasonable cost did not result and the furnace was abandoned. Later, at Boston, Massachusetts, a successful furnace was designed and operated, beginning in 1870. Following this success, similar furnaces were built at Nashua, New Hampshire, and in Pittsburgh, Pennsylvania, the latter by Singer, Nimick and Company, in 1871. The Otis Iron and Steel Company constructed two 7-ton furnaces at their Lakeside plant in 1874. Two 15-ton furnaces were added to this plant in 1878, two more of the same size in 1881, and two more in 1887. All of these furnaces had acid linings, using a sand bottom for the hearths.

A furnace with a basic bottom, rammed from Austrian magnesite, produced basic steel at the Otis plant in January, 1886. Production rate on the basic furnace was so low compared to that normally achieved with an acid bottom that the basic bottom was torn out after four months, and replaced by an acid bottom. Following the Otis installations, the following companies also installed open-hearth furnaces: The Cleveland Rolling Mills (now part of the American Steel and Wire Division of United States Steel Corporation); The Pennsylvania Steel Company (later Bethlehem Steel Company); the Schoenberger Works (now dismantled, but

formerly part of the American Steel and Wire Division); and Carnegie, Phipps and Company (now Homestead District Works of United States Steel Corporation).

It was in the last-named plant that the commercial production of steel by the basic process was achieved first, the initial heat being tapped March 28, 1888. By the close of 1890, there were 16 basic open-hearth furnaces operating. From 1890 to 1900, magnesite for the bottom began to be imported regularly and the manufacture of silica refractories for the roof was begun in American plants. For these last two reasons, the construction of basic furnaces advanced rapidly and, by 1900, furnaces larger than 50 tons were being planned.

While the Bessemer process could produce steel at a possibly lower cost above the cost of materials, it was restricted to ores of a limited phosphorus content and its use of scrap was also limited. The open hearth was not subject to these restrictions, so that the annual production of steel by the open-hearth process increased rapidly, and, in 1908, passed the total tonnage produced yearly by the Bessemer process. Total annual production of Bessemer steel has decreased rather steadily since 1908. The main reasons were the flexibility of the open-hearth process, with respect to its ability to produce steels of many compositions, and its ability to use a large proportion of steel or iron scrap, if necessary.

In 1953, approximately 89.5 per cent of the steel produced in the United States came from the basic open hearth, with about 3.5 per cent from the acid Bessemer, and 6.5 per cent from the electric furnace. The preponderance in favor of the basic process for production of steel for ingots is based on the ability of this process to remove phosphorus (and some sulphur) while the acid process is limited to removal of carbon, silicon, and manganese and must use carefully selected raw materials that contain less than specified amounts of phosphorus and sulphur.

At the beginning of 1954, the steelmaking capacity of the United States was divided between the various processes as follows:

Process	Capacity (Net Tons Per Year)
Basic open hearth.....	108,279,490
Acid open hearth.....	815,240
Total open-hearth capacity.....	109,094,730
Bessemer	4,787,000
Electric furnace*	10,448,680
Grand Total	124,330,410

These figures demonstrate the dominance of the basic open-hearth furnace as the preferred source of large tonnages of quality steel. Table 14—I in the chapter on "The Pneumatic Steelmaking Processes" shows the production of steel ingots by various processes over an extended period.

* Includes 40 tons of steel manufactured by the crucible process.

SECTION 2

BASIC OPEN-HEARTH PLANT LAYOUT

Because of their greater extent and complexity as compared with acid open-hearth steelmaking plants, only the arrangement of large, modern, basic open-hearth plants engaged in the production of steel ingots will be considered here.

Layout of the Main Furnace Building—Open-hearth furnaces are housed in large, steel buildings, designed

and arranged to facilitate the charging of the furnaces, the making of the heats, and handling of the finished molten steel.

The furnaces are installed in a long, straight row, approximately along the centerline of the main furnace building, with the same distance between the ends of adjacent furnaces (Figure 15—3). Space may be left at

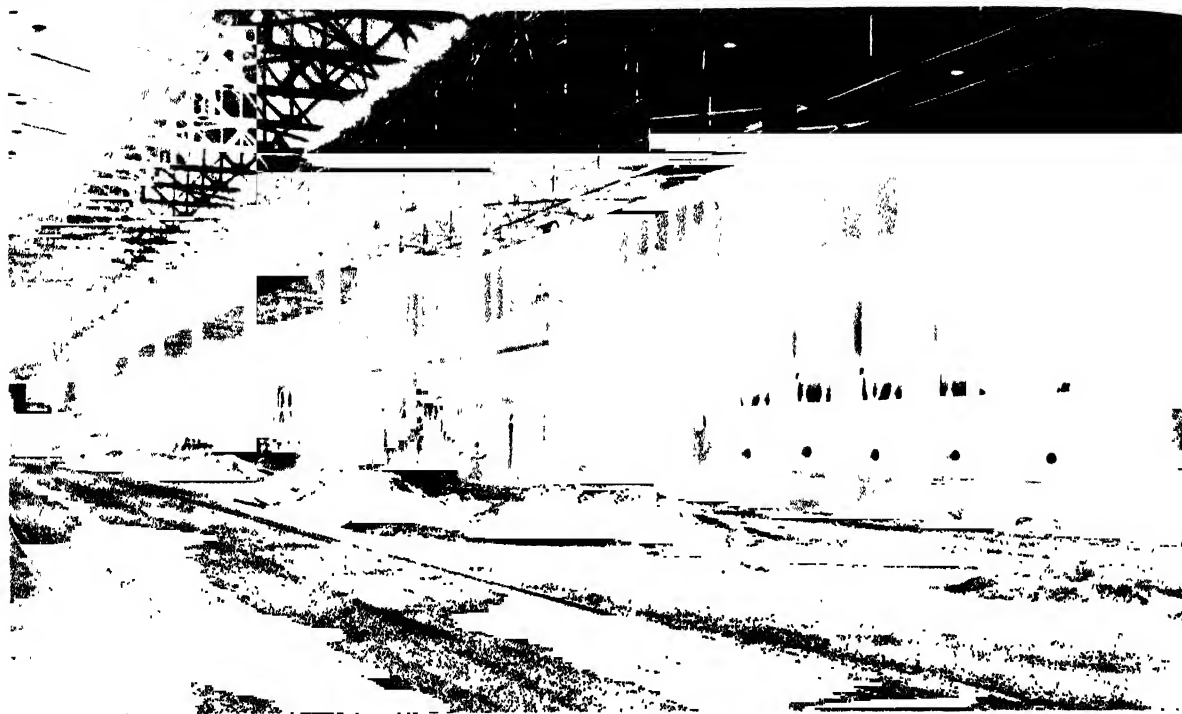


FIG. 15-3. General view of the charging side of a group of modern furnaces, showing charging machine in left background.

either end of the line of furnaces for storage or other purposes. It is preferred by most designers that not over 12 furnaces be placed in one row. Between each pair of furnaces is a vertical, composite steel column which performs two functions (Figure 15-4). The tall central section of each column rises to a height sufficient to support the main roof truss above it. On opposite sides of this section are two shorter, heavy steel sections that each support one of the tracks of the two electric overhead traveling crane runways that extend the length of the building. The other track of each of the two runways is supported by columns at the two sides of the building. The distance between these columns may be from 75 to 117 feet, depending upon the length of the furnaces. The longitudinal centerline of the furnaces is parallel to the line of the columns between them, but several feet in front, so that the bridge of the charging-floor crane extends over most of the top of each furnace when positioned over it. This permits

the crane hoist to be operated directly over the furnace when occasion demands.

That part of the floor area in front of the row of furnaces is called the charging floor and, in modern shops, it is elevated about 22 feet above the general yard level of the plant as shown in Figure 15-5. The part of the furnace building above the charging floor is 75 to 85 feet wide. Shops with this arrangement are designated as two-level shops, to distinguish them from plants of older design in which the charging floor was at general yard level.

The remainder of the floor space of the building, located on the tapping side of the furnace, is called the pouring floor or pit side, and is at general yard level.

Charging Side—On the charging side of the furnaces and close to them is laid a broad-gage track on which operate the locomotives (usually Diesel in modern plants) that move the buggies which carry the charging boxes loaded with the scrap and other solid ma-

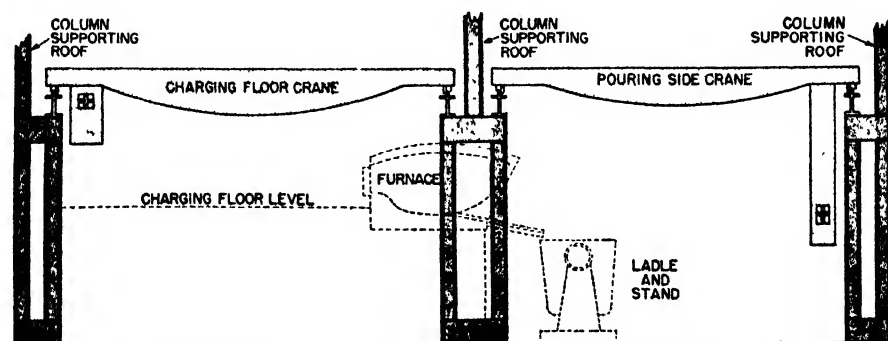


FIG. 15-4. Arrangement of the columns supporting the roof trusses and crane runways of a modern open-hearth furnace building.

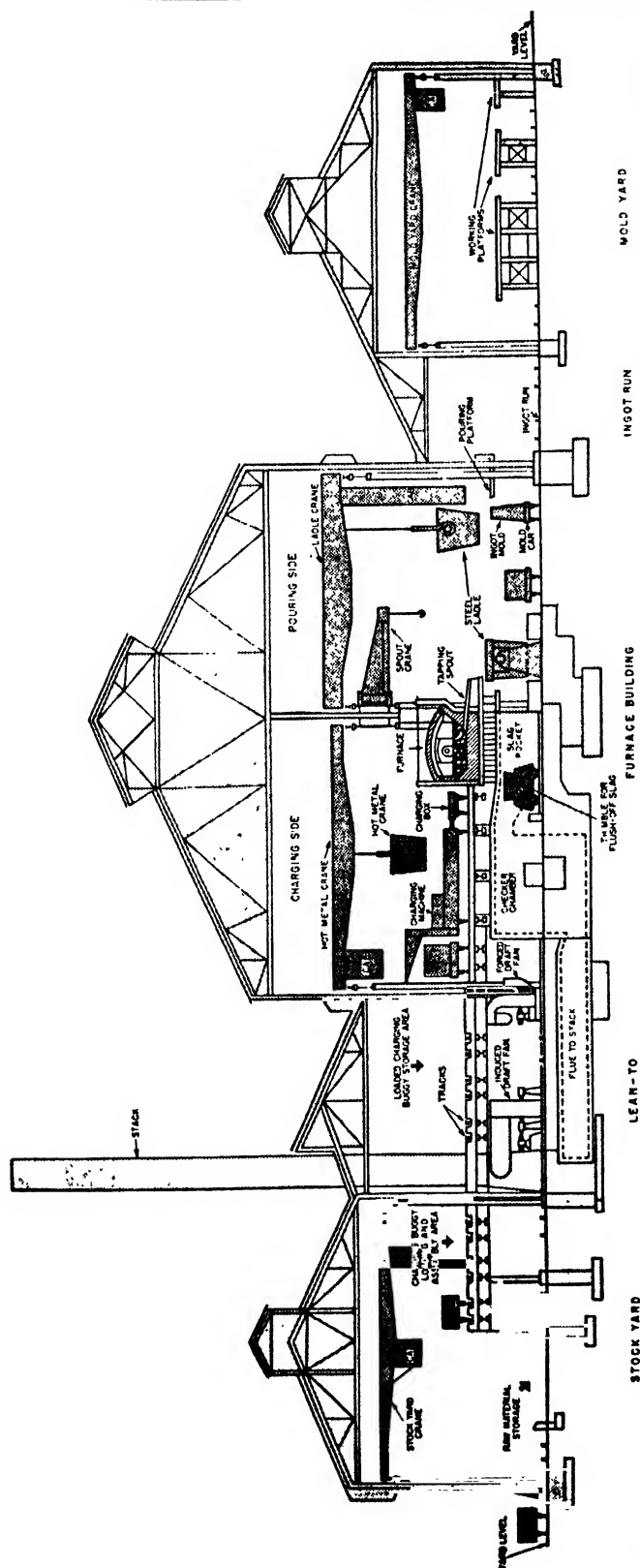


FIG. 15-5. Diagrammatic cross-section of a modern open-hearth plant, showing the relative locations and sizes of the various buildings and the disposition of the chief operating units required for charging and operation of the furnace and disposition of the finished steel.

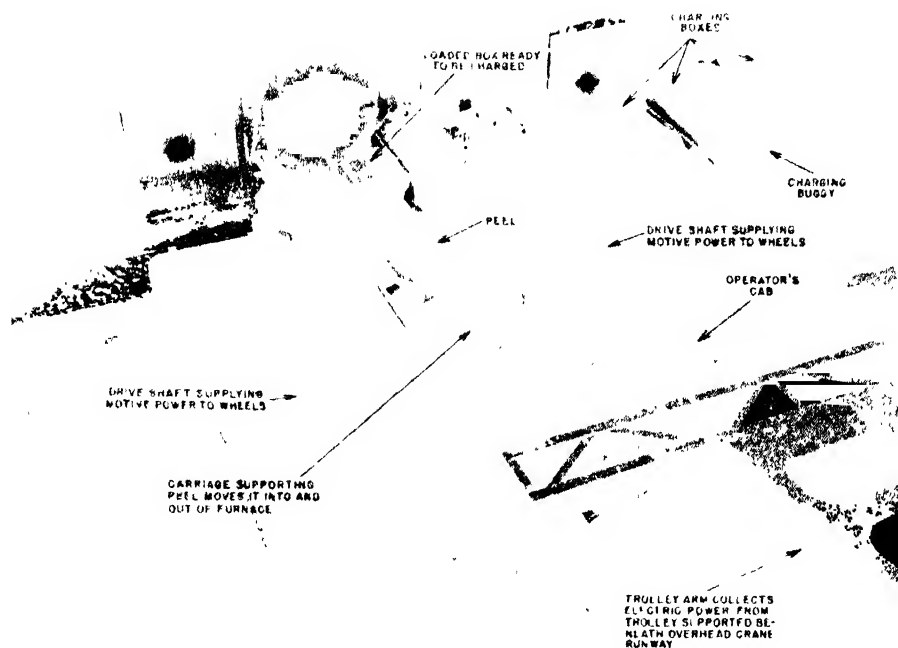


Fig. 15-6. Typical charging machine in position before an open door of an open-hearth furnace, with a charging box engaged by the peel and ready to be moved into the furnace and dumped

materials to be charged into the furnaces. Parallel to this track on the charging floor is a special track of very wide gage on which operate the charging machines, illustrated in Figure 15-6. These electrically operated machines are equipped to pick up the charging boxes (which have a capacity up to 50 cubic feet) one at a time, from the buggies in position before the doors of a furnace, thrust the boxes through an open door into the furnace and turn them to dump their contents on the hearth; following this, the boxes are withdrawn from the furnace and replaced on the buggies (Figures 15-7 and 15-8). The empty boxes then are hauled away and refilled in preparation for the next charge. Two to four charging machines of 10 to 15 tons capacity may be provided.

Electric overhead traveling cranes operating on the crane runway over the charging floor are known as **charging floor cranes**. They have a main hoist designed with capacity for handling ladles of molten iron from the mixer to be charged by pouring into the open-hearth furnaces, and also have an auxiliary hoist for miscellaneous work on the charging side. The main hoist capacity ranges from 100 to 150 tons and the auxiliary hoist capacity may approach 40 tons.

A mixer serves the dual purpose of (1) having a supply of molten pig iron available whenever needed for charging, and (2) mixing successive "casts" or lots of iron from blast furnaces, so that possible irregularities in the iron are smoothed out or compensated. A hot-metal building or mixer building usually is located at one end of the charging floor. This arrangement permits the transfer of molten pig iron (in ladles) from the metal mixer by electric overhead traveling crane or, better, by an electric ladle car running on a separate track laid on the charging floor. In the latter case, the cranes are employed to dump the ladles in charging the molten iron into a furnace. Hot metal mixers have been built with capacities up to 1500 tons. In plants of recent design, the holding capacity has been divided between two mixers, each of 800 tons capacity.

Hot-metal mixers generally are barrel-shaped, constructed of steel plate and lined with suitable refractories. The steel shell of the mixer is mounted on rocker-shaped castings which rest on rollers so that the mixer can be tilted by an electric-motor drive to pour off the desired amount of metal into a transfer ladle for transport to the open hearth furnaces (see Figure 15-9).

Mixers have been lined with natural siliceous rock (high silica content) or dense fireclay refractories. While these types of linings still predominate, more difficult service conditions have led in recent years to trials of superior refractories such as the super-duty fireclay and high-alumina types.

The stockyard where scrap and other solid materials are loaded and the *lean-to aisle* where loaded charging boxes on cars are stored while awaiting charging will be described later.

Pouring Floor or Pit Side—The pouring floor, or pit side, of the furnace building extends along the tapping side of the furnaces. It is usually about 70 to 80 feet wide and as long as the charging floor, but at general yard level. The **pouring platforms** are situated along the inside of the wall of the building, opposite to the back or tapping side of the furnaces. Two to four such platforms are installed, at a height convenient to the tops of the ingot molds standing on mold cars ready for filling (Figure 15-10).

Facilities for relining and drying ladles are installed on the pouring side, and equipment for preparing stopper rods is accessible to or from this area, as are grinding and mixing facilities for preparing plastic refractory mixes. The pouring side may be longer than the charging floor if pits needed for pouring very large ingots are required.

Jib-type cranes of $5\frac{1}{2}$ to 7 tons capacity are installed on the tapping side of each furnace to handle such items as furnace tapping spouts.

Stock Yard—The stock yard where scrap and other solid materials are received and loaded should be lo-

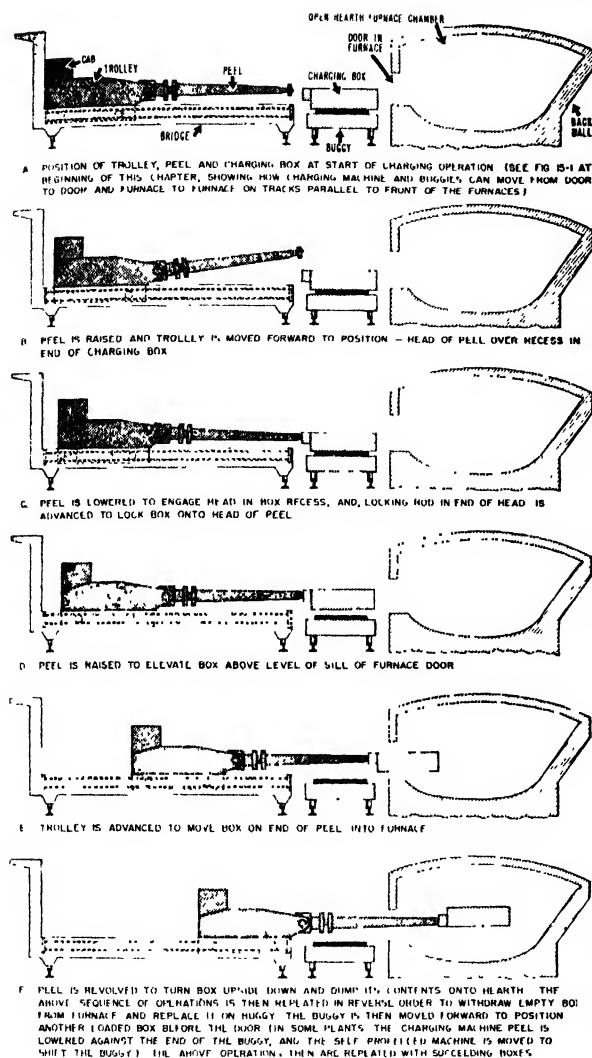


FIG. 15-7. Sequence of operations of a charging machine.

located as conveniently with respect to the charging floor as possible. Stock yards in most modern plants are located parallel to the charging floor, and on that side of the furnaces beyond the furnace stacks. The stock yards are 70 to 100 feet wide, almost as long as the main or furnace building, and preferably are roofed in. Three or four electric overhead traveling cranes of 10 to 15 tons capacity and equipped with lifting magnets usually are required to handle the raw materials. It is preferable to have two levels under the crane runway. The upper, at charging floor level, may have two or three standard-gage tracks on which charging buggies are loaded; the lower or general yard level may have one standard-gage track where melting stock and furnace materials are received and where some scrap may be stocked (Figure 15-5).

Lean-to—Between the stock yard and the charging floor is the lean-to (Figure 15-5). This may be as much as 60 feet wide, and usually is not equipped with cranes. The main floor of the lean-to is at charging floor level, and carries tracks on which the charging cars or buggies carrying loaded charging boxes from the stock yard are stored until needed at the furnaces. The furnace stacks usually pass upward through the lean-to. In the

spaces between columns at the sides of the main floor of the lean-to, enclosures may be built to house furnace control equipment, laboratories, locker rooms, weighing equipment, bins for various ferroalloys, and so on.

Beneath the main floor of the lean-to and at general yard level (Figure 15-5) are installed the waste-heat boilers and the fans that regulate the flow of combustion air to the furnaces. This area usually is equipped with standard-gage tracks.

Trackage—There should be sufficient trackage, interlinked with cross-overs, from the stock yard to the charging floor to prevent interference in the movement of charging boxes on cars and their placement before the proper furnace at whatever time they are needed. It is preferable to use standard-gage track throughout this portion of the plant. This permits the use of a charging car of greater width that can accommodate larger charging boxes than when using narrow-gage track, and this facilitates the charging of scrap and other solid materials. Another advantage is that direct loading in and out of standard-gage railroad cars in all parts of the plant is feasible when standard-gage tracks are available.

Mold Yard—The mold yard is housed in a building located as conveniently as possible with respect to the pouring platforms. In it are facilities for cleaning and coating molds and replacing them on the mold cars.

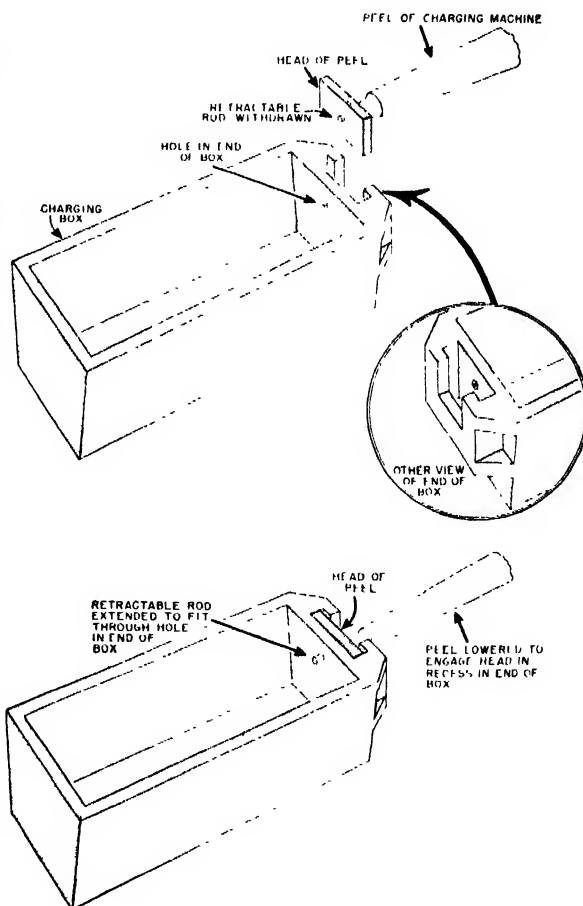


FIG. 15-8. Sketch showing one method by which the head of a charging machine peel is temporarily locked onto the end of a charging box to permit the box to be turned upside down for dumping its contents onto the bottom of an open-hearth furnace.

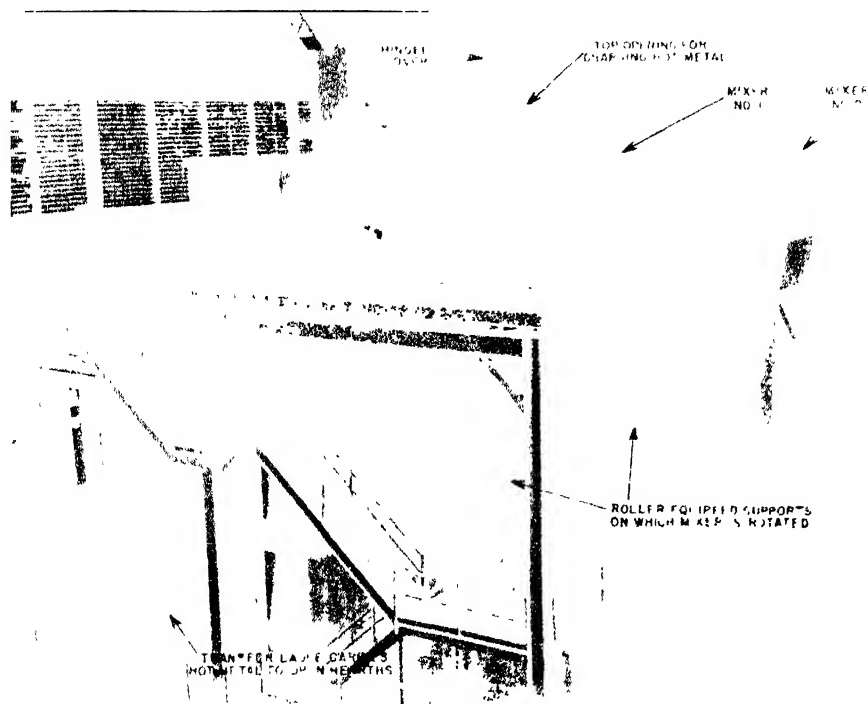


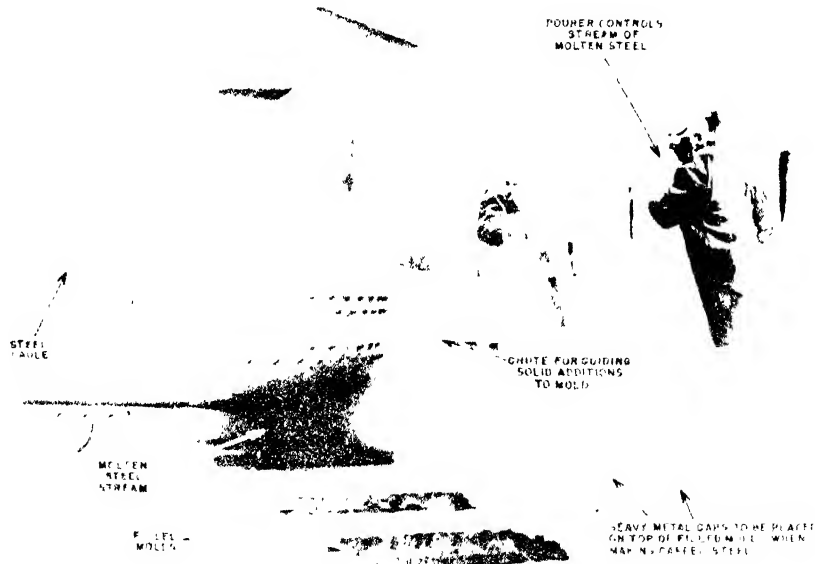
FIG. 15-9. Two identical hot-metal mixers, each of 800 tons capacity, installed in a separate mixer building at one end of the charging floor of an open-hearth furnace building. The mixers rotate counter-clockwise to pour molten pig iron into the transfer ladle shown.

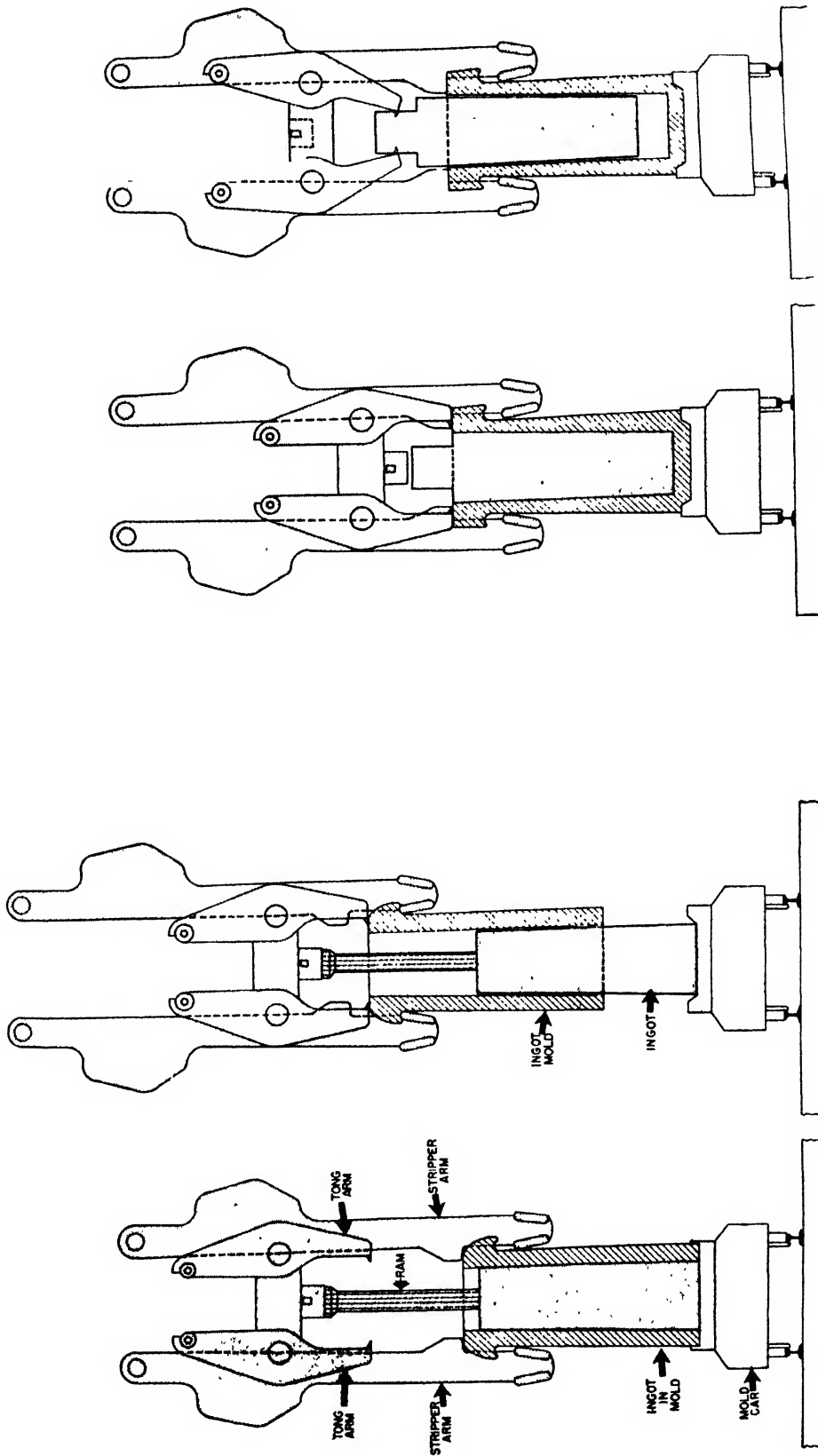
For modern plants, it is 70 to 100 feet wide and somewhat shorter than the main furnace building. The mold cars are handled in sets called drags, and for these the tracks are usually paralleled by a platform from which the molds may be inspected and cleaned and the hot tops adjusted. Mold-storage space must be provided, and all facilities are served by two or three electric overhead traveling cranes of 25 to 60 tons capacity. Where the mold yard parallels the main building, a lean-to from the mold yard to the main building provides a desirable cover over the tracks over which mold

cars are transported or on which they are stored ready for placement at the pouring platforms.

Miscellaneous Facilities—Other buildings making up the complete open-hearth plant are the slag yard where slag is broken to proper size and steel scrap recovered from it, the scrap drop where steel scrap is broken or burned with cutting torches into desired sizes, and a stripper building which, in best practice, may be located adjacent to the building where ingots are heated in soaking pits for rolling. After the ingots have solidified in the molds, having been held at the pouring plat-

FIG. 15-10. By manipulating the lever, the pourer opens and closes the nozzle of the ladle to control the flow of molten steel from the ladle to the molds. The operation shown is called "teeming" or "pouring."





STRIPPING BIG-END-DOWN INGOTS

Pressure is applied to the top of the ingot by the ram to hold it down while the projections on the bottoms of the stripper arms engage the lugs on the mold and exert an upward pull to strip the mold from the ingot. The tong arms do not operate when stripping this type of ingot.

STRIPPING BIG-END-UP INGOTS

Projections of the sides of the stripper arms hold down the ingot mold while the tong arms are closed to bite into the projecting hot top of the ingot and then draw it from the mold. Use of the ram is not necessary when stripping this type of ingot, so it is designed to be removed simply by withdrawal of a key.

FIG. 15-11. Schematic representation of the action of an ingot stripper in removing the molds from (left) big-end-down ingots and (right) big-end-up ingots.

forms for periods determined by metallurgical experience, the drag of molds with their contained ingots are drawn by motive power to the **strippers** which usually are special electric overhead traveling cranes. With big-end-down molds, the stripper lifts the mold from the ingot car while a plunger forces the ingot down against the ingot stool on the mold car (Figure

15-11). Molds after lifting from the ingots are transferred by the stripper to a parallel drag of cars. With big-end-up molds, it is necessary to pull or push the ingot up and out of the mold while the latter is held against the stool and car (Figure 15-11). Modern stripper cranes, rated at 200 to 250 tons capacity, usually are designed to handle either type of molds.

SECTION 3

FURNACE CONSTRUCTION

Description—In order to avoid repetition, the following description of the open-hearth furnace is made to serve for both basic and acid furnaces. They are similar with respect to form and arrangement of their parts, the chief difference being in the kind of materials used in constructing their hearths or bottoms in which the metal is refined. The present development of the use of basic materials has introduced basic brick into certain sections of the basic open-hearth furnace structure which may not be permissible in furnaces operating on the acid open-hearth process, because basic material introduced by erosion or spalling into the acid open-hearth slag would interfere with the close control of the acid process.

Since, in this country at least, from the standpoint of the number of furnaces and annual production of steel, the basic process is the more important, the description which follows is based on the basic furnace, and only such parts of acid furnaces as require different materials for their construction are described. However, it should be emphasized that even basic furnaces at present differ as to details of construction, principally because of the different fuels used. The number of the various furnaces and the range of capacities are given in Table 15-1.

Parts of the Open-Hearth Furnace and Their Arrangement—The open-hearth furnace is both reverberatory and regenerative. It is reverberatory in that the charge is melted on a refractory hearth, which is

shallow in relationship to the length of the hearth, by a flame passing over the charge so that both the charge and the relatively low (above the hearth) roof, built of refractory brick, are heated by the flame, with a part of the heating effect on the charge accomplished by radiation from the heated roof. Radiation from the flame is an effect the intensity of which varies with the kind of fuel used and the method by which it is burned.

Open-hearth furnaces are regenerative in that the hot gases (from the combustion of fuel) pass out of the reverberatory furnace chamber through passages into chambers known as **regenerative chambers** containing fire brick. These brick are so arranged as to give a large surface contact with the hot gases, which give up part of their heat to the brick. The direction of gas flow is reversed periodically, and the cold incoming air for combustion (and the fuel gas, if necessary) is pre-heated in the regenerative chambers so as to increase the temperature of the flame by a very considerable amount above that which could be obtained by burning the same fuel without pre-heating the combustion air (see Figure 15-12).

The parts of the open-hearth furnace and their arrangement are shown in Figures 15-13, 15-14, 15-15 and 15-16. From these illustrations, it will be observed that an open-hearth furnace consists of the furnace proper, containing the covered hearth on which the charge is placed; ports for admitting the fuel and air for combustion to produce a flame which flows over

Table 15-1. Open-Hearth Furnaces in the United States
(As of January 1, 1954)

Rated Capacity Per Heat (Net Tons)	Basic Stationary	Basic Tilt-ing	Total Basic Furnaces	Acid Stationary	Acid Tilt-ing	Total Acid Furnaces	Total Both Groups
11- 30	4	3	7	7	..	7	14
31- 50	13	..	13	8	..	8	21
51- 70	33	..	33	9	4	13	46
71- 90	22	..	22	2	..	2	24
91-110	59	..	59	2	..	2	61
111-130	146	..	146	1	..	1	147
131-150	136	12	148	148
151-170	124	9	133	133
171-190	42	3	45	45
191-210	99	6	105	105
211-230	67	..	67	67
231-280	79	..	79	79
281-330	19	..	19	19
331-380	5	..	5	5
381-400	1	..	1	1
401-450	5	..	5	5
451-500	11	..	11	11
501-550	2	..	2	2
Total	867	33	900	29	4	33	933

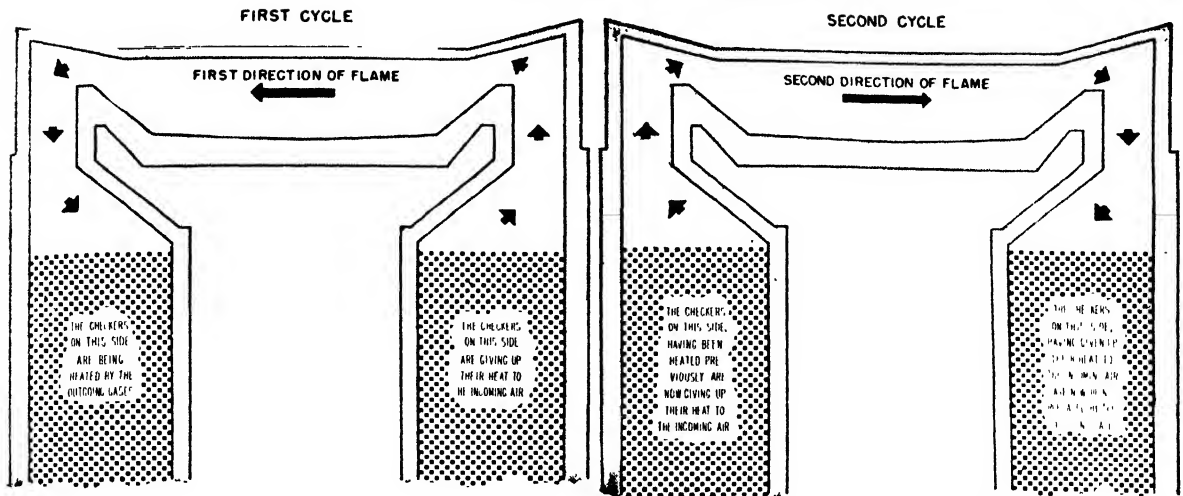


FIG. 15-12. These diagrams show the function of the checkers in preheating air for combustion in regenerative furnaces. Fuel is admitted to the furnace at the same end as the preheated air for each cycle.

the charge from the incoming port to the outgoing port; **regenerative chambers** containing brick known as the **checker-work**, or **checkers**, which store up heat transferred to them from the products of combustion, and subsequently impart a large part of this stored-up heat to the fuel (when the fuel gas is preheated) and to the air for combustion; **fantail flues** and **uptakes** connecting the regenerative or checker chambers with the furnace

proper; **slag pockets** which are located at the base of the uptakes; **flues** leading from the air supply (and the gas supply, if producer gas is used) to the regenerative chambers, with connections to the **stack** and to the **waste-heat boiler** (when the latter is installed); **valves** or **dampers** (Figure 15-16) for regulating flow of air, fuel gas and waste gases; and the **stack** itself.

The furnace proper is located with respect to the level of the charging floor so that it can be charged easily. It rests on steel bed beams which are supported, in modern practice, on steel columns which are bricked in for protection from the heated atmosphere and from molten slag or metal.

The slag pockets, regenerator chambers, flues and valves are located on a lower level in an area known

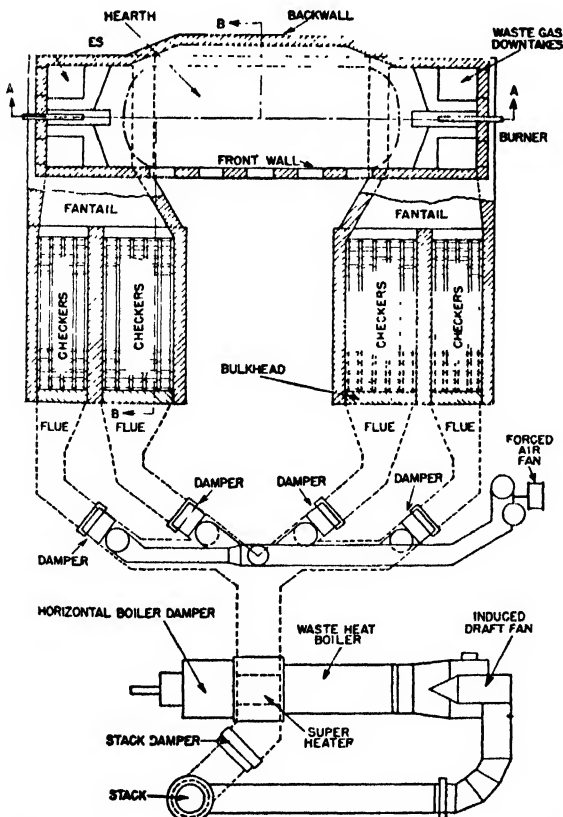


FIG. 15-13. Sectioned schematic plan (not to scale) of the arrangement of a liquid-fuel fired open-hearth furnace, showing location and names of principal parts (see also Figures 15-14, 15-15 and 15-16).

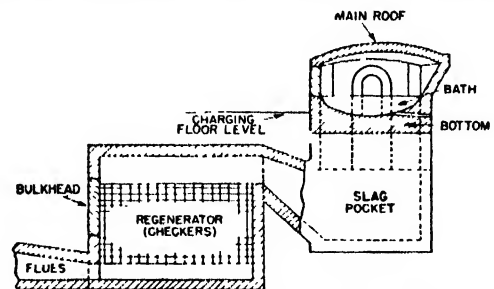


FIG. 15-14. Vertical section across the width of an open-hearth furnace, not to scale, corresponding to Section B-B of Figure 15-13, indicating names and relative locations of principal parts.

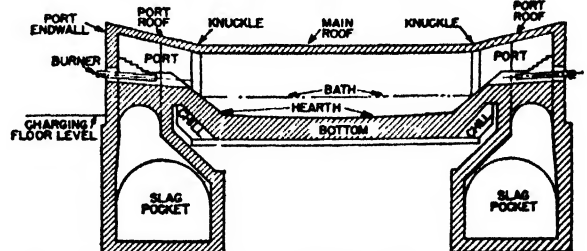
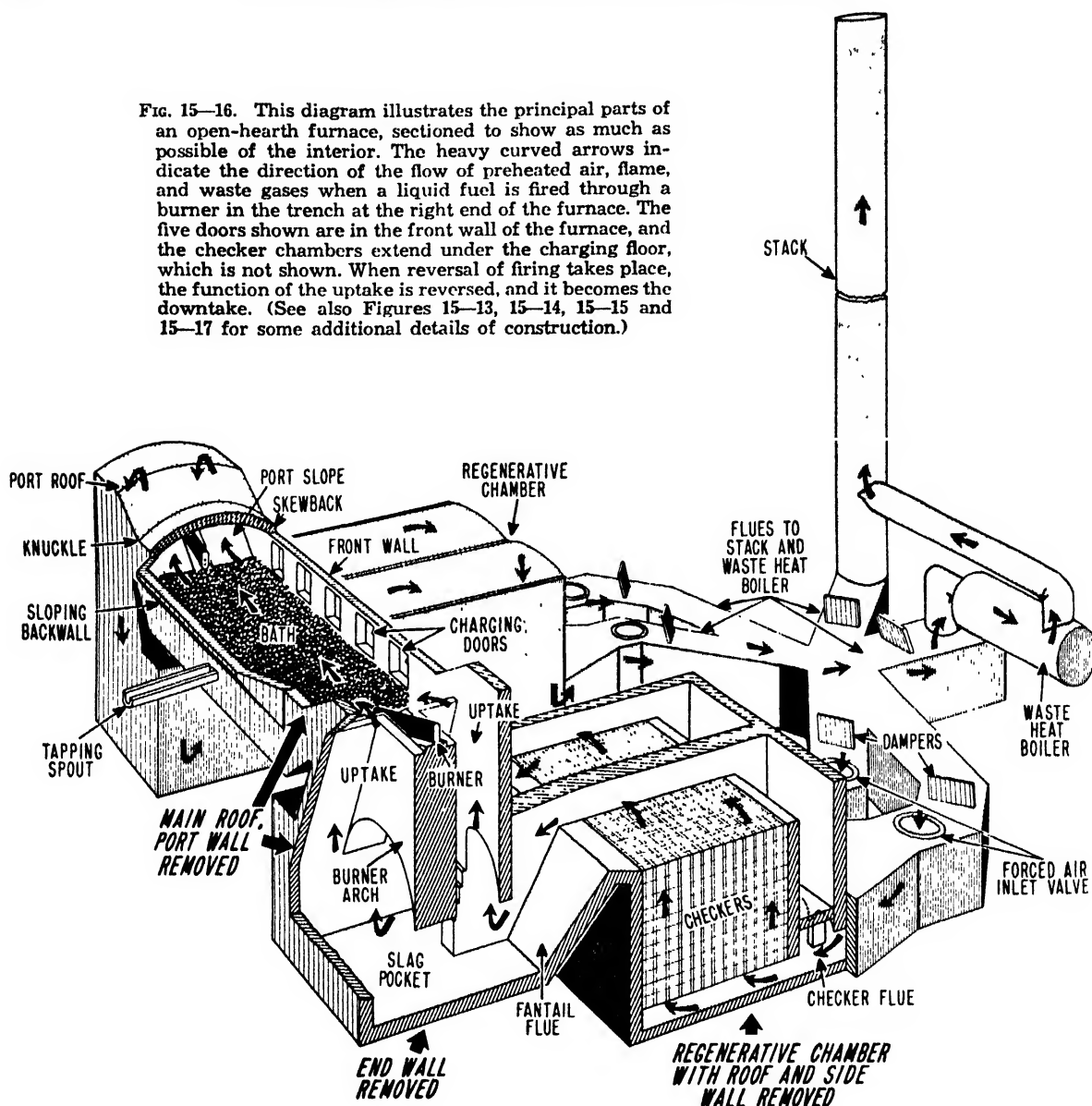


FIG. 15-15. Diagrammatic vertical section (not to scale) along the length of a liquid-fuel fired open-hearth furnace, corresponding to Section A-A of Figure 15-13, giving nomenclature of major parts.

FIG. 15-16. This diagram illustrates the principal parts of an open-hearth furnace, sectioned to show as much as possible of the interior. The heavy curved arrows indicate the direction of the flow of preheated air, flame, and waste gases when a liquid fuel is fired through a burner in the trench at the right end of the furnace. The five doors shown are in the front wall of the furnace, and the checker chambers extend under the charging floor, which is not shown. When reversal of firing takes place, the function of the uptake is reversed, and it becomes the downtake. (See also Figures 15-13, 15-14, 15-15 and 15-17 for some additional details of construction.)



as the cellar, the general ground level of which is, in modern plants, twenty-two feet below the charging-floor level. The base of the stack is set on an approximate level with the bottom of the regenerator chambers.

The Furnace Proper—This is a rectangular brick structure, supported on the sides and ends by steel buckstays in the form of steel beams, channels, or (for the walls on the front and back of the hearth) slabs, bound together at their tops, both longitudinally and crosswise above the furnace chamber and ports, by steel struts and tie rods. The buckstays rise vertically from their place of attachment to the steel bed under the furnace hearth, except those supporting the back wall. The back wall buckstays rise vertically from the steel bed to approximately the level of the furnace door sill level, then follow the sloping back-wall contour to approximately the top of the back wall, when they again become vertical.

The Basic Hearth—The hearth (Figure 15-17) covers that part of the furnace below the charging door sill

level, including the bottom and the banks. Older furnaces have "solid" bottoms resting on concrete foundations. Such construction would not be duplicated with present standards of construction. Modern furnaces have "pan" bottoms of closely spaced steel beams covered with steel plates and, thus, the spaces between the beams under the bottom are open to the atmosphere. Wide variations are found in bottom composition and thickness, the latter varying from 15 to 44 inches in different furnaces and varying with the hearth capacity. On the steel bottom plate, the usual practice is to place 1 to 3 inches of insulating concrete or insulating brick, although this may be omitted in the area around the tap hole. Fireclay brick next are laid to a depth of 5 to 12 inches. Over the fireclay brick, from $4\frac{1}{2}$ to 18 inches of basic brick, such as burned chrome brick (used to the greatest extent), burned magnesite, or burned or unburned chrome-magnesite brick are laid. Both the fireclay and the basic brick courses may be laid in increasing vertical thicknesses from the flat

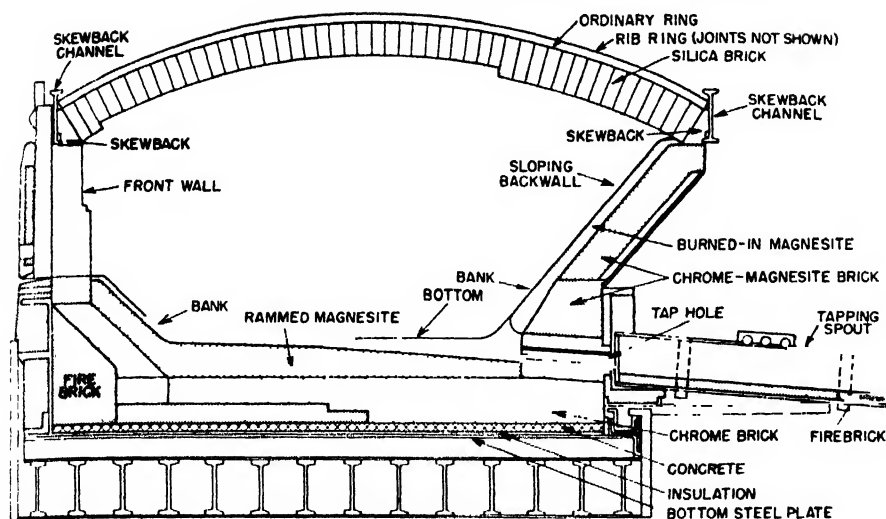


FIG. 15-17. Vertical cross-section of a modern open-hearth furnace through the tap-hole, indicating materials of construction of the various parts. Buckstays and other binding elements are not shown.

centerline section to the banks and hearth ends. From the contour established by the hearth brick construction, the completion of the bottom may proceed according to any of the following general methods:

1. Burned-In Grain Magnesite Bottoms. This is the earliest type of open-hearth furnace basic bottom used after magnesite became available in this country, and was the most commonly used until recent years. After raising the temperature of the furnace chamber to 2900° to 3000° F and maintaining this temperature for 16 to 24 hours, dead-burned grain magnesite, intimately mixed with 8 to 20 per cent of ground basic open-hearth slag, is burned in, or sintered, layer by layer, starting with the upper surface of the basic brick courses mentioned above. The layers are about an inch in thickness, and require about 2 to 3 hours of heating to sinter each one at temperatures between 2900° and 3000° F. Thus, the bottom is built up gradually to the desired thickness and contour. The final contour must provide for proper drainage of steel and slag from the furnace through the tap hole.

During the burning-in process, the slag content of the bottom-making material may be decreased as the total burned-in layer increases in thickness. When the hearth contour is of the proper shape and elevation, the bottom is "slagged"; in this step the banks of the hearth are dusted with ground basic open-hearth slag which fuses and runs down to form a pool on the bottom. This pool shortly is drained off through the opening at the tap hole. The hearth then usually is chilled, after which the furnace must be brought back to operating temperature. Any cracks which appear in the burned-in bottom then are filled with fine magnesite, after which the furnace is charged.

2. Rammed Magnesite or Plastic Chromite Sub-Hearths with Burned-In Working Hearth. The rammed or plastic type of bottom is being employed to a considerable extent, the rammed subhearth replacing an equivalent thickness of burned-in magnesite, thus permitting faster installation which makes it possible to charge the furnace sooner than in the case of a completely burned-in hearth.

The special, prepared, grain-sized, cold-setting magnesite or dolomite-magnesite materials, containing 5 to 6 per cent moisture, generally are rammed to a depth of 6 to 10 inches with pneumatic rammers, while the plastic chromite mixtures are installed in thicknesses ranging from 3 to 6 inches. On top of the rammed-in bottom

material, after raising the temperature in the furnace chamber to from 2900° to 3000° F and maintaining this for 16 to 24 hours, the top layer of the final bottom, from 3 to 10 inches in total thickness, is burned in with grain magnesite, layer on layer, as described in the previous section on burned-in grain magnesite bottoms.

3. All-Rammed Bottoms. Many of these bottoms have been installed in recent years, as they require a shorter time for installation than other types, and provide more accurate contours. The specially grain-sized magnesite mixtures mentioned previously are rammed in over the top of the brick courses, using forms and careful tamping procedure. When ramming is completed, the bottoms are dried with flames from gas burners, after which they finally are heated to approximately 3000° F for about 48 hours, after which any cracks that develop are filled with ramming material. The hearth then is "slagged" as described previously, chilled, and again dressed with the fine refractory material, after which it is brought up to operating temperature for charging.

The foregoing procedures describe the installation of a new hearth. Such a new hearth will last for possibly five years or more. Its life thus covers the making of possibly 3,000 to 6,000 heats, and is called a "campaign." After each heat is tapped from a furnace during a campaign, it usually is necessary to make at least minor repairs to the banks and bottom. The techniques employed for making these repairs to maintain the furnace in good operating condition are described in a later section.

Basic Hearth Tap Hole—The tap hole, located midway between the ends of the hearth and at a point in the back bank toward which the hearth slopes, should have a continuous fall (about 1 inch per foot) from the lowest level of the bottom. It is about 6 inches in diameter and is formed by first placing a steel pipe through a circular hole in the hearth brickwork which is lined with basic brick. The opening between the pipe and the basic brick is rammed with magnesite, magnesite and chrome ore, or basic bottom ramming material. Before the furnace is charged, the tap hole is filled with burned dolomite, being faced with this same material on the inside of the furnace. In the rear, or tapping-spout end, a mixture of burned and raw dolomite is tamped into the hole. A plug of clay loam completes the sealing of the rear end. Tap hole and spout locations are shown diagrammatically in Figure 15-17 mentioned previously.

A casting on the ladle-pit side of the furnace, known as the **tapping-hole casting**, provides a seat for the **tapping spout** against the tap hole, so that the flow of molten steel from the tap hole through the tapping spout to the ladle is smooth and free from undue turbulence. The tapping spout is formed from a steel plate formed into a U-shaped trough, which is lined with fireclay brick and may be surfaced with fireclay, magnesite, magnesite and cement, magnesite and chrome ore, occasionally with graphite in the mix. A refractory joint of plastic refractory is made between the spout and furnace.

When the percentage of molten pig iron in the charge is high enough to require a slag run-off or flush-off in the period following the addition of the molten metal, provision is made in the back bank and through the back wall, between the tap hole and the end of the furnace, for a **run-off notch**. This is to provide an opening through the back wall through which the early slag may be run off through a spout and into a slag pot. In the case of several modern installations, the sill plate of the middle furnace door is designed to permit the drainage of slag from the furnace so that it may be removed from both the back and front of the hearth (see Section 4).

The Acid Hearth—The hearth of an acid open-hearth furnace is similar in contour to that of a basic open-hearth furnace, the difference in construction being that acid brick instead of basic brick are used in the hearth structure and only acid material is used above the hearth brickwork.

The average capacity of acid furnaces in the United States is approximately 60 tons, and the largest acid furnaces are of 125 tons capacity. A modern acid open-hearth furnace would have the same steel bottom pan as used in the latest designs for basic furnaces.

One and one-half inches of insulating concrete may be used next to the steel bottom plate. Fireclay brick next are laid to a depth of 5 to 11 inches. Over the fireclay brick are laid silica brick to a thickness of 11 to 12 inches. On top of the silica brick, the upper part of the hearth, which receives the charge, is built up with fritted or sintered silica sand to a minimum thickness of 9 inches at the tap hole, and to an increasing thickness progressing towards the ends of the hearth. The drying and heating of the acid furnace preliminary to making the bottom of silica sand are conducted in the same manner as for the basic furnace. Before the furnace is heated to too high a temperature, the tapping hole must be put in shape.

Acid Hearth Tap Hole—In most furnaces, the opening left in the brickwork for a tapping hole is square in section, except at the top, which is arched. This opening, measuring about sixteen inches across, must be lined with some acid refractory material that will set hard and resist the eroding effect of the liquid metal and slag. To form this lining, a tapered plug of iron or wood, from four to ten inches in diameter at the large end, varying in accordance with the size of the furnace, is inserted centrally into this hole in the brickwork, and the space between the plug and brick is rammed full of ground ganister (silica sand) containing a little plastic clay as a binder. When this mixture has set, the plug is removed, and the round opening is filled with anthracite coal, which is covered immediately on the inside of the furnace with a little sand, so the coal will not be burned out during the heating of the furnace. Outside, the coal is held in place with a plug of fire clay. Some shops prefer to use a graphite plug, such as might be made from a used electric furnace electrode. In this case, the plug is surrounded by silica sand and is left

in place while the bottom is being made. The tap hole is approximately 6 to 8 inches in diameter. Some melters prefer to tamp the hole full of ganister containing 5 to 7 per cent moisture, and then dig out with a sharp, curved spoon some of this filling to form a small cylindrical-shaped hole, which is left open until after the bottom and banks have been fritted in. This hole is then stopped as described above.

Making the Acid Bottom—After the tapping hole has been made ready, the heating of the furnace is carried on as described under the basic process. In the meantime, preparations are made for fritting or glazing the sand bottom and banks. As a first step, enough sandstone or granite chippings, if such chippings are procurable, are scattered over the bottom and banks to cover the surface. Then, as soon as the furnace has come to a red heat, more fuel is admitted and the valves are reversed at about twenty-minute intervals until these chippings begin to fuse. This procedure permits the bricks in the banks and bottom to absorb more heat than if sand is spread directly upon them, and the partly fused chippings form a better bond between the almost infusible silica brick and the sand fritting. By means of shovels, rabbles, and long-handled spoons, silica sand is now spread evenly over the surface of the bottom and banks to a depth of about one-fourth inch, and the heating is continued until this sand begins to fuse, thus forming a glaze over the entire surface. Another layer of sand is then spread and glazed on as before, and these operations are repeated until the bottom and banks have been built up to the desired thickness, which varies, according to the size of the furnace, from nine to twenty inches. The bottom may now be compared to a big oval dish, the semi-vitrified layers of sand forming practically a solid one-piece wall about the sides and bottom.

After the bottom and banks have been fritted properly, or glazed in, the tapping hole is punched through from the outside and again made up as before, using a mixture of anthracite coal or coke and sand, and securing this in place with a plug of fire clay rammed in from the outside, at some shops. The hearth then is filled nearly half full of acid open-hearth slag. If the slag is not at hand, red brickbats or other easily fused siliceous material is substituted, and some light scrap, such as sheet trimmings, is scattered on top. Other shops prefer to add only as much slag as the bottom and banks will absorb. When this charge has melted, rabbles are inserted through the doors and the liquid slag is splashed up against the banks to wash them down. Then the bottom itself is raked over carefully to smooth it, after which the tapping hole is opened and the slag is drained out of the furnace. This treatment tends to consolidate the bottom and banks, and make them much more resistant to the buoyant action of the liquid metal of subsequent charges. According to some practices, the next two or three charges will be small, not over two-thirds the capacity of the furnace, and will consist mainly of cold pig iron. Then, if it is apparent that the bottom is holding well, the size of the charge and the proportion of scrap used may be increased gradually until the furnace is working on its normal charges. Some melters prefer to test each layer of the bottom as it is fritted in by scraping it with sharp rods. When each layer has shown a hard glazed surface, and examination of the material removed in shaping up the tap hole indicates a firm vitrified bottom, the first charge may be of normal size.

The quality of the sand used for this purpose is of very great importance. If the sand is very pure, that is, composed almost entirely of silica, great difficulty is

Table 15—II. Chemical Compositions of Different Sands Used for Acid Open-Hearth Bottoms and Banks

GRADE	Good	Bad	Fair	Fair	Good	Mix*	Mix*
Composition	(%)	(%)	(%)	(%)	(%)	(%)	(%)
Silica, SiO ₂	95.83	88.46	97.38	95.01	96.34	98.45	92.36
Iron Oxide, FeO	0.40	0.96	0.45	0.51	1.05	0.13	2.90
Alumina, Al ₂ O ₃	2.76	8.27	1.48	3.44	1.60	0.85	2.85
Lime, CaO	Trace	Trace	Trace	0.10	0.21	Trace	Trace
Magnesia, MgO	0.13	0.23	0.08	Trace	0.21	Trace	Trace
Alkalies, K ₂ O + Na ₂ O	None	None	Trace	Trace	None	Trace	Trace
Ignition Loss	0.88	2.08	0.64	1.05	0.62	0.52	1.57

* These sands may be mixed in about equal proportions.

experienced in fritting it on the bottom, while if it contains too high a percentage of impurities, it will be fritted easily but also will be worn away rapidly after the furnace is put into operation. The silica content of the sands, or sand mixtures, that gives the best results is usually between 94 and 97 per cent. No definite figures can be given for the composition of a good sand, because the adaptability of the sand for this purpose depends not only upon the kind and relative amounts of impurities present, but also upon the physical make-up of the sand itself. Certain natural sands, including some beach sands and a few deposits in Ohio and other mid-western states, give the best results. In Western Pennsylvania some of the most desirable material occurs as hard sedimentary rock, which has to be crushed to render it suitable for this purpose. In fineness, this stone is crushed so that all will pass a one-fourth inch screen. Frequently a relatively coarse and pure sand mixed intimately with a certain proportion of a fine and less pure sand will give good service as bottom material, even when either sand alone has proved unsatisfactory. Table 15—II has been found helpful in the selection of suitable sands for this purpose.

Front Wall and Doors—The walls of all open-hearth

furnaces begin on the top course of the brick which form the upper rim of the hearth. The front wall extends from the door sill line to the bottom of the skewback channel that supports the skewback brick on which the roof is supported. The walls, in the larger modern furnaces, are generally at least 13½ to 18 inches thick, being thickest at the base of the wall. Basic brick for basic furnace walls usually are laid up to 12 to 15 inches above the sill line. The courses of brick above these are usually of silica brick where brick arches above the doors are employed and of basic brick when archless door frames are used. With silica walls, it is preferred that the underlying basic brick be chrome-magnesite, their composition being predominantly chrome ore, to limit fluxing of the silica brick. Front walls built entirely of basic brick often employ steel-encased magnesite-chrome brick in which magnesite is the predominating ingredient of the composition. These are "tied" to the furnace buckstays with steel plates laid horizontally between selected courses, usually by spot-welding them to the buckstays, to provide greater stability and resistance to mechanical abuse (See Figure 15—18). Silica brick front walls sometimes are insulated by applying plastic clay-bonded exfoliated vermiculite by a spray gun or trowel to a thickness of 1 to 2 inches.

When the walls are built of basic brick, it is not usual to insulate them, but rather to seal them against air infiltration by using a thin coating of the materials ordinarily used for insulation.

Modern stationary furnaces generally are equipped with five doors. Recently, some enlarged furnaces have had the number of doors increased to seven. Practically all doors in use on open-hearth furnaces are water-cooled, and generally are lined with used or new fire-clay brick, although increasing use is being made of plastic chrome ore, rammed on steel studs set in the steel door plate (Figure 15—19).

A peep hole, called the wicket, is provided on each

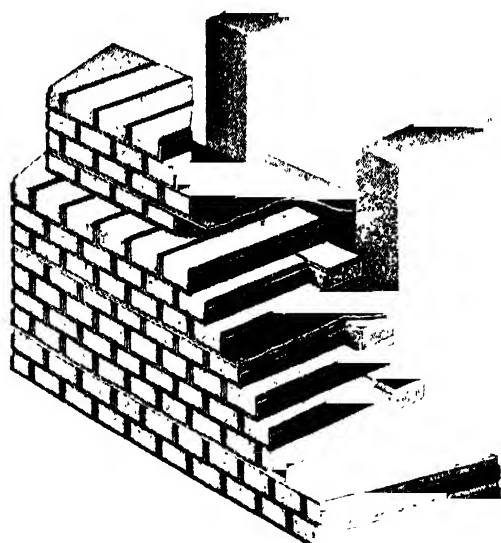


FIG. 15—18. Sketch showing how steel plates are inserted between every fourth course of metal-encased basic brick and then welded to bars welded to the solid buckstays to make a more stable front-wall construction for an open-hearth furnace. Note increased thickness of wall near the bottom. (Courtesy, Harbison-Walker Refractories Co.)

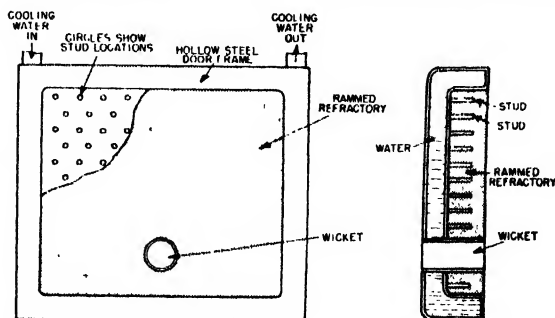


FIG. 15—19. A hollow steel casting, through which cooling water is circulated, has a studded recess which is rammed with a plastic refractory to make one type of an open-hearth furnace door.

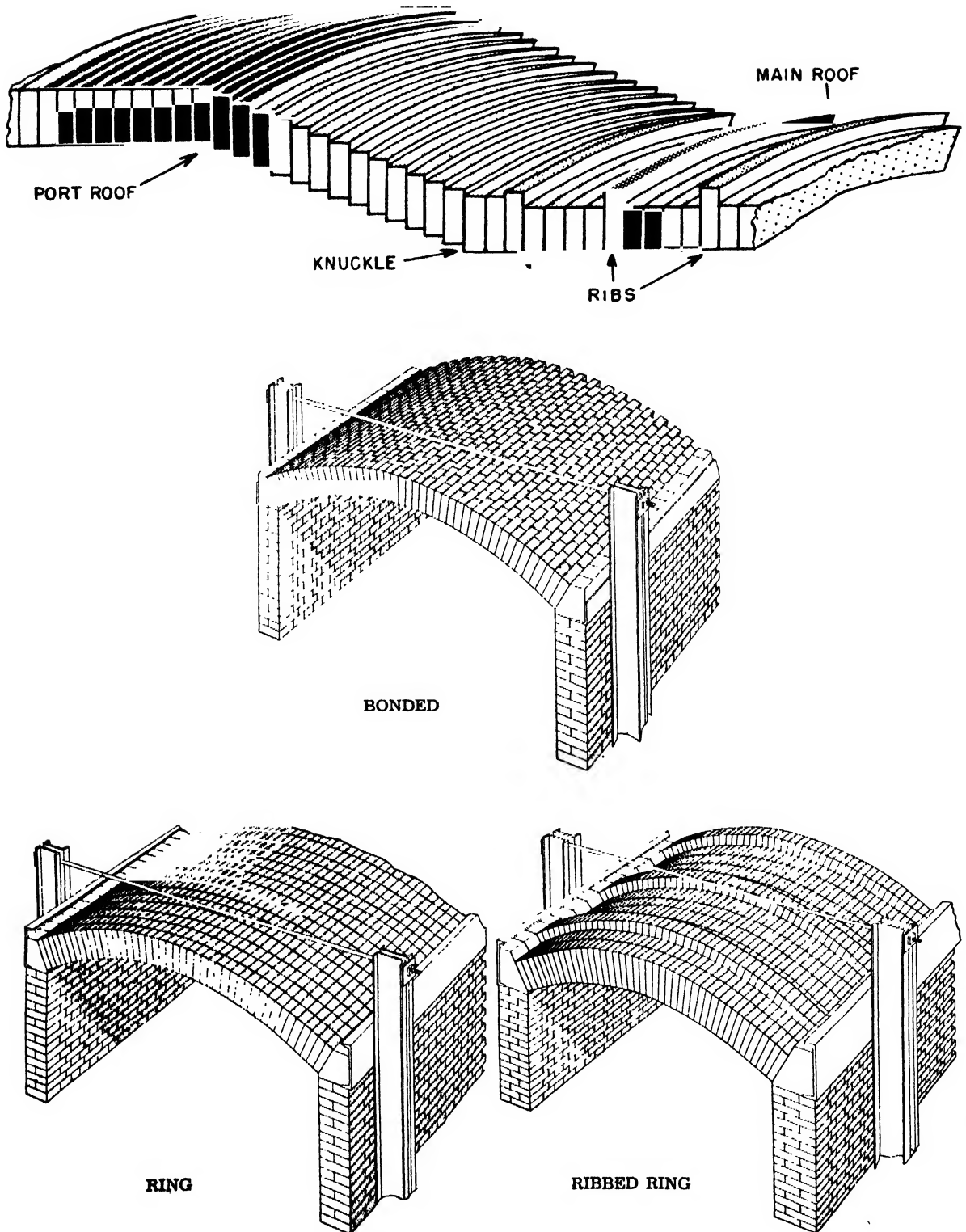


FIG. 15—20. At the top is shown a section of an open-hearth-furnace main roof, illustrating the ribbed-ring-type of construction. Joints between brick in the rings and ribbed rings are omitted. Note that port roof is of ring-type construction, but not ribbed. Below are three schematic drawings showing differences between the "bonded," "ring" and "ribbed-ring" methods of laying roof brick. These are not sketches of an open-hearth roof, and are intended only to illustrate the three methods of bricklaying.

door to provide for inspection of the interior of the melting chamber, and to provide access to the furnace for dipping out molten samples, for stirring, for poling, and for taking temperature measurements.

The doors slide on water-cooled welded-steel frames which are placed between the front-wall buckstays and against the front wall. The doors are raised and lowered by electric door hoists in modern installations.

The front and back walls of acid open-hearth furnaces are built entirely of silica brick, 13½ inches thick.

Back Wall—The back wall may be vertical, or so-called full sloping (40 to 50 degrees with the vertical), or a compromise between the two. Modern practice dictates a slope which will provide easy maintenance through the doors of the furnace (Figure 15–17). In this case, the back wall slopes in such manner as to be essentially a continuation of the sloping back area of the furnace banks. Underlying the hot face of the sloping back wall are about 13 inches of basic brick, or of basic brick over firebrick. Vertical back walls, like the front walls, may be all basic or silica in the upper sections. Magnesite-chrome steel-encased brick are used quite widely. Back walls, particularly of the vertical type, may be insulated the same as front walls if the brick are silica, restricting the coating to a sealing coat if basic brick are used (see the description of front walls, above).

Main Roof—The main roof of the furnace arches over the hearth from front to back wall between the vertical planes which intersect the junction of the port roofs with the main roof as the port roofs slope down to meet the latter. These planes define where the port ends and the furnace chamber begins. Usually, this line of junction of main roof and port roof is called the **knuckle** (see Figures 15–15 and 15–16).

Silica brick are used almost universally to construct open-hearth furnace roofs, although operating tests have been conducted on basic furnaces whose roofs are built of basic brick set in a sprung arch with the brick suspended and pressure applied to the back skewback by springs. Both **bonded** and **ring** silica brick roofs are used in different thicknesses or combinations of thicknesses, although 12, 13½, 15 and 18 inch roofs are most common.

In the bonded roof, the brick are overlapped to “break” the joints, while in the ring roof, the joints are not broken. Ring roofs may be plain or ribbed. It is common practice to employ the ribbed construction for greater stability. As may be seen in Figure 15–20, the ribs run from front to back of the furnace. The ribs may be 3 to 6 inches thicker than the rest of the roof and extend above the regular roof brick at every third, fourth or sixth ring. The location of the rings depends on the width of the roof span.

In some shops, basic brick are used in the roof adjacent to the front and back skewbacks, generally in alternate rings with silica brick.

The rise of the roof arch is generally 1½ to 2 inches per foot of span. **Skewback brick** (Figure 15–17) which take the thrust of the arch at each of its ends, should be designed very carefully so as to have the angle best suited for distributing bearing pressure, and there should be the minimum number of shapes in one skewback line. In some plants, sloping channels are used to replace the shaped skewback brick.

Roof design and thickness require careful consideration in each plant with respect to maximum economical life as related to the fuels commonly used and normal operating practices. All other parts of the furnace should be designed so as to be “balanced” with roof life and thus require replacement in intervals that are

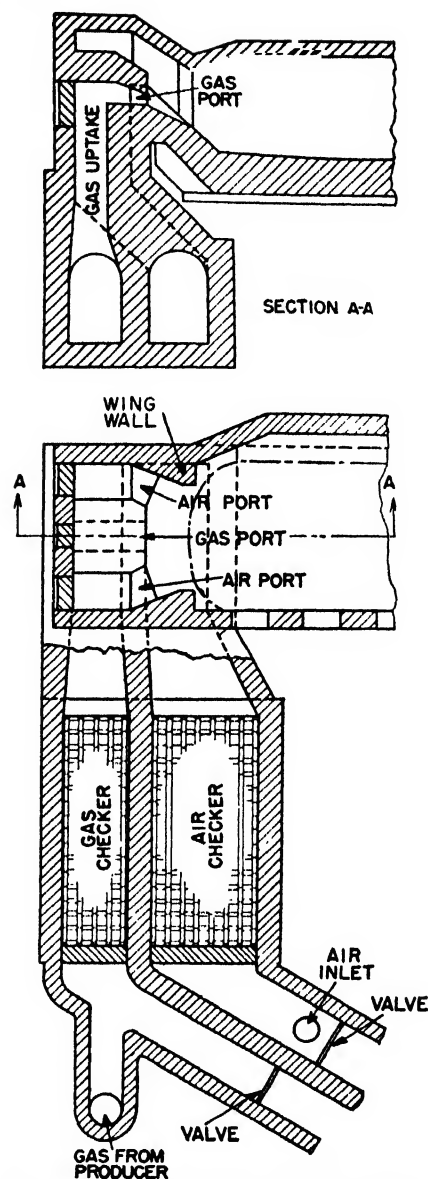


FIG. 15–21. Schematic diagram (not to scale) showing arrangement of checker system for preheating both air and gaseous fuel for an open hearth. Only half of the furnace and the flues on one side are shown, since the other side of the plan is exactly similar. Upon reversing, the air inlet and gas inlet are both closed and the two valves opened, giving access to the stack, while exactly the reverse operations are performed at the opposite end of the furnace.

even multiples or fractions of roof life. This desirable condition can be promoted through careful selection of refractories and their proper applications.

In producer-gas or natural-gas fired furnaces, of which relatively few installations remain, the ends of the main roof slope to form, with the port roof, a V at the knuckle (Figure 15–21). The roofs of oil-fired furnaces are designed to form a shallower V at this point, a higher level at the knuckle, and a straight contour from knuckle to knuckle (see Figure 15–15). Different types of water-cooled skewback channels are employed

frequently, particularly at the front-wall side of the roof.

Silica roof brick are bonded with silica cement or may be laid dry with a surface grouting of silica cement. In the relatively small number of cases when insulation is applied to furnace roofs, exfoliated vermiculite is employed either as a loose blanket or in plaster form, although an intervening layer of sand or crushed silica brickbats containing a small amount of lime may be used. The total thickness seldom exceeds 3 inches.

Acid open-hearth furnaces have roofs built of silica brick, 12 to 15 inches thick.

Port Ends—The port ends (Figure 15—16) include the port side and end walls, fuel ports, port roof, port slope, and uptakes from slag pockets to the port roof. Some furnaces have wing walls (also called monkey walls) built entirely of chrome-magnesite brick, or faced with these brick and backed up with fireclay brick; they form part of the throats where the gases from the ports enter or leave the hearth chamber and may be water-cooled to maintain their contour, with the consequent effect of maintaining control of the flame.

The end walls may be built completely of silica brick, 13½ to 27 inches thick, or of basic brick from slightly below charging floor level to the port roof, in which case the walls are 9 to 13½ inches thick. The basic brick may be burned or unburned magnesite-chrome and preferably steel-encased, or at least, laid with steel plates which are tied to the port-end buckstays every few courses.

Fuel ports on furnaces fired with fuels other than producer gas consist of water-cooled burners which are usually, but not always, covered by a doghouse of basic brick or silica brick covered with chrome ore, the whole generally supported on a solid, silica-brick arch dividing the uptake on each end of the furnace in the direction of the longitudinal axis of the furnace. Gas may be introduced through the same water-cooled burner as liquid fuel, or may be introduced through shaped brick at the sides of the ports near the uptakes.

Producer-gas fired furnaces require separate uptakes and regenerator chambers for the gas (Figure 15—21). The producer gas ports are of various designs, but the most popular design consists of a water-cooled steel arch-shaped "tank," lined inside and out with silica brick, although the outside surface may be covered with basic brick or chrome ore, particularly at the nose or outlet end. The gas uptakes may be lined with basic brick for some distance below charging floor level and built elsewhere of silica brick. Air uptakes may be built similarly.

The port roof usually is built of silica brick, with a minimum roof thickness of 9 inches except at the knuckle or junction of the port roof slope with the main roof, where the thickness is the same as the main roof (Figure 15—16). The port slopes or floors from edge of uptake to end of hearth are built similarly to sloping back walls with fireclay brick covered with basic brick, followed by a layer of magnesite bottom mixture or chrome ore.

Maintenance of port ends and furnace hearth ends is facilitated greatly by good design in the wind box or chill, indicated in Figure 15—15. This is a triangular-shaped steel box open at both ends, extending at both ends of the furnace across the furnace between the hearth side of the uptake wall and the end of the hearth, which portion of the hearth it supports.

Success has been attained with a port end built entirely of basic brick. Magnesite-chrome steel-encased brick are used, suspended, in both port-end roofs and vertical walls.

Insulation of end walls and port side walls is more common than on other wall surfaces above the charging-floor level and, in general, consists of a sprayed or troweled coat of plastic vermiculite, 1 to 2 inches thick in the case of silica brick and a sealing coat in the case of basic brick.

Modern acid furnaces have port ends (roof, side walls, ends) built of silica brick, the walls 13½ inches to 9 inches thick and the port roof 12 inches thick.

Slag Pockets—The slag pockets extend downward from the bottom of the uptakes (see "Port Ends") to a level which provides sufficient volume for the accumulation of oxides. These oxides are in the form of fine dust carried by the hot exit gases that have passed over the charge. The oxides are mainly iron oxide, but may carry other dust; for example, lime after an addition of burned lime to the furnace. The oxides are deposited from the gases descending from the outgoing port without blocking the entrance of these gases to the fantail flues and regenerator chambers. The walls of the slag pocket are built of silica or fireclay brick or, often, of silica brick backed by fireclay brick walls, usually insulated. False walls of loosely laid used brick generally are used inside the silica brick walls to facilitate slag removal. Slag pocket floors are covered with a layer of sand over hard-burned, low-duty fireclay brick. Location of the slag pockets is shown in Figures 15—14, 15—15 and 15—16.

In a number of cases, the bottoms of the slag pockets are built so as to incorporate some mechanical means of loosening and breaking up the oxide deposit, including means for inserting explosives, so as to shorten the time of cleaning out the pockets at the time of furnace rebuilding.

In the case of acid open-hearth furnaces, the silica brick extends on down from the port ends and uptakes to meet fireclay walls extending from the bottom of the slag pockets half way to the charging-floor level.

The fantail flues (Figures 15—13 and 15—16) are built of silica or of fireclay brick. The slag pockets and fantail walls are insulated with 3 to 4½ inches of diatomaceous or fireclay insulating brick or vermiculite blocks, which may be coated with an inch of plastic vermiculite. This is used, as a rule, on the fantail arches to a depth of 2 to 3 inches. From the fantail, the gases flow to the regenerator chambers.

Regenerator (Checker) Chambers—To obtain sufficient flame temperature and economical fuel consumption, the air for combustion must be preheated and where fuels of low calorific value are used (such as producer gas), the fuel also must be preheated. The preheating is accomplished by a regenerative system which consists of brick chambers partially filled with brick which take up about 60 to 75 per cent of the chamber volume and which are known commonly as checkerwork or checkers. The brick comprising this volume are arranged so as to leave a great number of passages through which the waste gases from the furnace pass on their way to the stack and, later on, as the flow is reversed, the air for combustion and the fuel gas (in the case of producer gas) pass in the reverse direction to be heated on their way to the uptakes and the furnace ports. If producer gas is used, both the air and gas are preheated in separate regenerator chambers, uniting and burning only when they enter the furnace (Figure 15—21). With natural gas, coke-oven gas or liquid fuel, only the air is preheated. The checkerwork and the chamber, flue, uptake, and port walls abstract a large part of the sensible heat from the outgoing waste gases and return it later to the incoming air or to the gas which is being preheated. Since a large number of furnaces now in operation were built originally to use pro-

ducer gas, they were constructed with two regenerator chambers at each end of the furnace, the smaller chamber preheating the gas and the larger the air. Such chambers find use even when employing fuels of higher calorific value, such as natural gas, because all of the chambers are then used to preheat air. Furnaces converted from producer gas to liquid fuel continued to use all chambers for preheat and the air for combustion passed through all of them. With later designs not intended for fuels requiring preheat, modifications have been made to arrange the chambers to obtain the best preheat of the air.

The principal parts of the regenerator system are:

- (1) The chambers enclosing the checkerwork.
- (2) The checkerwork.
- (3) The rider wall system which supports the checkerwork and forms the passages conducting the cooled gases from the checkerwork passages to the exit end of the chamber, or, in reverse, conducting the air from the air-inlet valves and flues to the checkerwork.

The entire system of regenerator chambers, including the roof, checkerwork, side walls and flues, generally is built of high-heat-duty fireclay brick, although silica brick may be used in the top courses of the checkerwork and lower quality clay brick in the flues from chamber to stack. There are a large number of checkerwork designs involving the method of laying the brick, the brick size and shape, and the method of allowing horizontal and vertical openings. Simple brick shapes that can be reused are most in favor (Figure 15-22).

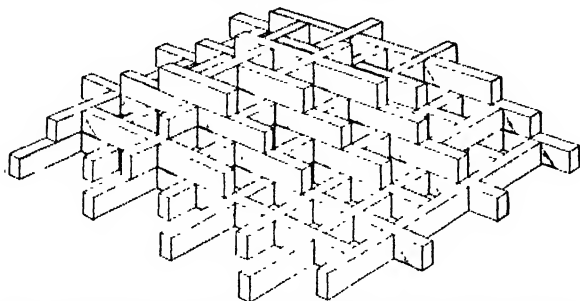


FIG. 15-22. One type of checkerwork construction, showing how simple, rectangular refractory shapes are laid in an interlocking manner to provide a series of smooth, vertical flues.

The brick of the checkerwork usually are laid with fairly large openings, generally 6 inches (or more) square on a horizontal plane, which permits easy cleaning during furnace operation, thus retaining a high regenerative efficiency during a furnace campaign. The regenerator chamber roof is generally a sprung arch although, recently, suspended arches have been used successfully. The latter construction permits a greater checkerwork volume because of greater permissible height of checkerwork and the opportunity to eliminate the division wall for furnaces not using preheated fuels. For maximum efficiency of regeneration and, consequently, of furnace performance, air infiltration through the walls of regenerator chambers, slag pockets and furnace ends must be kept to a minimum. All of these walls should be well sealed. The regenerator chamber walls usually are insulated with $3\frac{1}{2}$ to 4 inches of block insulation of vermiculite, magnesia, or asbestos composition. The outside of the chamber walls, including the block insulation, often are encased with light steel plates. When steel is not employed, the insulating blocks are coated with asbestos or vermiculite cement for seal-

ing. These plastic coatings, or insulating blocks plus coatings, are used to a depth of 3 to 4 inches on the chamber roofs. Vermiculite or diatomaceous concrete may be used on the floors of the chambers. Insulating bricks or blocks of vermiculite or asbestos may be used on the flue walls and arches.

Forced Draft Fans—A forced-draft fan (Figures 15-13 and 15-16) is the most important auxiliary of relatively recent adoption for modern furnaces. When operating on natural draft, the only force causing air flow into the furnace ports from the regenerative system is the heat head or stack effect of the checker chambers and uptakes. Thus, not only the temperature of the air varies from one end of the air cycle to the other, but also the quantity of air. In addition, since the checkers are under a pressure less than atmospheric because of natural draft, air infiltration is induced.

It should be noted that the actual positive and negative pressures within the furnace referred to in discussing forced draft and natural draft are only slight, amounting to only a few hundredths of an inch of water pressure.

The benefits of forced draft are threefold:

(1) The volume of air entering the furnace through the regenerators is maintained constant at a given setting, being influenced only by changes in motor and fan speed or valve settings.

(2) The delivery of air under slight positive pressure to the furnace system by a fan lessens air infiltration through flues, checker chambers, fan tails and slag pockets during the cycle when air is passing through on its way to the inlet port.

(3) Delivery of air under pressure to the furnace constitutes an automatic warning system in the sense that it will cause a decided outflow of heat into the furnace cellar in case of leaks in the auxiliary system. This factor promotes prompt attention by maintenance crews to any leaks that may develop and keeps furnace efficiency at a constant high level.

Flues—The type of flue generally used for modern open-hearth furnaces between the regenerative system and the stack or waste-heat boiler is one which has a relatively flat arched top and is supported by a horizontal concrete pad under the flue with concrete retaining walls on the sides. They generally are built with $13\frac{1}{2}$ inch walls and arches of second-quality fireclay brick. On the top of the arch and between the side walls and the concrete retaining walls is used a layer of insulating concrete or similar material having an insulating and sealing effect. The floor of the flue is paved with second-quality fireclay brick, and between this and the concrete pad underneath is a 6-inch layer of insulating concrete.

Valves—Figure 15-23 shows a modern flue and valve system for an open-hearth furnace. To cover the case of a furnace which uses producer gas for fuel, which must be preheated, the diagram shows the position of the producer-gas inlets to the flue system and the gas-reversing valves, as well as the usual air reversing valves, air-fan connections to the latter, stack valve, and waste-heat boiler valve.

On the left side of the diagram, the source of producer gas is shown cut off from the furnace system since the slide valve "A" is down. Both regenerator chambers or checkers are connected to the stack because slide valves "B" and "C" are raised. The upper port in "C" is closed by the slide valve in the raised position, and air supplied by the forced draft fan is cut off from the air regenerator or checker.

On the right side of the diagram, slide valve "D" is

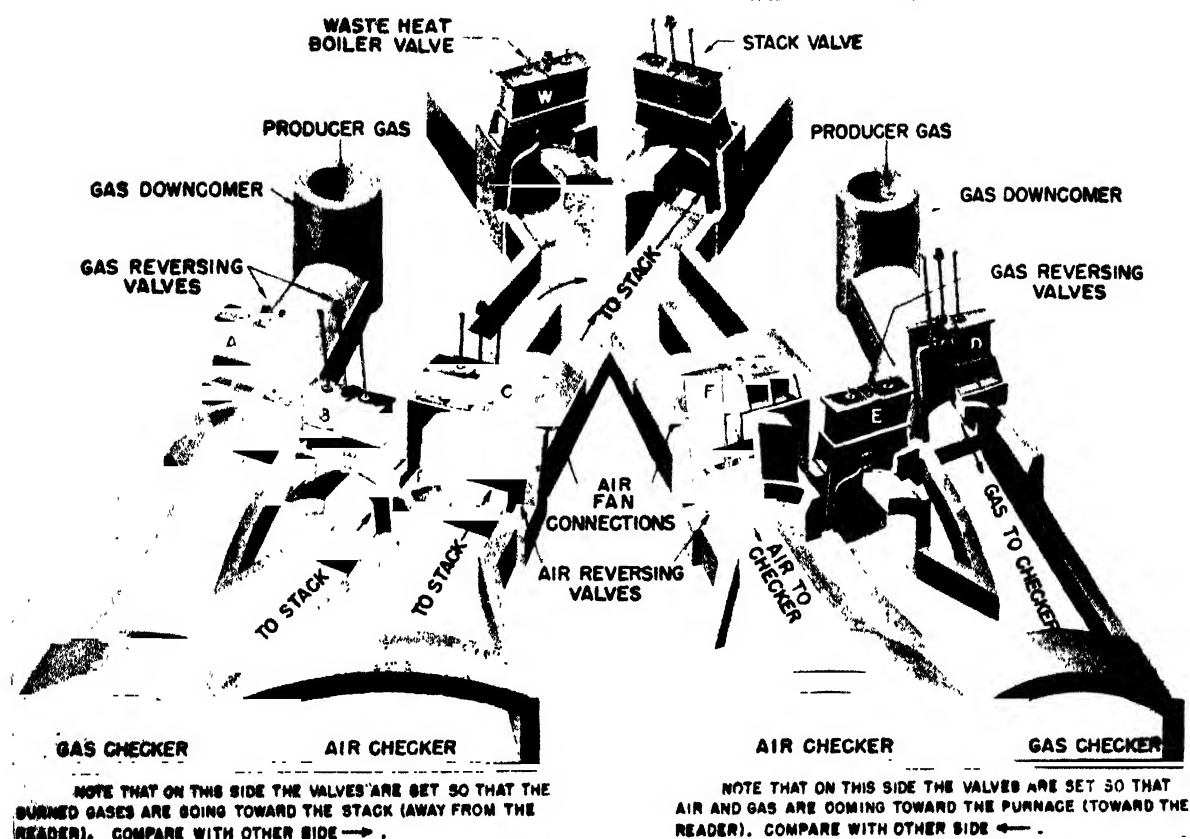


FIG. 15-23. Flue and valve system for an open-hearth furnace fired with producer gas, having the roofs and parts of the walls of the flues cut away to show valve positions and directions of gas flow. Reversing valves at left are designated in the text as Valves A, B and C, from left to right. Valves at right are Valves D, E and F, from right to left.

open, permitting flow of producer gas to the gas checker. Slide valve "E," lowered, has cut off connection of this checker to the stack, and the upper port in "F" is opened to admit air to the air checker by the same slide that, in being lowered, has cut off this air checker from the stack.

When the fuel is not preheated in the regenerative system (coke-oven gas, natural gas and the liquid fuels), valves "A," "B," "D" and "E" are eliminated, and valves "C" and "F" are the only ones needed. Of these two valves, one can serve each side of the system if the flues from the two regenerative chambers at both ends of the furnace are allowed to merge, or two sets may be employed, one valve for each of the two chamber flues at each end of the furnace to make a total of four valves, operating in pairs, in the case of the larger furnaces.

If the waste-heat boiler is in operation, the waste-heat boiler valve "W" has its slide raised to admit furnace gases to the boiler, and the stack valve "S" is lowered to permit flow of these gases from the boiler to the stack. If the waste-heat boiler is not in operation, the valve "W" is closed and the stack valve "S" is opened.

Figure 15-24 shows a simplified installation of one type of slide valves which could replace the air reversing valves "C" and "F" in Figure 15-23, for a smaller furnace using non-preheated fuel. It also illustrates the upper and lower ports of these two valves.

The sliding dampers or slide valves are fabricated from steel plates and are water cooled. The seats on which they slide are made of cast iron and also are

water cooled. The seats and dampers are machined to provide gas-tight closure.

Waste-Heat Boilers—Up to 33 per cent of the fuel input to an open-hearth furnace (gross heating value of the fuel) can be recovered in the form of steam by the use of a proper waste-heat boiler with an economizer. The 33 per cent figure is based upon the equivalent fuel energy represented by actual steam. Taking boiler efficiency into account (say, 80 per cent) the actual heat recovered equals 33 divided by 80 or 41 per cent equivalent fuel. The waste-heat boiler imposes additional resistance to the flow of waste gases and involves the use of an induced-draft fan (Figure 15-13). This fan, however, actually provides an improvement to the furnace system proper, because of the following reasons: (1) the induced-draft fan assures ample draft to remove waste gases from the furnace system almost regardless of the condition of the checkerwork. (2) Since the induced-draft fan is independent of temperature and temperature variation, and has the power to accelerate the movement of the column of waste gases, it permits more accurate control of furnace pressure.

While conventional water-tube boilers were used formerly in waste-heat recovery, the use of such boilers has been abandoned almost entirely in favor of the horizontal fire-tube type.

Waste-heat boilers, in the modern plants whose general layout was described previously, are located on the ground-level floor of the lean-to building, but, in any case, they must be located close to the stack.

Stacks—While in the case of a modern open-hearth

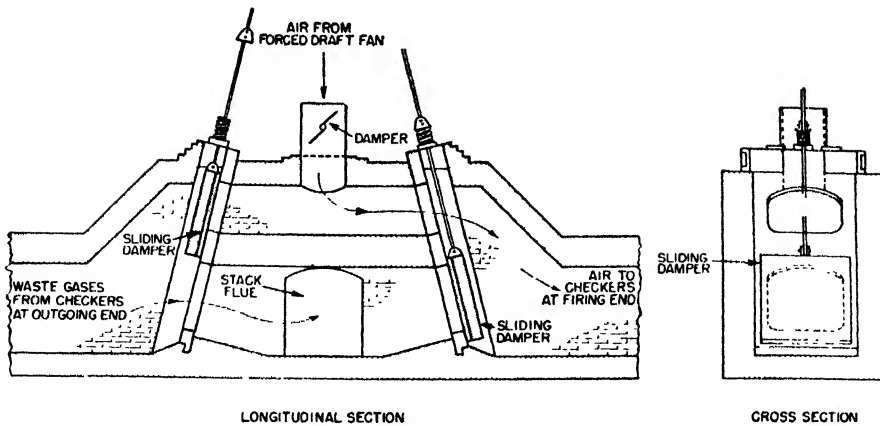


Fig. 15-24. Arrangement of one type of slide valve controlling flow of combustion air to, and waste gases from, an open-hearth furnace fired with non-preheated fuel. The two sliding dampers are operated simultaneously by the same winch, so that as one slides upward the other slides downward to reverse the directions of flow of air and waste gases through the system.

installation with waste-heat boilers, the induced-draft fans produce the larger part of the total draft effect in exhausting the furnace gases, stacks still must be provided. They must provide enough natural draft to enable the process to be carried on when the waste-heat boilers are on repair, or enough draft when the furnaces are being rebuilt and the regenerative system is being cleaned. This draft during cleaning is needed so that the regenerative system will be kept cool enough to permit entry by the repair crew for cleaning out dust under the checker work. Stacks are brick-lined steel shells, from 150 to 225 feet high above the base, with inside diameters of 7 feet in the case of modern furnaces.

A special type of stack, utilizing forced and induced

draft, forms part of a patented system (the Isley system) which utilizes two forced-draft fans and two relatively short venturi stacks (Figure 15-25). Alternately, one fan with the stack valve open is used to energize an ejector nozzle in the one stack at the waste-gas end while, with the other stack damper closed, the second fan, prevented from blowing air out of the stack because the stack damper is closed, pushes air for combustion to the furnace through the regenerative system at its corresponding end. In order to reverse the flow of air and furnace gases, it is only necessary to close the one stack valve and open the other one, and to regulate the amount of air blown by the fans to supply draft effect and air for combustion.

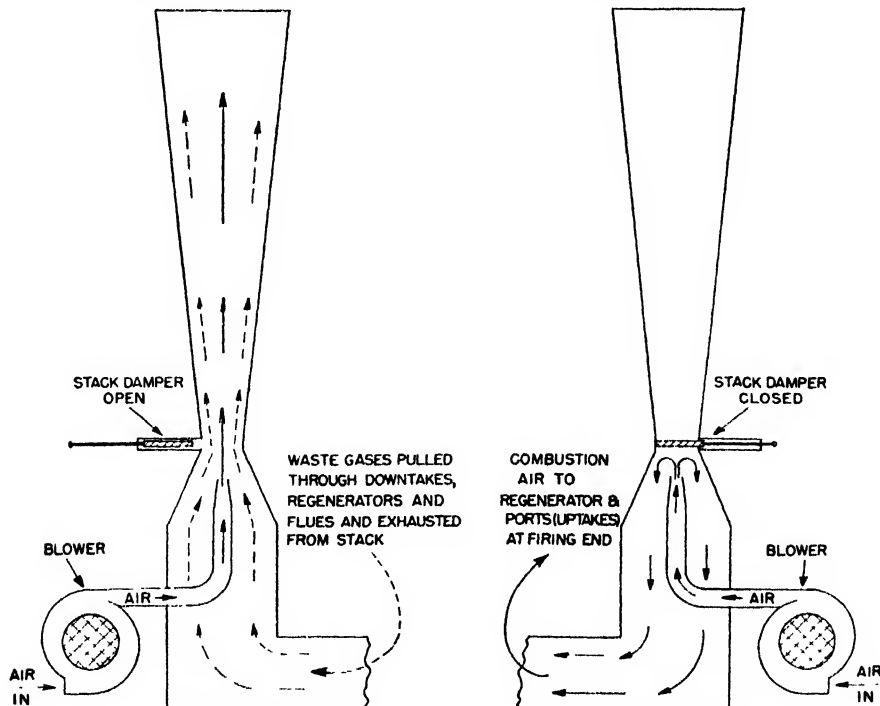


Fig. 15-25. Schematic arrangement of a special type of stack utilizing the venturi principle for controlling gas flow through an open-hearth furnace. There is a stack for each end of the furnace. Height of these stacks is considerably less than conventional open-hearth stacks that depend largely on natural draft for their operation.

SECTION 4

OPERATION OF A BASIC OPEN-HEARTH

Purifying the Metal—It was stated in preceding Section 3 that, so far as *construction* is concerned, the only important difference between acid and basic furnaces is the fact that basic materials are required in the bottom and banks of the basic furnace, while the acid process requires a bottom and banks lined with silica brick and sand. There is, however, a great difference in their *operation* and in the raw materials used for each, the difference being so great as to make a separate treatment of the two processes from this point necessary. Since the basic open-hearth process is the leading steel-making process in this country, it will be described first. The acid open-hearth process will be described in Sections 9 and 10.

Furnace Attendants and Their Duties—Each furnace requires an operating crew of three men: a first helper, a second helper, and a third helper, and supervising these men is a foreman, called a melter foreman, who has charge of a number of furnaces and their crews. Ordinarily, the first helper has charge of the furnace (with occasional checkups by the foreman) except at the tapping of a heat, and directs any repairs to the furnace hearth between the tapping of a heat and the charging of a succeeding heat or repairs during the heat.

Charging—When a basic open-hearth furnace is ready to receive its charge for the first heat after it is built (or, subsequently, when a furnace bottom or hearth has been replaced by a new one) it is usually considered necessary to use a special charge in which limestone, scrap and cold pig iron are charged in the order given. The amounts of scrap and cold pig iron are proportioned so that when the charge is all melted and the lime resulting from the calcination of the limestone has risen through the melted charge, the carbon content of the bath and its temperature will permit the finishing of the heat to make a product acceptable in the current practice of the steel works. The next three or four heats are made using a molten pig iron charge (if molten iron is available) that is less in amount than the normal molten pig iron charge currently used in the furnaces of the shop, and a correspondingly higher amount of scrap. Following these heats, the normal charge is used. The range in composition permissible in pig iron to be used in the production of basic open-hearth steel is given in Table 12—II.

After the first few charges, proportioning the amount of pig iron and scrap in the charge is usually a matter in which neither the melter foreman nor the open-hearth shop superintendent has a choice. There is a certain amount of scrap produced in the open-hearth process, including (1) that produced in the pouring of the heat into ingot molds, i.e., pit scrap, and (2) rejected ingots and ingots too short to roll, these two items comprising possibly 4 or 5 per cent; in addition there is item (3), the scrap produced when rolling the ingot and secondary products, which is, in general, possibly 25 per cent of the ingot weight. Such scrap usually is consumed in local melting rather than shipped to other company shops or sold. The limiting proportions of the total scrap and pig iron charge, and the proportionate parts of the pig iron in molten and cold forms, are governed and restricted by economic considerations. These economic considerations change with the location of the shop, with respect to the proximity of an adequate supply of raw materials, the amount of molten pig iron available, and the scrap available from integrated units.

In order to consider the matter of the charge for basic open-hearth furnaces in the United States, it is necessary to make a preliminary classification of charges as follows:

Type 1—All Liquid Pig Iron, where the blast-furnace capacity associated with a given shop is in excess of demand. This type of charge is very seldom used because it has been found better, when such a proportion of molten pig iron is available, to oxidize part of the supply of molten pig iron in a Bessemer converter and to charge the incompletely decarburized metal in a special type of open-hearth furnace (see duplex process, Chapter 17), or to speed up the melting rate of the usual open-hearth furnaces by making such blown metal a part of their charge.

Type 2—Liquid Pig Iron and Liquid Steel. This method grew out of the attempt to develop a practical and economical method of using 100 per cent liquid pig iron, as shown in Type 1 (see duplex process). Ordinarily, the charge is molten pig iron and blown metal from a Bessemer converter (where molten pig iron was charged and was blown with air). The greater proportion of the charge to the open-hearth furnace is blown metal in the case of tilting furnaces employed in the duplex process. Blown metal is used at present in stationary open-hearth furnaces for producing only a small proportion of the steel made in such furnaces.

Type 3—Solid Steel (Steel Scrap) and Liquid Pig Iron (with some cold pig iron). A quite common situation calls for the use of solid scrap and molten pig iron. Thus a mill which rolls its own steel ingots and has its own blast furnaces would use this method. Where the supply of scrap is restricted, liquid pig iron may also be used to make up such deficiency of local scrap. The amount of iron ore charged is proportioned to control the carbon content of the bath when it is completely melted.

Type 4—Solid Steel (Steel Scrap) and Solid Pig Iron. Where molten pig iron is not available locally, and where cheapness of available scrap prevents justification for building blast furnaces, the pig iron may be used in solid (cold) form. With the higher proportions of scrap, the use of burned lime as a flux may be advantageous, as mentioned under Type 5.

Type 5—All Solid Steel (Steel Scrap). Where no pig iron is available or the cost makes its use prohibitive, the charge may consist of all steel scrap. The carbon content at which the bath is completely melted is controlled by the weight of coke or other carbonaceous material charged with the scrap. The charging and working of heats of this type require considerable skill and the method is a specialized one not representing common practice.

There are some few cases in which the charge is proportioned between scrap, cold pig iron and molten pig iron in order to secure special features in the quality of the product but, in general, high quality steel may be produced from almost any of the common types of charge.

The solid parts of the charge are placed in the furnace and heated for a sufficient time before charging molten pig iron. The limestone is charged first and scrap is charged directly over the limestone except where ore is charged, in which case the latter is spread over the limestone. Sometimes several charging box loads of scrap are spread on the furnace bottom and the limestone is placed above this. In many shops there is a quantity of cold pig iron which must be used and, in practically all cases, there are pit scrap, ingots too short to roll, scrapped ingots, and broken ingot molds to be consumed.

It is very desirable in most cases to charge the solid materials as fast as the volume of this material permits. Steel scrap may vary from steel turnings and sheet shearings which are graded as "light scrap" to bloom shearings and ingot butts which are in the "heavy scrap" grade. The time required to get the solid charge into the furnace has a marked influence on the time required to make the heat of steel (taken from the beginning of charging to the tapping of the heat from the furnace). Since the scrap comprises the greater part of the solid charge and the weight per unit volume varies over a considerable range, its value as an item of the charge should be based in part on the space it occupies per unit of weight as charged in the furnace. Taking into consideration the space occupied in charging box or furnace and the desire that the total charge will become liquid at a required carbon content, it follows that light scrap requires more pig iron or less ore to be charged than does a heavy grade of scrap. In charging a group of furnaces, it is necessary to consider very carefully the proper proportioning of the amounts of light, heavy and medium scrap. Light scrap requires more charging boxes, which take up much track space and increase the time required for charging. Scrap that is relatively too heavy gives the minimum time in charging, but slower melting.

The charging practice for limestone and scrap is usually the same whether cold pig iron or molten pig iron is used. The quantity of limestone for the greater part of American practice ranges from 5 to 8 per cent of the total metallic charge; burned lime for equivalent fluxing effect being 3 to 4 per cent. The pounds of lime (CaO) required (to be added by these fluxes) must be proportioned on the basis of pounds of silicon, phosphorus and sulphur in the total charge, and silica in the limestone and ore, with additional consideration for the sulphur in the fuel and a knowledge of the quantity of lime and silica provided for slag reactions from the dolomite used as a refractory.

Before considering the amount of ore required to be charged, it is necessary to review the sources from which oxygen is available for the process. These sources are:

- Scale, rust (oxides, carbonates) on solid scrap.
- Oxidizing oxides in slags at different stages. (Oxidizing oxides are reducible oxides FeO, MnO, etc., but not stable oxides CaO, MgO, Al₂O₃, etc.)
- Carbon dioxide (CO₂) resulting from calcination of limestone.
- Oxygen in the liquid metal through direct oxidation of metal when boiling action exposes it to oxidation by the products of combustion.
- Iron ore, roll scale (including heating furnace cinder), sintered ore, nodulized ore, ore briquettes; as charged with initial charge or fed after bath is melted.

The quantity of ore to be charged must be proportioned so that, with the other sources of oxygen avail-

able up to the time the bath is melted completely, the net effect will lead to the desired content of carbon in the completely melted bath with the lime all up from the furnace bottom.

The ore charged is usually spread on the limestone. The quantity varies from 0 to 25 per cent; the high limit being required with unusually large proportions of molten pig iron. The ore charged is usually fine so that it is classified as "soft," but part or all of this natural ore may be replaced with sintered, nodulized or briquetted ore, or imported ore, some of which is harder and contains less fines than domestic ores.

When the per cent of molten pig iron in the total charge is greater than 45 per cent, the resulting higher carbon content makes it necessary to add a considerable quantity of ore after the charge is completely melted, or even to charge ore with the initial charge. With charges of above 45 to 50 per cent of molten pig iron, general practice is to charge ore with the initial charge, and then to feed ore to the completely melted charge if needed.

Solid pig iron is added immediately after the scrap charge or even after the scrap is melted down partially. Molten pig iron, on the other hand, must be added at a carefully judged moment, for the melting and oxidation of the scrap must progress to a stage which insures that the oxides start to work in oxidation of the metalloids of the molten pig iron as soon as it is charged. If the molten metal is added too soon, it will be chilled by the scrap. If the charging is delayed too long and the scrap is too highly oxidized, the carbon content of the bath, when the charge is completely melted, may be too low, and the progress in completing the process delayed. Molten pig iron is poured from the metal mixer or from large mixer type ladles into a transfer ladle. This may be carried on an electric overhead crane all the way to the furnace or it may be placed on an electric transfer buggy and transported to a point near the front of the furnace to be charged, lifted off by an electric overhead crane and its contents poured through a runner or spout inserted through one of the furnace doors by the charging machine (Figure 15-26).

Two of the more common types of charge, which should be listed under Type 3 above, are outlined below.

"Fifty-Fifty" Practice—This term refers to a charge which consists of roughly 50 per cent pig iron and 50 per cent scrap. Such an amount of molten pig iron is large enough to require the use of ore in the initial charge and, depending on the local procedures and efficiency of the furnaces in heat transfer to the charge, a run-off or flush practice is used. Limits of molten pig iron are usually 45 to 55 per cent and the charge may include a relatively small amount of solid pig iron, cast iron scrap or broken ingot molds. With this charge, equipment usually is provided for a run-off or flush practice (see "Melting Down After Molten Pig Iron Addition"). Limestone is normally used since burned lime is more expensive. Fine ore, sintered ore, nodulized ore, or ore briquettes may be used interchangeably. The amount of ore required in the charge is not sufficient to cause serious trouble from violent evolution of combined moisture (water of crystallization), unless an excess of combined moisture is present; i.e., ignition loss is over 5 per cent.

High Molten Pig Iron Practice—This includes those cases in which the percentage of molten pig iron is about 55 to 80 per cent of the total charge and leads to the use of a run-off or flush-off slag for economical operation.

A considerable amount of metallic iron is recovered

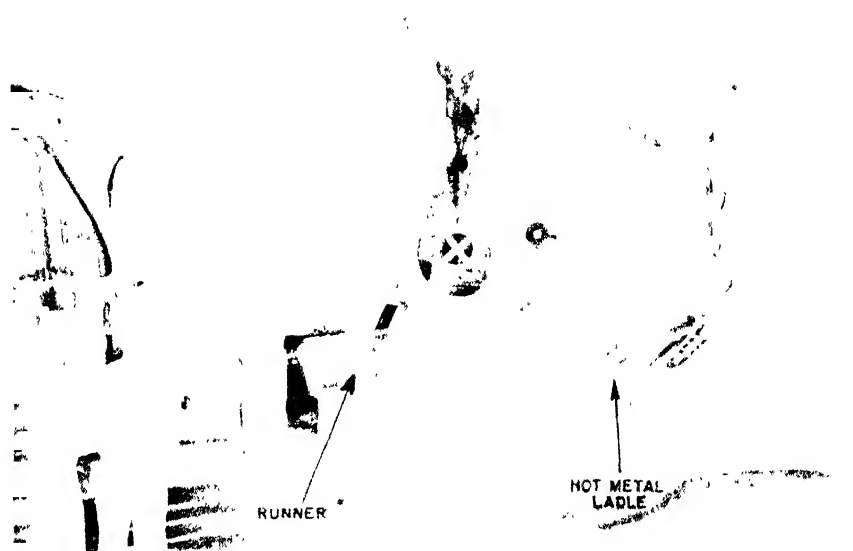


FIG. 15—26. Making the hot metal (molten pig iron) addition to an open-hearth furnace. Rate of flow of the hot metal is controlled by the rate at which the crane hook attached to the rear of the ladle near the bottom is raised.

from the ore which is necessarily charged, and this displaces scrap in the charge to the extent that necessary purchase of scrap is lessened. Because of the relatively large amount of ore used, the physical characteristics and chemical composition of the ore are critical in attaining the desired production rate and quality of product. Excessive fineness and higher silica and combined moisture contents are objectionable characteristics. The degree of fineness and combined moisture of many ores may be corrected partially by sintering, nodulizing and briquetting. The charged ore in this practice is placed on the limestone so that the latter is well covered. Since the proportion of scrap is low, it is important to oxidize the scrap well before charging the molten pig iron.

Melting Down the Charge—The melting period really begins when the first scrap has been charged. It is important to melt the scrap and other solid metallic elements of the charge quickly, and to oxidize them by sufficient excess oxygen in the flame so as to have them at such a temperature and degree of oxidation that, on the one hand, the molten pig iron charged will not be chilled by the scrap and, on the other hand, the oxidation of the metalloids of the pig iron will not be delayed by insufficient oxygen supply from the oxidized scrap. Therefore, a high rate of fuel input is desirable with a flame which transfers the maximum number of heat units to the charge over the largest possible area of the charge. This must be subject to the restriction that the roof refractories must not be fused nor the flame extend to the outgoing end of the furnace. When the solid charge is relatively cool in relation to flame temperature, the best condition for heat transfer to the charge exists and the roof will remain relatively cool. Recent

developments in burning fuel over the charge have permitted an increase in rate of fuel input and transfer of heat units to the charge, thus considerably decreasing the time from the start of charging to tapping. These developments include the use of oxygen or compressed air introduced in proper relationship to the fuel stream and the preheated primary air coming in from the "incoming" furnace port. In the case of liquid fuels, the use of more than one burner at each end of the furnace has increased the rate at which heat can be transferred to the solid charge.

Later in the melting down period, after charging molten pig iron, and still later when lime comes up, proportionately more heat units are absorbed by the roof and walls than in the earlier stages of melting when the charge was relatively cold. The fuel input must therefore be controlled carefully.

With proper attention to the techniques of fast charging, e.g., mixing light and heavy scrap, and to maintaining high fuel rate and a flame favoring efficient heat transfer, it is possible, with the aid of an experienced and skilled first helper, to have the temperature and oxidation of the charge advanced to the point which permits charging of the molten pig iron two hours after the start of charging. At this time, part of the solid metallic charge has melted and, mixed with a slag of oxidized metalloids, has trickled down over and through the charge.

Melting Down After Molten Pig Iron Addition—In the period following the addition of hot metal, important reactions take place, dictated by the chemical conditions of the process and the proportioning and timing of the charge. The chemical reactions have to do with removing carbon, manganese, phosphorus, sulphur and silicon

from the metallic bath. First to be removed are silicon and manganese. These are oxidized to SiO_2 and MnO , and become part of the slag. When these have been largely removed from the bath, the oxidation of carbon has already begun and now becomes vigorous. The carbon is oxidized to CO gas which, in escaping, causes agitation of the bath. Eventually phosphorus is oxidized to P_2O_5 and this becomes part of the slag. Sulphur is transferred to the slag and held there as CaS .

When using a high percentage of molten pig iron, the evolving CO gas causes foaming of the slag, which can begin to run from the furnace through openings provided for the purpose. The method of open-hearth operation that involves slag removal in this manner is referred to as **flush practice**. In addition to a notch or hole through the back wall that is common to all furnaces using flush practice, most modern furnaces have in addition facilities for flushing slag from the front of the furnace also. To permit **front flushing**, a notch or trough is cut in the dolomite of the front bank at the center door. Slag running from the flushing hole at the back or pit side of the furnace is carried by a spout to a slag pot set beneath the spout. Slag leaving the furnace over the bank at the front or charging side of the furnace passes through the opening in a hollow casting set in the charging floor and runs into a slag pot set on a car running on tracks at cellar level (see Figure 15-5). This car can be handled readily by locomotives. The openings for slag removal are maintained carefully at the best level to obtain a prompt and adequate slag run-off containing as little metal as possible. The openings in the banks originally are dammed with crushed dolomite, which is raked out to provide an adequate channel for the molten slag. The run-off period may last for an hour in good operating practice. From study of the composition of run-off slags, it is evident that considerable phosphorus and silicon (as oxides) and also sulphur (as sulphides) are removed by the run-off from the furnace. The oxidation of the liquid portion and the melting of the solid portion of the charge continues until the ore in the charge has completed its reaction with the liquid metal.

When carbon monoxide is generated by oxidation of carbon, the turbulence it causes is different in appearance from that caused by release of CO_2 in calcination of the limestone. The CO gas evolution or agitation is more even and gentle, and is commonly called the **ore boil**, and the more violent turbulence caused by CO_2 is called the **lime boil**. It is, of course, true that the oxidation of carbon and resulting gas evolution in the first case results from reaction of carbon with all the reducible oxides in contact with molten metal, not alone the ore, so that it could be more correctly designated as the **iron oxide boil**. This stage of the heat is suppressed in degree of oxidation when scrap is a high percentage of the total charge. Usually it is not until the slag run-off is completed that sufficient heat has been transferred downward through the charge to start the calcination of the limestone and begin the lime boil.

The melting of scrap and solid pig iron charges exhibit the same reactions as those in the practice using molten pig iron. Melting is slower because the total charge is cold at the start, and the vigorous nature of the hot metal reaction is missing. The lime boil occurs later.

One difficulty occurring with high molten pig iron practice when a poor flush is obtained is the tendency toward formation of foamy slags which, containing entrained gas bubbles, are relatively poor heat conductors. This condition results in a cold bath with sluggish action, and a lack of stirring action which further de-

creases heat transfer from the flame. This presents a difficult problem—how to transfer heat from flame to bath while absorption of heat by the bath is limited by foamy slags. Under these conditions, the liquid fuels (and even the gaseous fuels of higher specific gravity) have distinct advantages over a light fuel gas such as coke-oven gas.

As the carbon content of the bath decreases and its temperature rises, the ore boil subsides or changes character, the calcination of limestone becomes more rapid and the lime boil predominates. As it progresses, solid lime rises up through the liquid bath, which is bubbling violently, largely because of the CO_2 given off by calcining limestone; the CO_2 oxidizes some carbon to CO on its upward travel. Lime, by rising to the top of the bath, is now available to replace iron and manganese oxides in the phosphates, sulphates and silicates present and thus become part of the slag, with any excess of lime being taken up to make the slag more basic. The basic property of the slag makes it more capable of retaining both the phosphoric and silicic oxides in the same slag, and renders the phosphate less liable to be reduced with resultant return of phosphorus to the metal. Since the CO_2 released in calcination of the limestone has an oxidizing effect, limestone is equivalent in net effect to a certain amount of ore charged and, theoretically, this oxidizing power is equal to that of 60 per cent by weight of charged ore of the average quality. The lime boil has another function which helps to promote the process. It definitely helps to equalize the temperature and chemical composition of the bath as between levels from top to bottom. The ore boil also has this function but with less effect because in its phase the bath has not advanced as far towards complete fluidity and mobility.

The lime rises to the surface of the bath mostly in lumps that are appreciably smaller than those charged. In the case where burned lime is charged, it finally rises without a preliminary boil. The solid risen lime gradually dissolves in the slag formed from the earlier oxidation of Mn, Si, P and Fe, into which has been introduced some CaO and MgO contributed by erosion of the basic hearth during the early stages of the heat. Special fluxing agents are used in many cases to facilitate the solution of lime lumps in the slag, fluorspar being the one most used.

After the ore boil and the lime boil have subsided, the **working period** or **refining period** begins. The end of the melting period generally is taken as approximately the time when all the lime has risen from the bottom through a substantially melted charge. The usual indication of an approach to the condition known as "lime up" is the cessation of the lime boil.

As removal of carbon from the all-liquid bath proceeds, the melting temperature of the steel rises. This requires an increase in fuel rate to provide an increased transfer of heat to the bath to maintain proper fluidity. Efficiency of combustion of the fuel in this case is aided by a high level of preheat of the air for combustion, and this preheat level can be raised by utilizing the heat in the checkers, by reversing the flow of fuel and air more frequently.

The aims of the working period are: (1) to lower the phosphorus and sulphur contents to levels safely below the maximum level specified, (2) to eliminate carbon as rapidly as possible and still allow time for proper conditioning of slag and attainment of proper process temperature, and (3) to bring the heat to a condition ready for final deoxidation in the furnace or for tapping, with the slag having the proper viscosity and chemical composition and with the desired carbon and oxygen

levels in the bath and the proper final bath temperature for the composition and grade of the steel being produced. The slag must contain large quantities of oxidizing agents during the working period, but must be strongly basic at the end of that period. The conditions under which these aims are advanced to a conclusion depend on the grade of steel being made, whether it is to be finished as of fully killed, semi-killed or rimmed quality, and also on the carbon range specified.

The carbon content of the bath when melted is extremely important. The open-hearth process is used to produce steel with carbon contents as poured which vary from over 1.00 per cent carbon to less than 0.02 per cent. Usually, the procedure is to oxidize the carbon in the bath to a point slightly lower than the final content desired in order to allow for the inevitable increase in carbon content due to the carbon contained in the final ferroalloy additions. Modifications of this method have been used frequently in past practice but in up-to-date practices are used in a relatively small proportion of the heats of steel produced. These modifications involve oxidizing the carbon in the bath to a much lower level than is desired in the finished steel and then "re-carburizing" the metal by addition of molten pig iron into the furnace just before tapping or by adding carbonaceous material, such as coke or coal, to the steel as it runs into the ladle. In all cases, it is necessary to have the heat melt at a higher carbon content than the percentage desired when the heat is ready to tap. The general procedure is to balance the charge in the beginning of the process so that the heat will melt 0.30 to 0.50 per cent carbon above the carbon content at which the heat is to be tapped. This is based on the principle that, during the time required to oxidize this amount of carbon from the bath, ample time for controlling slag and bath composition and temperature will be available.

It was stated above that, in the melting period preceding the working period, a certain proportion of the carbon, manganese, phosphorus and silicon was elimi-

nated from the bath by oxidation. During the working period also the final controlled elimination of these elements is accomplished by oxidation, the sources of oxygen being oxygen from the furnace gases, iron oxides such as feed ore added to the bath, and the blowing of gaseous oxygen by a lance inserted under the metal surface. With respect to the oxidizing influence of the flame, a given open-hearth furnace, burning a selected fuel, has a characteristic range in oxidizing effect which changes to some extent with the life of the furnace. The limit of this oxidizing rate also is affected by the requirement that the maximum heat output must be obtained from the fuel. The oxygen from the iron oxide fed to the bath provides the predominating oxidizing effect so that it, rather than oxidation by the flame, is used to provide the major control element for regulating rate of oxidation. The iron oxide fed also hastens the solution of undissolved lime floating in the slag and generates a boiling action of the bath which assists in removing gases and controlling the temperature.

Lump ore, fine ore, ore briquettes and, to a minor degree, roll scale, are the forms in which iron oxide is added to the bath. Lump ore or briquettes float, so as to project through the slag into the liquid metal and, therefore, give relatively more contact with the metal bath than does fine ore or roll scale. As a result, oxidation occurs much more rapidly when the relatively bulky oxides are used than with the finer, lighter material. Lump ore has been found preferable but at present an adequate supply of "hard" lump ore is becoming more difficult to obtain and the procedure of compacting fine ores into sintered or briquetted masses equivalent to lump ore is receiving increasing attention. The silica content of these iron oxides is important because of the lowering of basicity of the slag by silica. The iron oxide addition causes a decrease in bath temperature, compensated for, to some extent, by the effect of the vigorous action of oxidation in providing a more efficient heat transfer from flame to slag and metal.

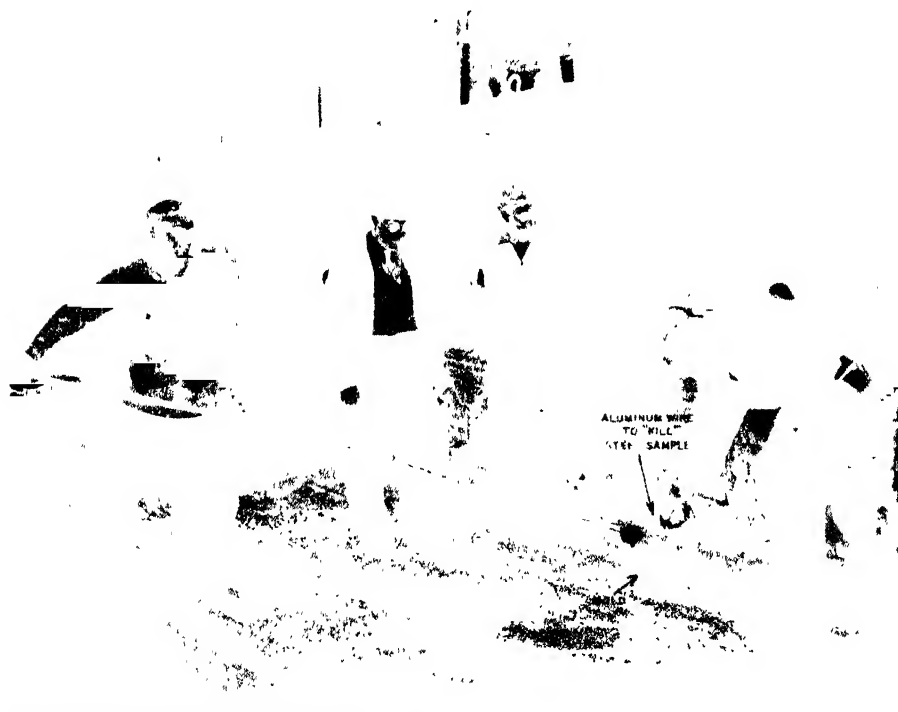


FIG. 15—27. A sample of molten steel, withdrawn from the bath by a long-handled spoon, is "killed" with aluminum as it is poured into a small mold.

When the bath receives the oxidizing agents it must be sufficiently hot to prevent freezing by the additions and must also be at a temperature sufficient to promote the endothermic reaction by which oxygen leaves the iron oxide and combines with carbon, silicon and phosphorus (see Section 8).

On the lower carbon grades of steel, it is customary in many plants to add pure oxygen in gaseous form. This is accomplished by inserting a steel lance below the metal surface through one of the furnace doors. The pure oxygen reacts rapidly and raises bath temperature; furthermore, no impurities are added. For each furnace and for each practice in making a desired product there is a rate of adding iron oxide which appears best for the combined objectives of high production rate and adequate quality level. The quantity of iron oxide added and the time between additions is governed by the rate at which heat can be introduced into the bath. If too much ore is added at one time or the additions are too close together, the bath will be chilled and its temperature will approach too closely its melting point. Such a heat would be too cold to tap and pour successfully and, because of its sluggishness, would be very difficult to heat back to proper temperature. After each addition of iron oxide has been allowed time to react, a test is taken (Figure 15—27) in order to make an approximation of the carbon content of the steel by examining the fracture of a broken, cold sample, or to obtain a more accurate carbon determination by chemical analysis, or by an instrument such as the Carbo-meter. In the last half hour or hour of the process, no iron oxide will be added. To obtain the agitation necessary to final oxidation of carbon to the level desired before adding deoxidizers to the furnace or tapping, stirring with steel rods, or the turbulence caused by inserting green-wood poles into the bath, may be employed.

On the steel samples taken soon after the heat is melted, determinations of phosphorus and sulphur are often made by rapid laboratory methods in order to determine what additions and slag-control procedures must follow in order to obtain less than the maximum content specified for these elements. Sulphur removal is relatively slow and difficult to predict and its introduction through the scrap, pig iron, fluxes, fuel and additions should be kept to a minimum. The basicity and viscosity of the slag must be controlled because certain chemical compositions and physical conditions of the slag are favorable for various stages of the heat and finally for various types of heats. Higher basicity levels are necessary to prevent subsequent phosphorus reversion at higher carbon levels. Its chemical composition may be adjusted by additions of burned lime, limestone, iron oxide or silica sand; its basicity is increased through lime additions, and it may be made more fluid by additions of fluorspar and roll scale. CaO is usually provided by burned lime, but sometimes by raw limestone. Fluorspar should be used sparingly. Slag samples may be taken by a spoon either for the purpose of obtaining samples for chemical (or spectrographic) analysis or for the purpose of observing the physical appearance of slag pancakes made by pouring molten slag on a steel plate or in a mold. From experience with slags of many compositions, this latter method of judging the basicity and the iron content of the slag has become dependable. This procedure is discussed further in Section 8 of this chapter.

The appearance of the slag and metal in the furnace, and that of the metal as poured from the sample spoon into a sample mold, are used by furnace operators to check other methods for determination of temperature,

and composition. Each slag or steel sample must be considered as revealing only part of a history of events so that a number of samples are necessary for proper interpretation of conditions and appraisal of future developments.

Actually the first step in slag control is the control of composition of the open-hearth furnace charge since this, with the other factor of the amount of silica and phosphorus removed in the run-off slag, determines the amount of limestone to be charged. Most of the elements oxidized, which combine with lime from limestone or from eroded refractories to form slag, come from the molten or solid pig iron charged. This leads to the conclusion that slag control begins with the control of blast furnace operations. In this connection, it may be pointed out that some sulphur may be eliminated from the molten pig iron on the way from the blast furnace to the open hearth by combining with manganese in the iron, if the sulphur and manganese are above certain limits, and the time of transportation from the blast furnace to the open hearth is long enough, including time in the mixer, to permit the reaction to take place under the proper cooling conditions. Thus, there may be some removal of sulphur before the chemical reactions in the basic open-hearth process take place.

The aims of slag control may be stated as follows:

1. To remove phosphorus and sulphur originally in the bath to below the desired levels in the finished steel.
2. To produce the required degree of oxidation of the bath at the end of the refining period.
3. To arrive at the proper level of oxidation of the bath in the minimum of time consistent with the steel quality desired.
4. To use the minimum quantities of lime and deoxidizers.
5. To produce the required product with the minimum quantity of iron lost in the slag.

The second aim of slag control given above, to produce the required degree of oxidation of the bath at the end of the refining period, will vary within a wide range, depending on the type of steel being produced.

There are three general types of steel, classified according to their behavior in the molds and the resulting ingot structure: rimming steels, semi-killed steels and killed steels. The production of a particular type of steel involves control of the degree of oxidation of the bath at the end of the refining period. How the oxygen content of steel affects the manner in which steel solidifies in the ingot molds is discussed in detail in Chapter 20.

Duration of Various Periods in Production of Basic Open-Hearth Steel—Based upon employing a modern, large, basic open-hearth furnace of 200 to 225 tons capacity, fired with a liquid fuel such as tar or oil, the amount of time consumed during various stages of a heat are as follows:

Melt Down (simultaneous with charging)	2.5 hr.
Hot Metal Addition	0.5 hr.
Ore Boil	3.0 hr.
Lime Boil	1.5 hr.
Working Period	2.5 hr.
Total Heat Time	10.0 hr.

After the carbon level has been lowered by ordinary methods to 0.25 per cent carbon, the use of oxygen thereafter may lessen the working period by 0.5 hr. By improved techniques for burning liquid fuels (oxygen-enriched air for combustion, etc.) the time for melt down gradually is approaching an average of about 2 hours.

SECTION 5

TAPPING AND POURING

Tapping—The finishing temperature of a basic open-hearth steel heat will be in the neighborhood of 3000° F, varying according to the composition and grade of the steel.

When the heat of steel is ready to be tapped, the second helper, working from the tapping side, digs out the clay-loam plug and most of the dolomite with which the tapping hole had been closed before the furnace was charged. The hole either is burned out with oxygen from the tapping side or, according to a recently developed method, opened by a jet tapper which consists of a hollow explosive charge that concentrates its energy when discharged against the obstruction in the taphole, quickly opening the hole. Sometimes, opening of the hole must be completed by driving out dolomite remaining in it by the use of a long steel tapping rod inserted through the wicket hole of the middle door on the opposite or charging side of the furnace (Figure

15—29). When the hole is completely opened, the furnace is emptied of its molten contents, which are directed into the steel ladle through the tapping spout (Figure 15—28). Since the tapping hole is located with its highest level at the lowest part of the hearth and slopes downward to meet the tapping spout, the greater portion of the steel flows out of the furnace before slag appears in the spout and this relatively late appearance of slag permits additions of alloying, recarburizing and deoxidizing materials to the spout and, principally, to the ladle. It is not permissible practice for these materials to come into contact with the slag, since some of the phosphoric oxide in the slag may be reduced and the phosphorus re-enter the steel. Further, a coating of slag on these materials may slow down their melting and solution in the steel and reduce the efficiency of the additions. The vertical axis of the ladle usually is placed with respect to the direction of the tapping

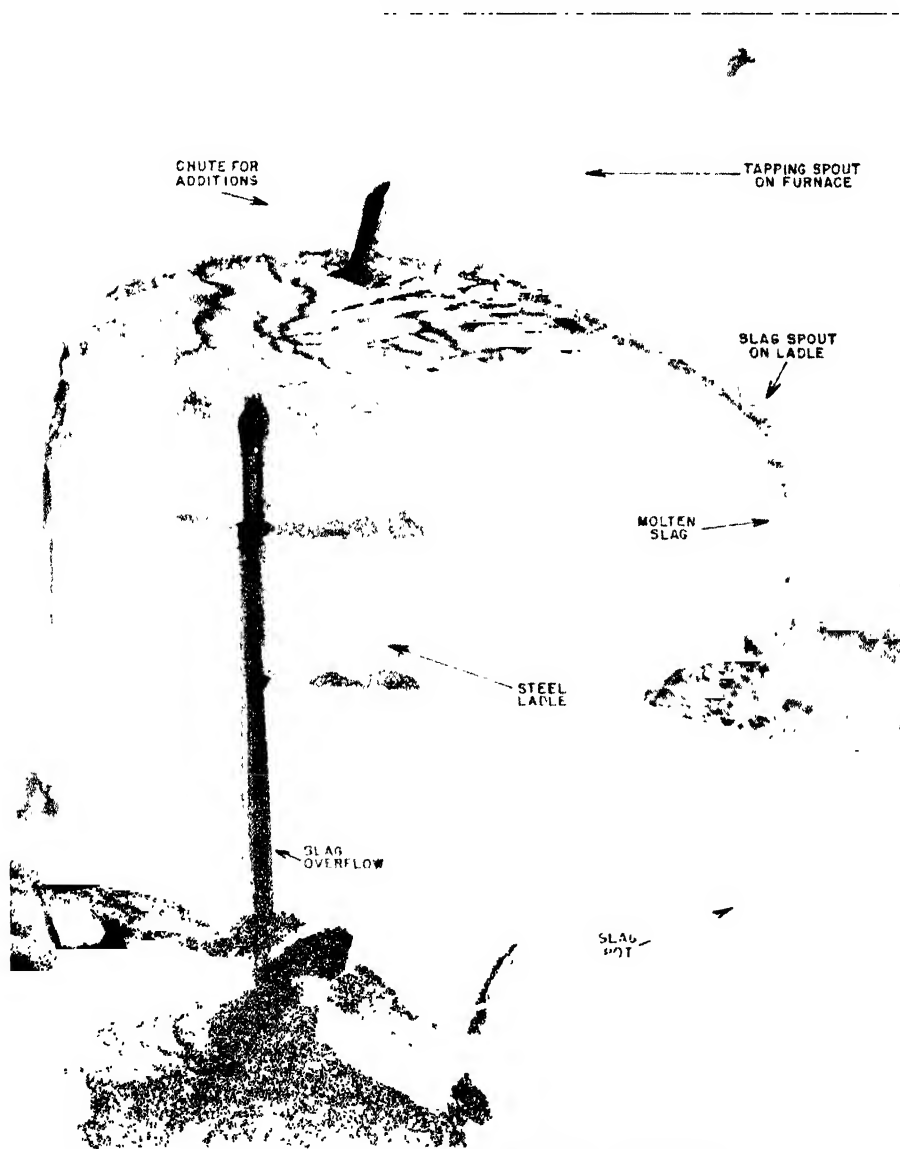


FIG. 15—28. A steel ladle, supported on stands, in position beneath the tapping spout of an open-hearth furnace, near the end of a tap

FIG. 15—29. Workmen, manipulating a long steel rod through the wicket hole of the center door of an open-hearth furnace, drive out the solid material remaining in the partially cleared tap-hole at the rear of the furnace to "tap the heat."



spout so as to give a swirling motion to the metal in the ladle which tends to mix and make it more homogeneous. As soon as the liquid stream from the furnace no longer contains steel, and a depth of slag regulated by good practice remains as a covering for the steel in the ladle (excess slag having flowed from the ladle through slag spouts into a slag pot), the tapping spout is removed by a spout hoist and the ladle lifted from its supporting ladle stands by one of the pouring cranes and conveyed to a position over the ingot molds at the pouring platform.

Ladles—A vertical section through a typical ladle used in basic open-hearth steelmaking is shown in Figure 15—30. The ladle shell itself is fabricated from steel plate by riveting or welding. The horizontal section usually is circular, but a greater capacity often is obtained for a given distance between the trunnions by use of an oval section. The diameter of the ladle increases toward the top to facilitate removal of the shell of metal that occasionally solidifies on the sides and bottom of the interior of the ladle and which commonly is called a *skull*.

The refractory lining of a ladle varies to some extent with the size of the ladle, the thinnest being $3\frac{1}{2}$ inches thick. Thicker linings than this are the rule, and lower portions of the side wall usually are installed with an

extra thickness of brick in that location to compensate for the additional time that this part of the lining is in contact with the molten metal. Extra thickness also may be provided in that portion of the side wall opposite the furnace spout which receives the impact of the stream of metal flowing from the furnace.

The type of brick in most common use is fireclay brick which softens at a temperature as low as 2350°F ; the brick bloats and swells to one and one-quarter times its original size. In service, only a fraction of an inch of the brick beneath the surface exposed to the molten metal actually reaches a temperature above 2300°F , but this is an important consideration since it is this portion which expands (bloats) enough to seal the joints between the individual brick of the lining. This characteristic of ladle brick provides a smooth, tight, almost monolithic working surface that increases ladle life by preventing the penetration of metal into the joints with subsequent premature loss of the lining during removal of a skull. Therefore, though it may have a somewhat lower resistance to erosion by molten steel and slag than a more highly refractory material, the fireclay type of brick usually is the most acceptable for this service. Ladle linings may be installed with a fireclay mortar but, because of the effectiveness of the bloating action in sealing joints, such linings frequently are installed without mortar except as a means for leveling up courses. All ladle linings must be dried carefully by gas, oil, or solid fuel fires prior to use.

The working lining (that next the molten steel) lasts from 10 to 25 heats, and the life varies with the depth, basicity and fluidity of the slag, the composition and temperature of the steel, and the length of time both steel and slag remain in the ladle.

Practically all basic open-hearth steel is cast from the bottom-pour type of ladle shown in the illustration. With this method of pouring, the stopper assembly acts as a valve to control the flow of metal through the nozzle at the bottom of the ladle (Figure 15—31).

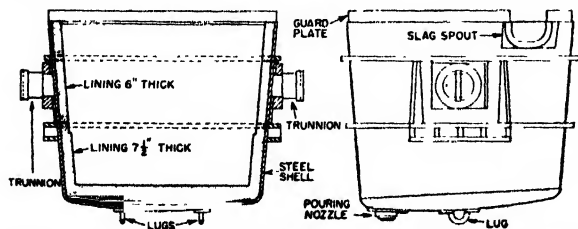


FIG. 15—30. (Left) Vertical section of a steel ladle through the trunnions, showing increased thickness of lining in its lower portions. (Right) Side view of same ladle.

Ladle Nozzles—The external size of the nozzle is controlled largely by the capacity of the ladle and the thickness of the lining, but the length must be sufficient to produce a smooth, solid stream of metal through the central orifice, as free from turbulence and spraying as possible. The diameter of the orifice is determined by considerations such as the grade of steel or size of ingot, and may vary from $1\frac{1}{4}$ to 3 inches in diameter. A typical nozzle for a 100-ton capacity ladle would be approximately 6 inches in outside diameter and 12 to 15 inches long, with an orifice $1\frac{1}{4}$ inches in diameter. General purpose nozzles most frequently are made from low heat-duty fireclay and possess properties very similar to those of the ladle brick previously described. They normally are very dense to reduce orifice erosion and, because of their low refractoriness, the surface of the nozzle seat softens to a depth of about $\frac{1}{4}$ inch when in contact with molten steel. This condition provides a soft, yielding seat for the harder stopper head. A new nozzle usually is inserted into the **nozzle retaining plate** after each heat. Above the nozzle seat, a shallow well usually is formed with plastic clay or loam for the purpose of guiding the stopper head into the nozzle seat. A

pocket block or well block made of low or intermediate-duty fireclay burned to brick sometimes is used to form the well. Both nozzle and well must be dried thoroughly prior to use.

Stopper Rod Assembly—A typical stopper-rod assembly, as shown in Figure 15—31, consists of the steel stopper rod, refractory stopper rod sleeves, refractory stopper head, and steel stopper head pin by which the stopper head is keyed to the stopper rod. While other methods for securing the stopper head to the rod are used, the pin and key arrangement is employed by the majority of operators.

The stopper head and nozzle constitute a valve and best results are obtained when one member is harder than the other at steel-pouring temperatures. Since the surface of the nozzle seat is normally quite soft under steel-pouring conditions, the industry now almost universally employs the relatively hard clay-graphite stopper head composed of 15 to 25 per cent graphite blended with various highly refractory clays.

Prior to tapping the heat into the ladle, the stopper is keyed rigidly into the nozzle seat and the entire nozzle and head area are preheated to insure dryness.

The sleeves used to protect the steel stopper rod are manufactured in 9 and 18-inch lengths from fireclay which, when burned, has a similar or somewhat greater refractoriness than ladle brick. Their essential properties are resistance to thermal shock, resistance to slag and metal attack, low thermal conductivity, and snugly fitting male and female parts. A plastic fireclay or air-setting refractory mortar is placed between the joints and the entire series of sleeves is drawn together by the nut on top of the rod. The metal flange at the lower end of the stopper rod takes the thrust of the sleeves during the tightening operation, preventing breakage of the head. After the completed stopper rod has been dried thoroughly and installed in the ladle, the sleeve nut is backed off about $\frac{1}{8}$ inch per foot of stopper rod to provide for the thermal expansion of the sleeves.

As shown in Figure 15—31, the ladle rigging required for raising and lowering the stopper rod consists essentially of a barrel within which a square or cylindrical slide is operated by an appropriate lever arrangement placed at the lower end. The stopper rod is bolted, at its upper end, to a rigid stopper carrier, which in turn is keyed or bolted securely to the upper end of the slide. The entire rigging is attached to the side of the ladle shell in a manner that permits a certain amount of lateral movement of the stopper with respect to the nozzle and the well, through proper adjustment of the adjusting wheel and screw placed within reach of the steel pourer. This adjustment can be made while setting the stopper or during the pouring of the heat.

Proper manipulation of the pouring assembly to start or stop the flow of molten steel to the mold requires a thorough knowledge of the manner in which pouring affects the surface quality of steel ingots as well as an understanding of the refractories and equipment.

Pouring—The pouring operation (Figure 15—10) is critical, first in controlling molten steel so that it does not endanger personnel or damage equipment, second because poor practice in the assembly, adjustment or manipulation of pouring equipment may seriously and adversely affect the surface quality of the product.

One of the most important factors affecting the surface of the ingot and subsequent product is the rate of flow of the pouring stream and the resultant rate of rise of molten steel in the mold. This rate of rise is determined primarily by nozzle size, mold size, temperature and fluidity of the steel and height of metal in the ladle.

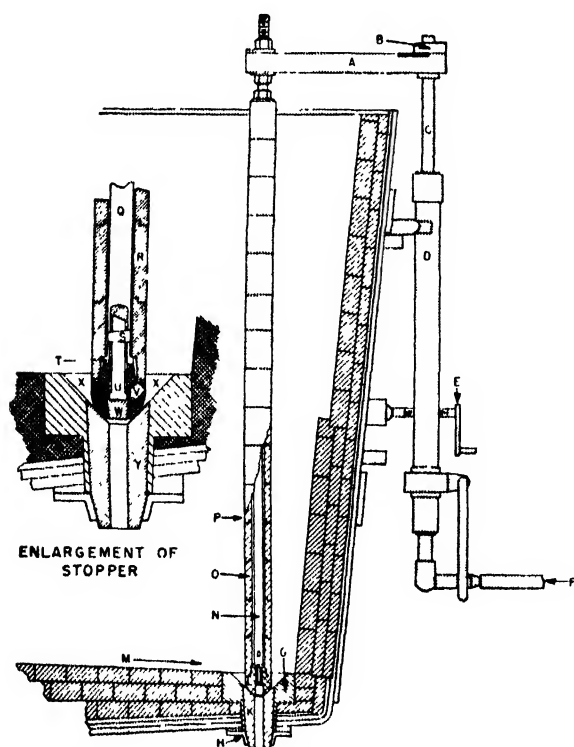


FIG. 15—31. Stopper rod, well, and nozzle assembly for a modern steel ladle, indicating materials of construction and method of installation.

LEGEND

- | | |
|--------------------|------------------------------|
| A — Gooseneck | O — Sleeve |
| B — Key | P — Stopper |
| C — Slide | Q — Low-carbon steel rod |
| D — Barrel | R — Sleeve (fireclay brick) |
| E — Adjustment | S — Steel key |
| F — Control lever | T — Flange on rod end |
| G — Rammed well | U — Low-carbon steel bolt |
| H — Nozzle plate | V — Head (graphite and clay) |
| K — Nozzle | W — Undercut fill |
| L — Head | X — Ladle well profile |
| M — Slope to drain | Y — Fireclay nozzle |
| N — Rod | |

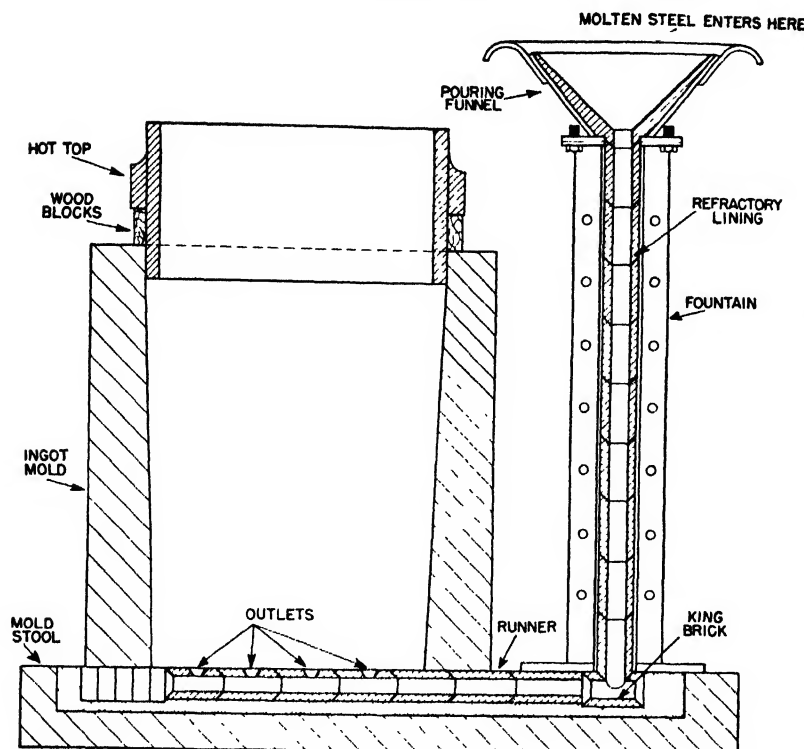


FIG. 15-32. Schematic arrangement and nomenclature of parts of a mold assembly for bottom pouring.

The optimum pouring rate is determined primarily by the grade of steel, but is also dependent upon such factors as mold design, type of mold coating, and temperature of steel.

Caution should be exercised to prevent dirt or refractory material from falling into the molds and becoming entrapped in the ingot.

Tun Dish or Basket Pouring—As an alternative to casting steel directly from the ladle to the molds, an intermediate pouring vessel may be used. Such vessels are termed **pouring baskets**, **pouring boxes**, or **tun dishes**. They may be suspended from the main ladle, attached to the pouring platform or carried on an auxiliary carriage, or supported directly by the mold. The method usually involves pouring from the tapping ladle through a large nozzle into the pouring vessel. Steel may be poured into the molds from one or two smaller nozzles in the pouring vessel, one nozzle to a mold.

Use of this practice is more costly than conventional pouring, but offers the following advantages: (1) greater opportunity for non-metallic inclusions to become separated from the metal, (2) less splashing in the mold, (3) the possibility of better control of pouring rates, and (4) a reduction in pouring temperature.

Bottom Pouring—Some of the disadvantages of top pouring may be eliminated through the use of a bottom-pour assembly in which the steel is cast into a funnel lined with refractories, passes down through hollow refractory runner brick, and finally emerges to enter the bottom of various molds through outlet brick (Figure 15-32). The assembly of funnel and vertical runner which receives the molten steel from the ladle is sometimes called a **fountain**. The metal rises steadily in the molds with very little agitation since the main falling force of the stream from the nozzle is absorbed in the runner. One or more molds may be cast from the same fountain, so that the rate of rise can be varied also by

the size and number of molds used. The cost of this operation involves more expense than for top pouring. However, it usually provides better surface quality for certain grades of steel. On the other hand, erosion of fountain and runner brick provides an additional source of non-metallic inclusions which may be a disadvantage in some steels.

Sampling the Steel—During the pouring of the heat, samples of the molten-steel stream from the nozzle (with the flow momentarily slackened) are taken in a steel spoon, containing a required small amount of aluminum wire to "kill" the sample. From the spoon, steel is poured into suitable diminutive cast-iron molds, depending on the method of analysis, to form test specimens for the chemical laboratory where they are analyzed by various methods for the elements which are significant. These are called the **ladle analyses** and are supplemented on some grades by additional **check analyses** on samples taken from blooms, billets or the finished products.

After the pouring of the heat of steel into molds is complete, the slag remaining in the ladle is dumped into slag ladles or pots which are subsequently conveyed to the slag yard on special cars. The mold cars with the filled molds thereon are held at the pouring platform for various intervals of time, depending on the grade of steel made and are then transferred to the stripper cranes where the molds are removed from the ingots or the ingots withdrawn from the molds. The removal of mold from ingot is called "**stripping**" (Figure 15-11).

Ingot Molds are of many types, shapes and sizes. In shape they vary from round, square or oblong molds to those occasional ones that roughly take the form of the section into which the steel is to be rolled. An example of the latter is the mold used to cast an ingot which is roughly of a beam shape which is finally finished as a large, wide-flanged beam. Generally, ingots to be rolled

are of square or oblong cross section, corrugated or fluted in some cases to minimize the cracking of the ingot surface as it solidifies and continues to cool. Ingots to be rolled into blooms or billets are usually of square cross section and those to be rolled into slabs and then into flat-rolled product are usually of oblong cross section. Ingots for forging are generally of a round or many sided cross section, corrugated or fluted.

Ingot cross sections and weights must be established according to the final cross section in rolling and forging, and the weight per piece of final product, with due consideration to the rolling or forging equipment of the individual plants which process the material to the final product.

Molds usually are tapered from one end to the other so that the top of the ingot cast in them is either smaller or larger than the bottom, according to whether the mold is of the big-end-down or big-end-up type (see also Chapter 20). Taper is necessary in both types to facilitate the stripping of the ingot from the mold. In the case of the big-end-up molds, taper also helps to emphasize the freezing from the sides and bottom in such a manner that the last part of the ingot remaining liquid is directly under the hot top, thus promoting soundness. The hot top itself may contain 10 to 15 per cent of the total ingot weight.

Originally hot tops were refractory hollow-tile shapes, or metal casings lined with refractory material,

placed upon the tops of the mold which usually were good for one pouring. At present, the larger proportion of hot tops are refractory-lined steel or iron castings with suitable coatings applied between pourings. The linings may last for twenty-five heats or more. Practically all hot tops, whether of the lined-metal-casing or hollow-tile type, are now inserted into the top of the mold and are supported at various heights depending on the length and required weight of the main body of the ingot. Chapter 20 describes and illustrates the effect of a hot top.

Mold wall thickness ranges from about 4 inches for a 20-inch by 20-inch ingot to about 6 inches for a 32-inch ingot. For molds used to cast large ingots for slabbing mills, the wall thickness may be made somewhat greater for optimum mold life. The life of ingot molds has been found to be a function of the ratio of mold weight to ingot weight.

Ingot Height—There are limits to ingot height, determined by such factors as desirable ingot structure, desirable rimming action in pouring, and design of the pit furnaces in which ingots are heated.

Usually ingot molds are made of blast furnace or cupola iron, a relatively small proportion of cast steel. Big-end-down, big-end-up, open-top and bottle-top, open-bottom, plug bottom, and solid-bottom molds, along with mold coatings, are discussed at some length in Chapter 20.

SECTION 6

KEEPING THE BASIC OPEN-HEARTH FURNACE IN REPAIR

Preparation of the Furnace for the Next Charge—After the runner or tapping spout is lifted and thus detached from the furnace, the slag and any steel that remains in the furnace flow out of the tapping hole into the cinder pit. The second helper must keep the tapping hole open until everything that can flow from the furnace has done so. Fluorspar usually is thrown in on the slag left so that it becomes sufficiently fluid to flow out and its building up on the bottom of the furnace prevented. Often the effective draining of the bottom is prevented by a ridge of lime adhering to the bottom, forming a dam which holds back the slag and steel. Usually at the start, lime ridges hold back only a small portion of the steel. This may seem insignificant, but if allowed to remain, these small pools will develop into larger pools of oxidized steel. Eventually this will result in a bottom boil that will allow the intrusion of steel below the surface by bringing up the basic material forming the bottom of the pools. Liquid slag and liquid steel held back by lime ridges or remaining in the bottom depressions can be disposed of most effectively by blowing with either compressed air or steam. This is done by reducing the fuel input to the furnace and inserting a long steel pipe through the wicket hole of the furnace door and applying air or steam under pressure. After all the steel and slag are removed, the smaller holes may be filled with prepared dolomite; however, the practice of repairing larger ones may differ in that magnesite may be used instead of dolomite, and the work of repairing the bottom may progress by sintering layer on layer of the refractory material in the holes.

To reduce the time required for bottom repairs, prepared mixes high in magnesia content are now frequently used. These require a minimum of time for sintering since the composition is regulated to make them "quick-setting" when properly mixed with specified quantities of water at time of use.

Another method of draining holes is to rabble the metal toward the taphole by means of rabble plates attached to bars. When dry (drained of slag and steel), the bottom is repaired in the same manner as when using air or steam for draining.

The described methods are most advisable from the standpoint of bottom upkeep and economy of time.

There are times, however, when the depth of the hole or its location will necessitate preliminary draining. This may be done with an open tapping hole or by closing it after the furnace has been emptied. Next, fluorspar, scale, or ferrosilicon are placed on the ridges and the high spots of the bottom. The furnace is then operated with a hot flame which tends to soften and disintegrate the high spots in the bottom, allowing the metal in pools to run out of the furnace when the tapping hole is open.

The practice of using erosive material should be resorted to only when absolutely necessary, as it tends to erode the good portion of the bottom as well. Extreme care should be exercised in draining off all eroded material prior to building up the bottom. While the eroded portion of the bottom is being repaired with the fuel on, the entire hearth is examined and simultaneously reconditioned. The fuel is shut off as soon as repairs to the bottom have been completed.

Proceeding to the next step, the second helper and third helper remove the steel that has chilled in the taphole, rake out and free the hole of slag and steel, and close it up with dolomite. A plug of clay is used to seal up the outside of the hole and hold the dolomite in place. The banks are repaired by fettling either manually or with a dolomite machine and the furnace is ready for charging again. Use of the dolomite machine, in which a hopper-fed belt throws dolomite to the required position within the furnace chamber, has become almost standard practice, especially in maintaining the hearths and banks of the larger furnaces. Op-

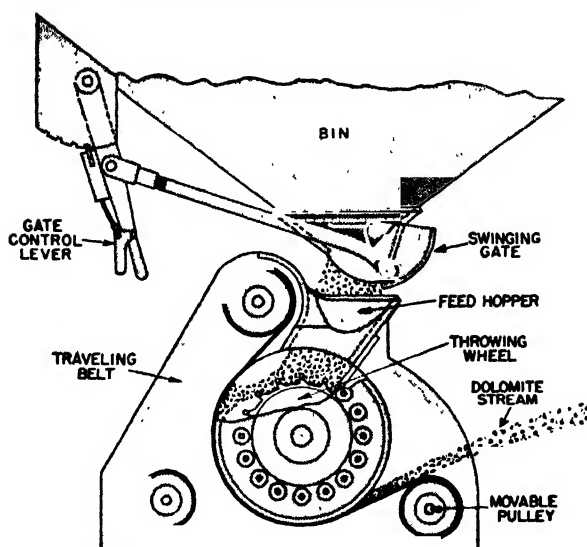


FIG. 15-33. Schematic diagram of the main working parts and principle of operation of a dolomite-throwing machine. Raising or lowering of the movable pulley (lower right) changes the trajectory of the stream.

eration of one of these machines is illustrated in Figure 15-33.

Generally normal bottom maintenance should be adequate to preserve bottom contour so that all of the steel and slag flows out of the taphole and assures a reasonable amount of time for repairing the bottom. However, during the years of bottom life it is possible that, either due to the method of bottom maintenance or to the prevailing steelmaking practices, the nature and extent of bottom erosion may require general resurfacing. The bottom is then given a thorough cleaning as described above and built up again with 5 to 8 inches of either magnesite or double burned dolomite applied as for a new bottom.

The thick layer of newly sintered bottom effectively reduces bottom delays for approximately 300 to 400 subsequent heats.

Furnace Troubles—In the operation of a furnace, troubles of a very serious nature may occur at any time, unless the furnace is watched closely and carefully handled. Thus, the taphole may break out prematurely if it is not properly tamped and capped, or it may become hopelessly clogged with frozen steel if it is not properly cleaned after each heat. Sometimes, sections of the bottom become detached, and these will rise at once due to the buoyant force of the metal; when this occurs the heat must be tapped at once, and no more heats may be charged until the damaged bottom is repaired. The ports in furnaces fired with producer gas require constant attention to prevent them from building up or melting down, thus changing the angle of the flame and spoiling the optimum heat transfer. Leaks may occur in the walls of the up-takes or down-takes of such producer gas furnaces, so that some gas is burnt before it reaches the hearth. The walls and roof often wear out long before the rest of the furnace needs repairing. Roofs usually last for about 200 heats. The roof can be repaired in a few hours, and a cave-in of the roof is of a serious nature only when it falls when a heat is nearly finished. The most disastrous mishap that can occur to a furnace is a breakout. Breakouts may be caused by several things. A hole near a bank may not have been noticed or may have

been insufficiently repaired, in which case the steel works down into it and gradually makes it deeper until, finally, the metal finds its way through the wall and out of the furnace. Sometimes, owing to a thin spot on the banks or to slag having worked down through the banks from above, the slag gradually cuts its way out through the walls, in which case it is usually followed by steel. Breakouts are always serious matters. Once a breakout occurs, the tapping hole should be opened immediately, and as much of the steel as possible tapped into the ladle or cinder pit. The spread of cinder and metal upon the floor where a breakout has occurred can be contained usually by throwing dolomite around it.

After several hundred heats, the checkerwork has become so badly clogged and the furnace brickwork is so eaten away that it becomes necessary to close down the furnace for general repairs, during which the greater part of the brickwork may be torn out and replaced.

Repair Materials—It is evident that, in making up the bottom and for doing the repair work about a furnace, much depends upon the materials employed. Great care always must be exercised to see that they are of the right chemical composition, and physically suited for the work in hand, as otherwise the best of workmanship in making the repairs will go for naught. The few remarks here are supplemented in the chapter on refractories.

Dolomite is found in deposits similar to those of limestone. Like the latter it varies in composition through quite wide ranges, but generally it may be stated that to be suitable for open-hearth work, it should not have over 1 per cent SiO_2 . Three types of dolomite are used for repairing the furnace:

- (1) Raw dolomite for normal fettling and banking the doors.
- (2) Single burned dolomite prepared locally by burning dolomite in shaft kilns at a temperature of 2200 to 2300°F, for normal fettling and minor bottom repairs.
- (3) Synthetically prepared clinker dolomite in which

lime and magnesia are transformed into a crystalline state by burning at a high temperature in a rotary kiln at which time some iron oxide is introduced as a flux. Generally this material is used to supplement raw dolomite and single burned dolomite, and for bottom repairs of a more serious nature.

Magnesite—Prior to the first World War, practically the entire supply of magnesite was imported from Austria. However, since that time, the demands of the American steel industry have been supplied predominantly by domestic sources, principal of which is Chewallah magnesite in the State of Washington. There are several brands of magnesite available, both of foreign and domestic origin. Magnesites from sea-water magnesia have been introduced in competition with natural magnesite.

Chrome Ore—Chrome ore has always been imported, as the limited deposits so far discovered in the United States and Canada are of an inferior grade for open-hearth use. It is received in the form of small lumps. It

is ground and mixed, in a wet pan, often with some magnesite or suitable bonding material, and is used in repair work where a chemically neutral substance (neither acid nor basic in characteristics) is required, such as in patching flues, tapping holes, ports, and so on.

The composition of an average sample of a satisfactory grade of this ore is shown below.

Ingredient	Per Cent
SiO ₂	4-9
FeO	10-14
MnO	Less than 1
Al ₂ O ₃	10-30
CaO	Less than 2
Cr ₂ O ₃	32-42
Loss on Ignition	1-3

Besides these materials, a quartzite containing 98 to 99 per cent silica is employed at some of the works, while all plants will use large quantities of loam and of fire clay for lining furnace spouts and ladles, for making up stopper-sleeve joints, and for other repair work of minor importance.

SECTION 7

FACTORS AFFECTING ECONOMY OF FURNACE OPERATION

Fuel Consumption—Beside the fuel required for carrying on the metallurgical process of steelmaking from the beginning of charging to tapping the finished heat, it is necessary to burn fuel for heating the furnace to a temperature which permits the operators to begin the "burning-in" of the hearth, and for maintaining the proper temperature during the installation of the furnace hearth. In the periods between tapping of one heat and charging of the next, fuel is required for drying and preheating ladles, drying ladle-stopper rods, maintaining metal mixers at the proper temperature level, heating tapping spouts, drying hot tops, preheating ferroalloys, and sometimes thawing materials. Fuels used for other than actual furnace requirements may be of lower calorific value and consequently cheaper than those needed in the open hearth.

The approach to an economical use of fuel in quantities per ton of steel produced is subject to a number of conditions, such as the kind of fuel used, continuity and rate of production, the design of the furnace (in which hearth area per ton of heat tapped and bath depth are important factors), the proper maintenance of the furnace, its walls, passages, the insulation and sealing against air leakage (on the parts of the furnace system where insulation and sealing are necessary), the degree of control of melting chamber pressure (so as to prevent infiltration of relatively cold air which lowers flame and bath temperature), the proportioning of air supply to fuel burning rate, the control of flame shape, its direction and coverage of the charge and bath, and the promotion of high radiating effects from flame to solid charge and to slag and molten bath, the proportion of molten pig iron used, the elimination of delays in introducing the charge into the furnace, the grade and type of steel to be made, and the care and intelligence with which the furnace is operated. Included in the last item are the very important matters of regulating fuel consumption rate in proportion to metallurgical requirements and of timing the reversals of the regenerator system and the fuel flow so that the transfer of heat from waste gases to preheat the air for combustion is made at the maximum level of efficiency. This timing is determined and carried out in modern practice by the assistance of automatic devices

which are actuated by pyrometers installed in the regenerator system. Modern control and indicating equipment give the furnace operator a good visual indication of fuel rate and help to select a rate of air supply for combustion according to predetermined proportion of air weight to fuel weight.

A list of the controls and instruments related to fuel burning, with which a large number of modern open-hearth furnaces are equipped, follows:

- (1) Control and recording gage for atomizing-steam pressure.
- (2) Control and recording flowmeter for coke-oven or natural gas.
- (3) Control and recording flowmeter for liquid fuel.
- (4) Automatic furnace-pressure control and recording pressure gage.
- (5) Recording draft gage.
- (6) Recording pyrometers for checker temperatures, waste-heat boiler, and stack temperatures.
- (7) Fuel-air ratio indicator.
- (8) Forced-air control, including predetermined automatic fuel-air ratio control.
- (9) Roof-temperature indicator, with automatic roof-temperature control of part of the fuel rate.

The calorific value of fuel usually is expressed by its gross heating value in British thermal units per unit of volume, weight or liquid measure. These concepts were discussed in Chapter 3 on fuels.

Fuel consumption in terms of total calorific value of the fuel used per ton of steel produced usually is expressed in total net heating value of the fuel (millions of Btu) per ton of ingots produced. In the steel industry, the range of fuel consumption by open-hearth furnaces is from 3,000,000 to 4,000,000 Btu per net ton for furnaces in which molten pig iron is used to 4,200,000 to 5,300,000 Btu when the charge is all cold. The heat balance of a modern basic open-hearth furnace is shown schematically in Figure 15-34.

Rate of Steel Production—Practically all of the conditions given above for an economical use of fuel are approximately though not necessarily precisely those which provide maximum rate of steel production for a given quality requirement in the product. Thus it may be

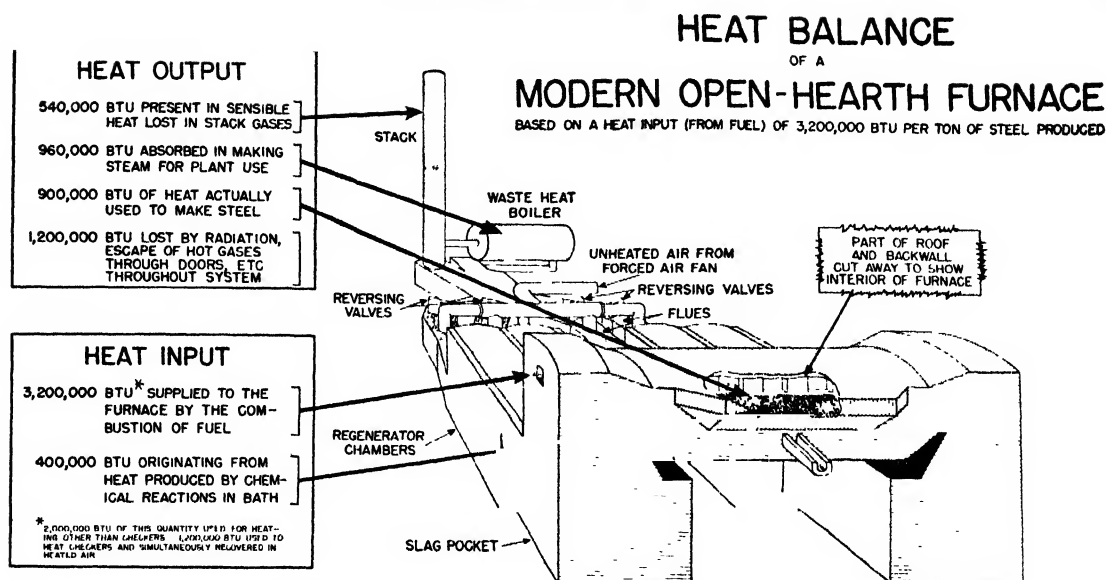


Fig. 15—34. Heat balance of a modern basic open-hearth furnace.

possible to operate a furnace on the low fuel rate of 3,000,000 Btu per net ton of ingots, but it may be necessary to increase this fuel rate by at least ten per cent above this in order to increase the tonnage rate to that desired. The conditions which improve production rate (in addition to those which were discussed in the section dealing with fuel consumption) include the following: relatively large hearth area per ton and relatively small bath depth (these can be considered as matters related to furnace design), size of heat in tons, ability to burn fuel at a high rate in melting down the solid part of the charge, iron ores for the charge which are relatively superior in physical and chemical properties (for example, more massive, low in fines, low in silica and water of crystallization), silicon in the molten and cold pig iron held within a specified maximum, bundling and pressing of light scrap to promote rapid charging, rapid preliminary analyses of bath samples, careful planning and carrying out of the process procedure for each grade and quality of product.

The use of oxygen or compressed air with the fuel stream and of oxygen in eliminating carbon from the bath increase the rate of production. Oxygen or compressed air are of particular value as combustion aids during the melting down of the solid part of the metallic charge (scrap, cold pig iron) and will show the greatest effect in speeding up production rate in furnaces as the ratio of the cold to molten parts of the charge increases, the effect being much less when large proportions of hot metal are used. The oxygen usually is introduced through the regular liquid-fuel burner, with the oxygen jet entering just below the fuel jet.

Twenty-four tons per 1000 square feet of hearth area per operating hour where the area of the bath is taken

on the brick work at the door sill level before rammed or burned-in sections of the furnace hearth are installed (see Section 3) may be considered a satisfactory operating rate for a larger furnace (200 tons and up). The effect of hearth area of open-hearth furnaces on production rate in tons per operating hour for various furnace capacities (rated in size of heat tapped) have been reviewed recently for the American steel industry, and the relationships are given in Table 15—III.

Table 15—III. Effect of Hearth Area on Open-Hearth Production Rate

Hearth Area Per Ton of Capacity	Output in Net Tons per Operating Hour		
	Furnace Capacity		
	200 Tons	150 Tons	100 Tons
3.50 to 4.00	18.4	13.7	9.2
4.00 to 4.50	19.2	14.4	9.6
4.75 and over	20.2	15.2	10.1

When waste heat boilers are used, the exhaust fans in the system assist in maintaining high production rates by maintaining uniform drafts to give the desired removal of waste gaseous products.

The kind of refractories used is important in preventing relatively early failure of the furnace hearth chamber and port ends, in maintaining the contour of the inner surfaces of the areas of the above exposed to high temperature and hot gases and, consequently, an important factor in promoting continuity of operation.

SECTION 8

THE CHEMISTRY OF BASIC OPEN-HEARTH MELTING

In melting down, the item of greatest interest is the action of flame and furnace atmosphere upon the charge. The charge at the beginning consists generally of steel

scrap, but may consist of both cold pig iron and scrap; the pig iron with a low melting range around 2100°F, and the scrap with a melting range of 2500 to 2700°F.

Table 15—IV. Compositions of First Slag Formed in Basic Open-Hearth Heats

Constituent	Slag from Furnace A (Per Cent)	Slag from Furnace B (Per Cent)
SiO ₂	8.54	1.00
FeO*	61.05	78.24
Fe ₂ O ₃ **	11.10	15.31
MnO	2.31	0.81
P ₂ O ₅	0.26	0.14
Al ₂ O ₃	1.98	2.70
CaO	9.13	2.70
MgO	5.48	1.12
S***	0.16	0.25

* As calculated from analysis for ferrous iron.

** As calculated from analysis for ferric iron.

*** Nature of sulphur-bearing slag constituents unknown.

At the start, the temperature is low due to the large heat absorption from the flame by the relatively cold charge. If the fuel contains sulphur, the metal may absorb some of this element as the temperature rises, although the amount may be small if conditions are maintained to give rapid and complete combustion of the fuel with a proper excess of oxygen as noted elsewhere. During the melting, which begins with the metal at the top of the charge and progresses gradually downward, iron may be oxidized by the gases oxygen, carbon dioxide, or water vapor. For a given furnace atmosphere, the amount of iron oxidized will depend on the "bulk density" of the charge, oxidation being the least in pig iron, and the greatest in the lightest steel scrap because of the much greater surface area exposed by the latter to the gases. The degree of oxidation of the scrap is an important factor as the oxides formed are equivalent in action to any ore which might have been charged to furnish oxygen for the elimination process.

The other elements which may be oxidized during melting either by the atmosphere or by oxides of iron from the oxidation of the scrap are carbon, manganese, phosphorus and silicon, all of which are always present, and chromium, vanadium, aluminum, titanium, tungsten, columbium, zinc, etc., all of which may or may not be present. The oxides of some of these elements may be reduced later, in the working period, depending on the amounts present and the composition of the slag formed. In the case of all the oxidizable impurities (except carbon, where the oxidation product is a gas), there will result eventually a condition approaching equilibrium where they distribute themselves between slag and metal as determined by the natural laws governing heterogeneous systems for the chemical reactions involved. Other elements which may be present, such as copper, nickel, molybdenum, cobalt, tin, arsenic, etc., are held in the iron because the iron would have to be completely oxidized before these elements could be removed by oxidation. Proof of these statements may be had by analysis of the first slags formed, which collect with the molten metal in certain parts of the furnace before all the scrap is melted. Results of analyses of two of these slags are shown in Table 15—IV.

CHEMISTRY OF THE ORE BOIL

The reactions following the melting period depend on the make-up of the charge. With a high scrap charge, no ore may be charged, because oxidation of the scrap during melting can furnish the desired amount of oxygen. With no charged ore, or a smaller charge of ore, the limestone charged also may be reduced by the amount

that would have been required to flux the silica in the ore, and the silica from silicon in extra pig iron. Likewise, the oxidizing effect of the CO₂ from calcination of the stone will not be as necessary because of greater oxidation of a larger amount of scrap. The following description is for the usual scrap and pig iron charge, with the pig iron added in the molten condition after a part of the scrap has been melted and the temperature of the hearth and contents is high enough to prevent solidification of the molten metal. As soon as this hot metal is added, reactions begin between the silicon, manganese, phosphorus, and carbon of the pig iron, first with the oxygen supplied by oxidation of the scrap and later with the oxygen supplied by the charged ore as melting progresses. These reactions are exothermic and help to keep the iron molten, as well as contributing to the general supply of heat to the furnace. In general, these elements are oxidized out of the metal in the order silicon, manganese, phosphorus, and carbon. The reactions representing the oxidation of these elements, may be represented by the following equations:

1. Si (in Fe) + 2O (in Fe) = SiO₂ (solid)
2. Mn (in Fe) + O (in Fe) = MnO (solid)
3. 2P (in Fe) + 5O (in Fe) +
4 CaO (solid) = 4 CaO·P₂O₅ (liquid)
4. 2C (in Fe) + 3O (in Fe) =
CO (gas) + CO₂ (gas)

The above reactions express equilibria; that is, they may proceed from left to right or right to left, depending on the relative activities of the substances involved. Reaction (1) results in the formation of the acidic oxide SiO₂, which is insoluble in iron, and in the liberation of heat. Reaction (2) likewise furnishes heat, but results in the formation of the basic oxide MnO, which is only slightly soluble in iron. It will be noted that reaction (3) includes the slag-forming compound CaO. The reason for this is that the high vapor pressure of P₂O₅ precludes the possibility of its existence in the free state, yet analysis of slags always shows phosphorus present as the phosphate, indicating that P₂O₅ is formed in the oxidation. It is probable, therefore, that the oxide of phosphorus combines with lime to form 4CaO·P₂O₅. This is borne out further by the fact that the reaction will be reversed quickly provided sufficient oxygen is not available or the amount of lime over that required to balance the other acidic oxides is insufficient, allowing the phosphorus to return to the metal. Reaction (4) produces the gases CO and CO₂ in proportions depending on the carbon content of the bath. Usually over 90 per cent of the gas is CO, which burns to CO₂, as it bubbles off the bath. The elimination of carbon, therefore, produces no oxide which requires a flux for its removal from the iron. There should be supplied to the furnace atmosphere sufficient air to oxidize the CO to CO₂, in order that oxidizing conditions will obtain.

In order that the transfer of silicon and manganese from the metal to the slag may be visualized clearly, the reactions above must be supplemented by others involving neutralization as well as oxidation. In the case of phosphorus, this neutralization already has been indicated.

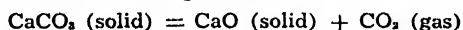
Oxidation and Neutralization of Silicon and Manganese—Since silica is strongly acidic and iron oxide and manganese oxide are basic, all or a large part will be neutralized to form fusible silicates of iron and manganese. These will be a major constituent of the first slag, which is formed before the lime boil. As soon as lime begins to rise from the bottom of the furnace, these silicates will be broken down with the formation of calcium silicates as explained later.

Behavior of Sulphur during Melting and the Ore Boil—Compared with the elimination of the other impurities, the removal of sulphur from the steel in the open hearth is relatively inefficient. As a consequence, an attempt is made to keep the amount of sulphur in all the raw materials at a low value. Fuels other than natural gas contain varying amounts of sulphur, and during the melting period it is not uncommon that the scrap may absorb sulphur from the furnace gases. This may be minimized by maintaining more oxidizing conditions in the furnace so that the sulphur is converted quickly to SO_2 , in which form it is absorbed less readily, and by melting rapidly so that the scrap is exposed for the shortest possible time.

It is probable that sulphur in liquid iron exists as iron and manganese sulphides, and it is possible that it exists in the slag either as calcium sulphide or as calcium sulphite or sulphate. Since it is impossible to analyze slag and metal for individual sulphides, no exact chemical mechanism for sulphur removal has been established. It is usual, consequently, to express the reaction as a simple ratio of "sulphur in slag" to "sulphur in metal." Neither iron sulphide nor manganese sulphide can be oxidized in the open hearth, so that the ratio is not affected by the state of oxidation of the metal, but only by slag composition. Higher ratios are found for the more basic slags, containing free lime. As the slag becomes more basic later in the heat, the ratio becomes more favorable for sulphur removal, but in spite of this fact, rarely more than 50 per cent of the sulphur in the metal can be removed without using excessive slag volumes.

CHEMISTRY OF THE LIME BOIL

Before the refining reactions described above are completed, the limestone will have reached its calcining temperature and begun to give off CO_2 in accordance with the following reaction:



This reaction starts at about 1475°F , but progresses only slowly even at higher temperatures, because it is endothermic and because the limestone is charged in large lumps exposing relatively little surface. The CO_2 formed by the calcination reacts with the iron or one of the metalloids to form oxides, and thereby contributes a considerable proportion of the oxygen required by the process. One end product of this oxidation is CO , which bubbles through the bath and produces a desirable agitation which promotes the absorption of heat from the flame by the bath.

The CaO which is a product of the calcination, floats to the slag and replaces iron and manganese oxides as previously stated, forming calcium silicates and aiding in the removal of phosphorus. As more lime becomes available, the basicity of the slag increases, favoring the removal of sulphur by increasing the sulphur-holding power of the slag.

The slag now represents a liquid mixture of some complexity, consisting of calcium silicates, or double silicates of calcium and other bases, sulphates or sulphides, calcium phosphates or double phosphates, iron oxide, manganese oxide, with oxides of the other oxidizable impurities in the charge, plus such compounds as alumina, magnesia, chromic oxide, etc., gathered from ore, limestone, and refractories. The nature of open-hearth slags is very complex and not completely understood, since it is very difficult to study the slag while in the liquid state.

Preliminary Adjustments in the Slag; Objects of the Run-Off—It is evident from the preceding discussion that a certain excess of calcium oxide over that required

to neutralize the silica is necessary to assure lowering the phosphorus content to the small proportion desired in nearly all basic steel. Also, it has been established that for practical phosphorus elimination, the ratio of lime to silica should be at least 2:1, corresponding to the formula $2\text{CaO} \cdot \text{SiO}_2$. Provided the total silicon content in the charge is known and all the slag formed is held in the furnace until the steel is tapped, this relation makes it easy to calculate the total limestone required.

With a high iron charge and, therefore, a high silicon content in the charge, keeping all the slag in the furnace may make it difficult to control the temperature of the bath because of the thickness of the slag layer. The fuel input needed to drive heat through a thick insulating slag layer may also lead to overheating the furnace roof.

Table 15—V. Typical Composition of Hot Metal, Run-Off Slag, and Finishing Slag for 55% and 70% Hot Metal Practice

Heat No.	55% Hot Metal Practice			
	Composition of Hot Metal (%)	Composition of Slag (%)		
		Run-Off	Finish	
1	C —4.26	SiO_2 26.4	18.4	
	Mn—2.36	CaO 14.1	45.0	
	P —0.271	FeO 22.3	12.2	
	S —0.029	Fe_2O_3 3.0	3.4	
	Si —1.33	MnO 21.6	8.6	
		Al_2O_3 2.9	2.5	
		MgO 6.5	5.9	
		P_2O_5 2.4	3.4	
		S 0.08	
2	C —4.35	SiO_2 22.2	18.1	
	Mn—2.23	CaO 14.6	43.2	
	P —0.273	FeO 29.3	11.6	
	S —0.028	Fe_2O_3 3.6	3.3	
	Si —1.43	MnO 18.4	7.7	
		Al_2O_3 2.3	2.8	
		MgO 4.9	7.8	
		P_2O_5 3.0	2.6	
		S 0.08	0.09	
	70% Hot Metal Practice			
	Composition of Hot Metal (%)	Composition of Slag (%)		
		Run-Off	Finish	
1	C —4.32	SiO_2 19.2	17.7	
	Mn—2.20	CaO 7.5	45.9	
	P —0.255	FeO 42.8	11.1	
	S —0.030	Fe_2O_3 4.6	4.3	
	Si —1.34	MnO 15.8	6.3	
		Al_2O_3 2.5	4.0	
		MgO 3.7	6.2	
		P_2O_5 2.5	2.6	
		S 0.09	0.06	
2	C —4.32	SiO_2 17.8	19.2	
	Mn—2.13	CaO 9.7	43.4	
	P —0.273	FeO 37.9	9.4	
	S —0.023	Fe_2O_3 7.4	2.5	
	Si —1.18	MnO 14.5	7.2	
		Al_2O_3 2.5	3.7	
		MgO 5.2	7.3	
		P_2O_5 2.7	2.9	
		S 0.07	0.06	

Furthermore, the silicon content of the charge may not be known at the time the limestone is charged. These difficulties may be overcome by the run-off. If the furnace has been charged to hold the limestone on the bottom and the run-off is timed to begin shortly after addition of molten pig iron and before the lime boil begins, a large portion of the silica and phosphorus are carried out of the furnace, as shown in Table 15—V. The run-off slag, of course, carries with it the iron and manganese oxides in this early slag which may make necessary further additions of ore to obtain rapid elimination of carbon. If the charging of the furnace, the timing of the addition of hot metal, and the run-off are managed skillfully, the silicon remaining to be neutralized with lime can be reduced to a small and almost constant amount, making possible a relatively small limestone charge. Since lime can be of service only in the slag and the calcination of the limestone absorbs heat, holding the limestone charge to a minimum not only decreases slag volume and cost of flux, but gives increased production with a decrease in fuel consumption.

Finally, attention should be given to the timing of the run-off to see that the temperature and fluidity of the run-off slag are sufficiently high and that the state of agitation of the bath is such that mixing of run-off slag and metal is not occurring extensively during the outward flow of slag. If conditions are not thus maintained, a disproportionate loss of metallic iron from the bath will result from metallic particles carried out with the run-off slag.

CHEMISTRY OF THE WORKING PERIOD

The chief aims during this period are to oxidize the remaining phosphorus out of the metal and neutralize it in the slag, to eliminate the carbon to the percentage desired for the start of the finishing period, and to raise the temperature of the bath to a point suitable for finishing and tapping the steel. To eliminate carbon, oxygen dissolved in the bath is necessary, and to oxidize and neutralize phosphorus both oxygen in the metal and calcium oxide in the slag are required, as indicated by the reactions in the discussion above. The rate of oxidation of the carbon must be kept in step with the rise in temperature, since the melting point of the metal bath rises as the carbon decreases. Although it has been pointed out that the reaction between carbon and oxygen dissolved in the bath is exothermic, it is not sufficiently so to furnish enough heat to supply that required for the decomposition and solution of the ore, so that the net over-all reaction requires the addition of heat. In addition, the oxidizing power and basicity of the slag must be controlled to give the best finishing conditions for the different grades of steel. Since it is through the slag that the refining reactions in the metal are accomplished, this control involves control of the temperature and the physical and chemical properties of the slag. The agents for controlling slag composition and properties are numerous, including ores, lime, limestone, sand, fluorspar, etc., but for economy the skilled melter will depend on iron ore, or other iron oxides, and calcium oxide. Any control system must be based upon chemical composition of the raw materials, and upon tests systematically made during the heat. The system may be based chiefly on slag tests, chiefly on metal tests, or both. Whatever the system, it must meet the requirement that it will give the melter a true picture as to the exact changes the metal is undergoing, both in temperature and composition. Much study has been given to all phases of this subject by many investigators and research workers, but practices still vary to

such an extent that no one example can be selected as truly illustrative of the many practices in this part of the process.

Limitations of the Various Systems of Control—All three items for control; namely, temperature, slag composition, and metal composition, as well as the three systems of control, are limited by the ability of the chemist, physicist, and metallurgist to provide reasonably accurate methods for the determination of the temperature and the various components of the slag and metal that are sufficiently rapid to be of value as instruments of control. The following determinations have been developed to a stage that can be considered satisfactory for this work:

1. **Temperature**—While most melters adhere to the empirical methods of "cutting a rod," visual observation, or pouring metal from a spoon to estimate temperatures, these methods gradually are being supplemented. By radiation pyrometers sighted on the roof of the furnace, the heat input to the bath can be controlled at the maximum allowable by the roof refractories. Similar pyrometers installed so as to sight on the checkers are used for automatic reversal of the burners, and the valves in the flues, thereby giving most efficient operation of the regenerative system. The temperature of the metal bath may be measured by "quick immersion" thermocouples or by photronic or radiation pyrometers sighting on the bottom of a tube inserted into the bath. The readings of these instruments are sufficiently reproducible to provide control. Optical pyrometers are used to measure temperatures on tapping and pouring.

2. **Slag Control**—Rapid chemical methods exist for determining contents of total iron, ferrous iron, ferric iron and manganese. The results usually are expressed as the oxides FeO , Fe_2O_3 , and MnO . Chemical methods for determining lime, silica, phosphorus, etc., are too slow for control purposes. Spectrographic methods now exist which will determine the lime-silica ratio in about 5 minutes. Other control tests are primarily physical. Slag pancakes may be used to estimate both basicity and iron oxide content, and serve well when used by experienced melters. Another control method is measurement of fluidity or viscosity by a flow test. The flowing properties of a slag depend on temperature as well as composition.

3. **Metal Control**—The carbon content of the bath can be determined in 3 to 15 minutes, depending on the composition of the bath. Physical tests for carbon, such as fracture, Carbometer, or Carbanalyzer tests, require 3 to 10 minutes. Chemical tests for carbon require 7 to 15 minutes. Manganese can be determined chemically in 10 to 15 minutes, and spectrographically in less time. Reproducible results for oxygen, sulphur, and phosphorus can be obtained in 20 to 30 minutes, where proper facilities for rapid preparation of the sample are available at the open-hearth floor. Chemical analysis for silicon requires 30 to 40 minutes for accurate results, although this element may be determined spectrographically in 5 minutes with accuracy comparable to chemical methods for control purposes. Chemical or spectrographic analysis is frequently made for the "residual elements," such as copper, nickel, molybdenum, chromium, tin, etc., which enter with the scrap.

The spectrographic methods are considerably more rapid than chemical methods for determining residual elements.

Laws of Chemical Action—The general laws of chemical action which relate to reactions within and between slag and metal are given below. As to their practical application to the control of reactions in the open hearth, much depends on the ability of the individual

who is attempting to apply them to analyze the complex mixtures he has to deal with and then isolate each simple component for study in relation to all conditions affecting a given reaction. However, these laws will serve as guide posts in predicting in advance what will happen when changes are made in the conditions existing in the furnace. These laws, five in number, are named and stated as follows:

1. *Nernst's Law of Distribution*—The ratio in which a dissolved substance distributes itself between two immiscible solvents is a constant at constant temperature.

2. *Raoult's Law*—The addition of a non-volatile solute to a solvent causes at any temperature a fractional lowering of the vapor pressure of the solvent equal to the mol fraction of the solute.

3. *Henry's Law*—The partial pressure of a substance in dilute solution is proportional to the mol fraction of the dissolved substance.

4. *Law of Mass Action*—Whatever the initial concentrations of gaseous or dissolved substances involved in any definite chemical reaction, such as may be expressed by the general equation $aA + bB = cC + dD$, the reaction will always take place so that when equilibrium is reached (at a specific temperature) the conditions are such that they will satisfy the equation

$$\frac{(C)^c \times (D)^d}{(A)^a \times (B)^b} = \text{a constant}$$

where (A), (B), (C), (D) denote the molar concentrations of A, B, C, and D in the equilibrium mixtures and a, b, c, and d represent the number of mols of the species A, B, C, D as expressed by the chemical equation.

5. *van't Hoff's Law*—The effect of temperature on a system in equilibrium will be such that the reaction will occur in such a direction as to oppose the change in temperature. For instance, if the temperature is raised, the reaction will proceed in the direction which will absorb heat.

The question of whether or not the open-hearth

process is an "equilibrium" process has been debated often. According to a strict definition of equilibrium, concerned with all phases of the process, obviously the system is not in a state of equilibrium because the conditions are being changed constantly by such things as additions to slag and metal or variation of temperature. At the temperature involved in steelmaking, it is probable that reactions progress at an extremely rapid rate and consequently follow the changed conditions closely. Therefore, it may be said that with respect to the reacting species, or individual reactions, equilibrium is approached closely; while the whole aim of the operator of the furnace is to shift these equilibria, by the means under his control, towards the conditions desired when the heat is to be tapped. There can be no doubt that the natural laws apply to the steelmaking reactions, so that while the thermodynamic relationships between substances reacting in an open-hearth furnace have not been thoroughly determined, the natural laws can at least be used to make intelligent predictions as to the effect of a condition change.

Composition of Slags—A normal basic open-hearth slag, such as the melter has to deal with at the end of the lime boil, may be considered as a fused mixture of the oxides formed by the oxidation of the manganese and metalloids in the charge, together with the calcium oxide formed by the calcination of the limestone, and iron oxides from the charged ore and from the oxidation of scrap. Other oxides which may be present are magnesia and alumina, derived from the ore charged and from the hearth refractories, and oxides of other oxidizable elements which may have been present in the charge in small amounts; for instance, chromium, vanadium, and titanium. Sulphur is also present in the slag, in a form at present unknown. These many compounds are capable of forming numerous chemical and physical combinations so that the properties of individual components may be no indication of the properties of the mixture as a whole.

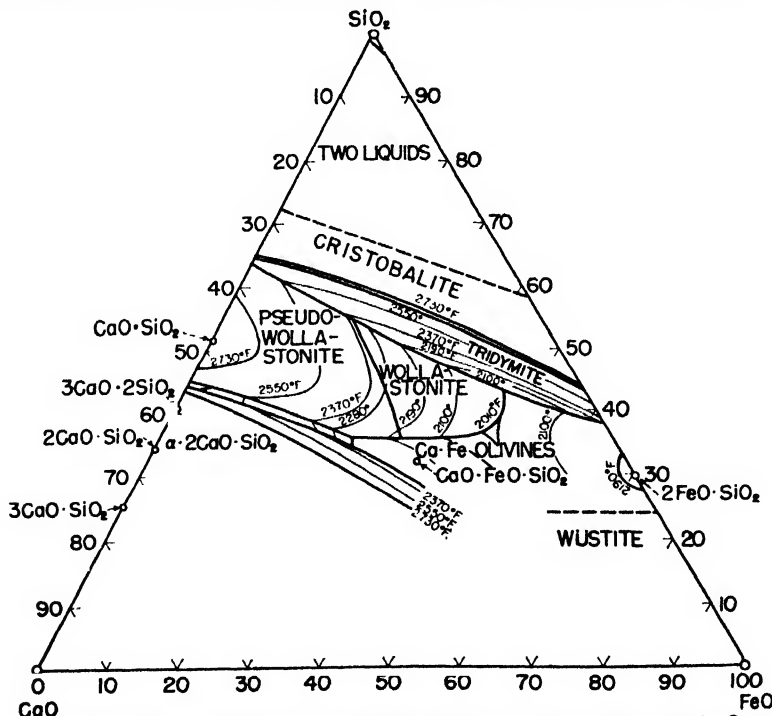


FIG. 15-35. Phase diagram of the system CaO-FeO-SiO_2 . (From "Basic Open Hearth Steelmaking," published by AIME).

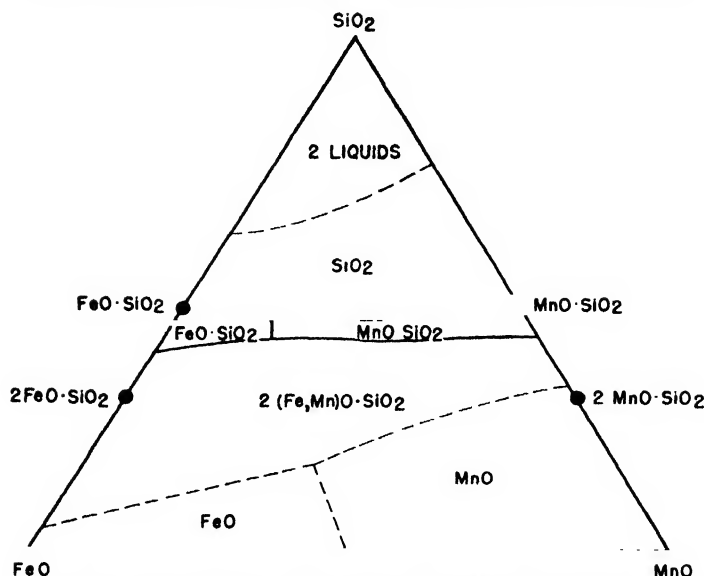


FIG. 15-36. Phase diagram of the system FeO-MnO-SiO₂. (Same source as Fig. 15-35).

A large amount of experimental work has been performed, chiefly by thermal analysis and by the petrographic microscope, to determine the molecular species which may be present in a slag at different stages of a heat. Since both of these methods are subject to limitations, the definite existence of specific species in liquid slag has not been proved, but there is sufficient evidence to lead to the belief that the complex compounds most likely to exist, perhaps partly ionized, at various stages of a heat, include:

2FeO · SiO ₂	} or 2(FeO,MnO) · SiO ₂ ..	{ Iron-manganese silicate
2MnO · SiO ₂		
2CaO · SiO ₂ , or 4CaO · 2SiO ₂ ..		Di-calcium silicate
3CaO · SiO ₂ ..		Tri-calcium silicate
4CaO · P ₂ O ₅ ..		Calcium phosphate
FeO · P ₂ O ₅ ..		Ferrous phosphate
FeS } or (Fe,Mn)S ..		Iron-manganese sulphide
MnS }		
FeO ..		Ferrous oxide
Fe ₂ O ₃ ..		Ferric oxide
CaO (free) ..		Calcium oxide

CaO · Fe₂O₃ Mono-calcium ferrite
2CaO · Fe₂O₃ Di-calcium ferrite

Phase diagrams for numerous slag systems have been developed which are useful in interpreting slag behavior. Examples of two- and three-component diagrams are given in Figures 15-35 to 15-38, inclusive, as well as in Chapter 8 on "Refractories."

Slag Composition in Relation to Refining Reactions—Since the basic open-hearth process is both chemically "basic" and oxidizing in nature, the melter must concern himself with the effects of changes in slag composition on these two factors.

Basicity of a slag may be defined as the ratio of the "basic" components of the slag to the "acid" ones. The most important "base" is CaO, while the most important "acid" is SiO₂; therefore, the simplest basicity ratio is CaO/SiO₂. Numerous refinements of this ratio have been suggested in an effort to arrive at a more exact measure of basicity, but for usual control purposes the simple lime-silica ratio is satisfactory.

Under a pure iron oxide slag the solubility of oxygen

FIG. 15-37. Phase diagram of the system CaO-Fe₂O₃. (Same source as Fig. 15-35.)

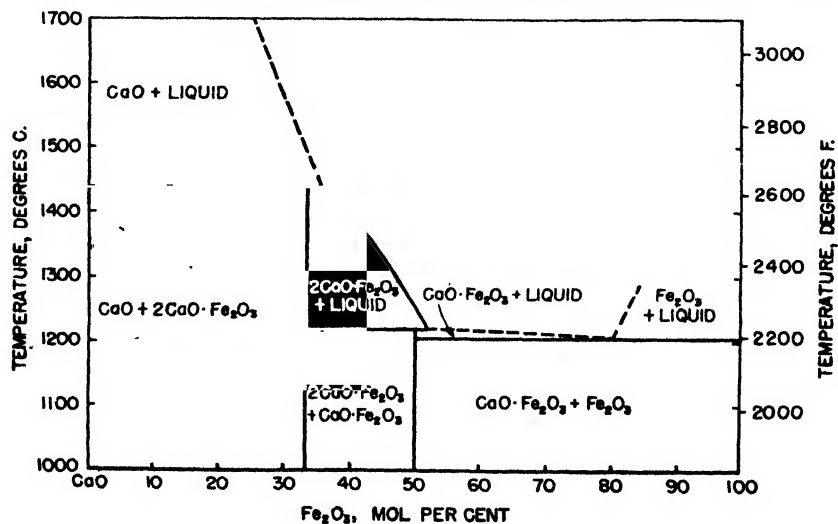
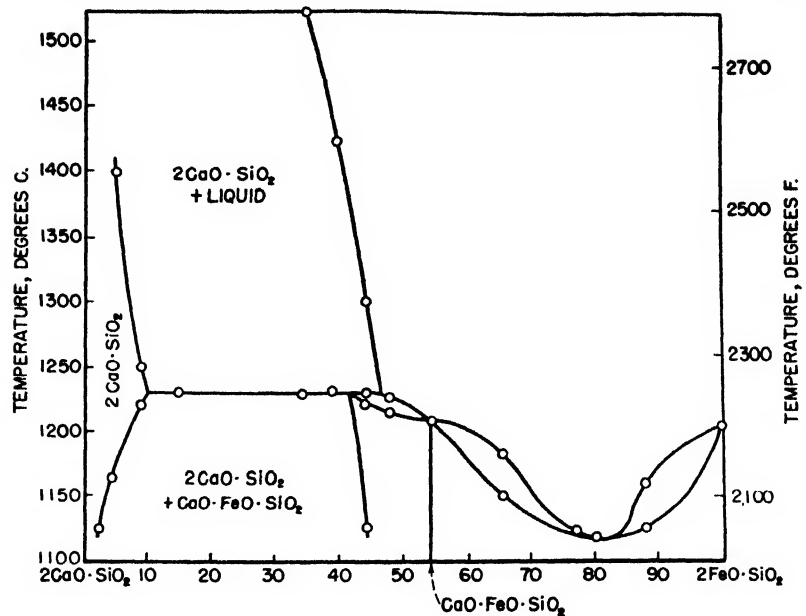


FIG. 15-38. Phase diagram of the system $2\text{CaO} \cdot \text{SiO}_2$ - $2\text{FeO} \cdot \text{SiO}_2$. (Same source as Fig. 15-35.)



in pure iron has been found to be a simple function of temperature, varying from about 0.18 per cent at the melting point of iron to 0.34 per cent at 3092°F (1700°C). In the open-hearth furnace neither pure iron nor a pure iron-oxide slag exists, and the solubility of iron oxide has been found on the average to be related to the simple CaO/SiO_2 ratio as shown in Figure 15-39. This average behavior was obtained from a statistical study of a large number of low-carbon heats and serves to point out the effect of slag composition on the all-important transfer of oxygen from slag to metal. For example, if the lime-silica ratio is 2 to 1, then (from the chart) the ratio of "FeO in slag" to "oxygen in steel" is 200 to 1. Then, if the percentage of FeO in the

slag is 10, the percentage of oxygen in the steel is 0.05. This simple relation does not apply to the slags formed during meltdown, but only to the mature slags formed after most of the lime has entered the slag.

The elimination of phosphorus requires that the slag be both highly oxidized and basic. In the early stages of the heat before the lime boil, a variable portion of the phosphorus may be held in the slag and eliminated in the run-off.

Winkler and Chipman, using synthetic slags, showed that increasing the lime-silica ratio produced dephosphorization up to a value of about 2.75. After this value was reached, only an increase in iron oxide in the slag produced further phosphorus removal. The importance of iron oxide in the slag is observed in the furnace where phosphorus may revert to the steel if the iron oxide content of the slag is reduced to too low a value by virtue of a furnace block or the addition of a deoxidizer.

The distribution of sulphur between slag and metal likewise is affected by slag composition. In general, the removal of sulphur from the metal is favored by a large slag volume, a high basicity, and low iron oxide content. However, the latter two items are usually incompatible in basic open-hearth slags, as high basicity (low silica) is usually associated with high iron oxide (see Figure 15-39).

CHEMISTRY OF THE FINISHING PERIOD

The finishing period of a heat is that period in which final adjustments for temperature are made in the furnace and adjustments of composition are made in furnace, ladle, or molds. Since many grades of steels, ranging from nearly pure iron to complex alloy steels, are made by the basic open-hearth process, it is obvious that the practices used during the finishing period will vary widely. At the beginning of this period, the elimination of manganese and the metalloids, with the exception of carbon, should be essentially completed, and the sulphur and phosphorus should be stabilized in a slag of the proper composition to prevent their reversion to the metal. The last of the ore additions will have been made and the carbon content will be falling at a slow, controllable rate to the desired value. The temperature

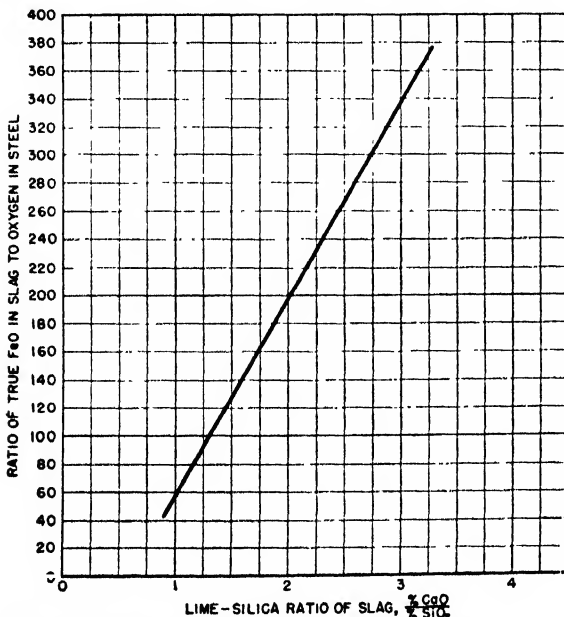


FIG. 15-39. Oxygen distribution between basic open-hearth steel and slag in the furnace as a function of the simple lime-silica ratio of the slag. (After Fettes and Chipman.)

is controlled by the rate at which fuel is burned, and additions to adjust the composition of the steel are made on the basis of chemical analyses made during this period.

Relation of Carbon to Oxygen in the Metal—The carbon and oxygen contents in the metal are closely related and the elimination of carbon is accelerated by a high proportion of active iron oxide in the slag. The mechanism by which carbon is removed is by diffusion of oxygen from the slag to the steel, where it reacts with the carbon to form predominantly CO and CO₂. The rate of diffusion of oxygen is relatively slow compared to reaction rates at the temperature involved, so that it is probable that the carbon-oxygen reaction closely approaches a state of dynamic equilibrium; that is to say, an equilibrium where the carbon content is dropping slowly but at no time is there a great excess of oxygen over that required for the chemical balance. The relation of carbon to oxygen in the metal is shown in Figure 15-40, where are plotted the results of analyses

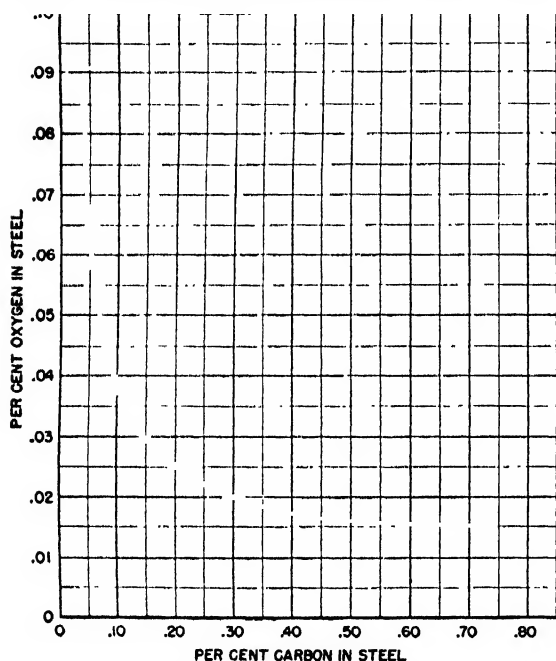


FIG. 15-40. Relationship between carbon and oxygen in basic open-hearth steel. (After Fetters and Chipman).

taken from a large number of open-hearth heats during the finishing period. The effect of temperature on the carbon-oxygen reaction is small but is such that more carbon and oxygen may exist together at higher temperatures.

Frequently it is necessary to stop the progress of a heat because the carbon content may be dropping too rapidly for accurate control by chemical analysis or because the drop in carbon content must be stopped until other control analyses may be completed. Holding the carbon content at a given level, or "blocking the heat," is accomplished by adding to the metal bath sufficient deoxidizing agent to lower its oxygen content below the amount required for reaction with carbon. These deoxidizers may be special-quality pig iron or one of numerous ferroalloys which contain elements having a high affinity for oxygen. Blocking the heat is only a temporary measure, and the effect of the blocking agent is often cancelled by diffusion of oxygen from the slag before the heat is tapped. In this case, the oxygen

content may again be estimated from the carbon analysis, provided the carbon-oxygen reaction has begun again to some degree.

The direct use of oxygen (or compressed air) for rapid elimination of carbon from the steel bath was discussed briefly in Section 7 of this chapter.

Chemistry of Steel in the Ladle and Molds—The chemistry of ladle additions is a subject of great importance, as indicated by the objects to be accomplished:

- (1) Final deoxidation to the point desired.
- (2) Final adjustment to the composition desired.
- (3) Final additions to develop special properties in the product, such as resistance to corrosion or aging, or machinability.

Every addition made is subject to reaction with the metal itself or some other element or compound in the metal that may be added to it. For this reason, ladle additions must be correlated with the treatments the heat has received in the furnace in order to produce the many types and grades of steel. In general, elements which are not oxidizable, such as copper or nickel, may be added to the furnace before tapping. Large amounts of deoxidizers, or easily oxidizable elements, are not commonly added to the furnace because of excessive losses due to the highly oxidized condition of the slag, and because of the danger of phosphorus reversion.

The degree of deoxidation obtained through ladle additions will depend on the kind and amount of deoxidizer used, as shown in Figure 15-41, which compares the effectiveness of several elements by showing the relative amounts in equilibrium with dissolved oxygen.

The additions which are made to molds are primarily for the purpose of obtaining the type of solidification desired. Small amounts of aluminum, usually a few ounces per ton of steel, may be used to control the rimming action in making rimming steel, while either ferrosilicon or aluminum are employed for control of the solidification of semikilled steels. In the manufacture

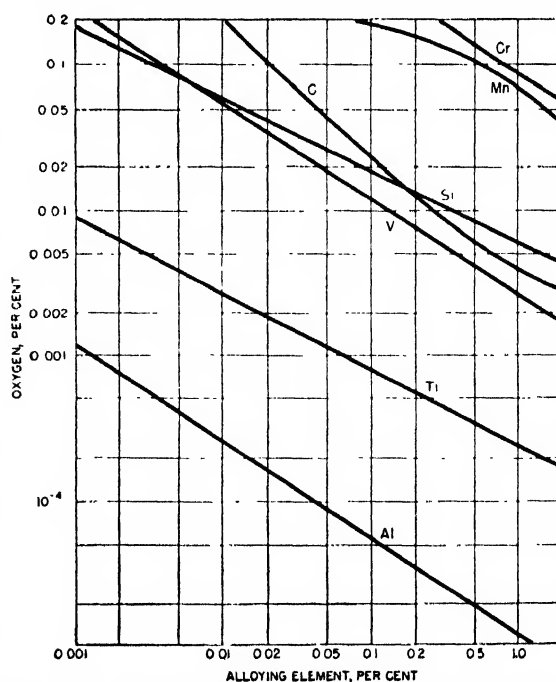


FIG. 15-41. Relative deoxidizing power of some of the elements. (From "Basic Open Hearth Steelmaking," published by AIME.)

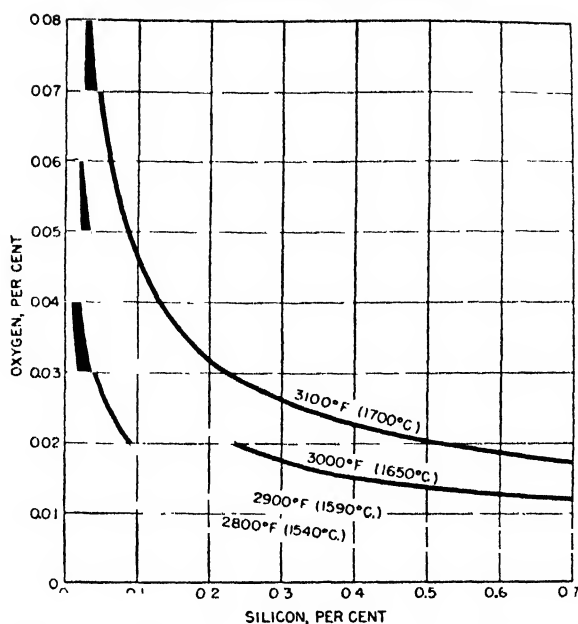


FIG. 15-42. Effect of temperature on the deoxidizing power of silicon. (Same source as Fig. 15-41.)

of some killed steels the deoxidation may be completed with aluminum during the filling of the mold. On some grades of steel which may be difficult to rim, "rimming agents," of which sodium fluoride is the chief example, are used to promote gas evolution. In general, mold additions are kept quite small because the opportunity for good mixing is not great and the pouring temperature is too low to insure the melting of large amounts of cold material.

NONMETALLIC INCLUSIONS

The nonmetallic inclusions which are normally found in steel may be traced to one of the following sources:

Refractory Material—Since the liquid metal is in contact with refractories until it is teemed into the molds, there is a possibility that small fragments of refractory material may be broken away or removed by erosion or reaction and enter the metal. Usually the particles are large and, unless trapped, will rise rapidly to the surface of the metal in the ladle or mold. As a result, non-metallic inclusions from this source are found only infrequently.

Oxygen and Sulphur—These elements are responsible for most of the nonmetallic inclusions, and only by their complete elimination from the process would it be possible to produce steels without inclusions.

While the steel composition is a factor, it is generally true that sulphur is quite soluble in steel at all temperatures above the melting point so that if sulphur is present, sulphide inclusions will not form until the steel solidifies, and thereby will be trapped in the ingot.

This is likewise true of oxygen in the case of steels which are not deoxidized. When strong deoxidizers, such

as silicon or aluminum, are added to the ladle, their oxides are formed with the dissolved oxygen in the steel. It is probable that the majority of these oxides will float to the top of the ladle and be eliminated. However, the deoxidizing power of most deoxidizers increases as the temperature falls so that oxides are continually forming as the temperature drops to the solidification point of the steel. This effect is shown for silicon in Figure 15-42.

The rate at which particles will rise from liquid steel is indicated by Stokes' law:

$$V = K \cdot \frac{2}{9\eta} \cdot r^2 g (d' - d)$$

where

V = Rising velocity

K = a constant

r = Radius of the particle

g = Acceleration due to gravity

d' and d = Density of liquid steel and inclusion, respectively

η = viscosity of liquid steel

The most important variable in this expression is " r ," the radius of the particle which should be large for rapid rates of rise. The most desirable deoxidizer would consequently be one which produces a liquid oxide, since liquids possess the ability to coalesce and form larger particles. Nevertheless, it will be evident from the above discussion that inclusions are inherently a part of steel as made by commercial processes, and it is their location and distribution throughout the ingot which will determine whether or not they are harmful.

SECTION 9

OPERATION OF AN ACID OPEN-HEARTH FURNACE

Materials for the Charge—In acid open-hearth practice, the initial charge consists normally of cold pig iron, or cold pig iron and scrap. No ore can be added with the

charge as in the basic process, for the iron oxide, being a base, would combine with the acid lining and rapidly destroy the bottom and banks. For the same reason the

melting of scrap alone would be bad practice for its oxidation products would have a similar detrimental effect. Hence, the use of hot metal (molten pig iron) to supply the whole of the pig iron part of the charge is impracticable, for it would necessarily have to be added after the scrap was hot and had been considerably oxidized.

The proportion of scrap to pig iron may vary over wide limits. As previously pointed out, Siemens originally used no scrap, and as ore was added to the pig iron after melting to hasten oxidation, his process was called the pig and ore process. Later, the Martin brothers used scrap with only enough pig iron to make the melt have the carbon content desired in the finished steel. As they used no ore, their method of working was called the pig and scrap process. The modern method, in which a charge of pig and scrap is melted, then "ored down" or "pigged up" as required, may be looked upon as a combination of these two methods. In it, the relative amounts of pig iron and scrap in the charge are controlled largely by plant and market conditions; that is, the supply and relative cost of these materials. If both are available, then the only thing considered in proportioning the charge is the carbon content it is desired the charge should have when melted. In times of extreme shortages of pig iron, substitutes such as Mexican graphite, silicon carbide, or silicon-steel scrap may be used with an increased scrap charge. Coke cannot be used because of its high sulphur content.

Grade of Scrap and Iron for the Charge—Only a trace of the phosphorus and none of the sulphur are eliminated in the acid open-hearth process. Indeed, the finished steel may contain a slightly higher percentage of both of these elements than the average of the charge. This increase is due to the fact that, while the weights of the sulphur and phosphorus remain practically constant, the weight of the metallic bath may be decreased appreciably, owing to losses through oxidation of iron and metalloids. In addition, the bath is likely to absorb some sulphur from the flame. Consequently, great care must be exercised to see that the average sulphur and phosphorus content of the charge is somewhat below that required in the finished steel. Silicon, manganese, and carbon are oxidized very readily in this process; consequently, require less consideration in the selection of materials. The specifications for acid open-hearth pig iron usually call for a silicon content under 2.00 per cent, manganese 1.00 to 2.00 per cent, and phosphorus and sulphur under 0.03 per cent. In steel for some grades of sheets, the phosphorus content may be somewhat higher, under 0.06 per cent being satisfactory. It is an easy matter, either at the blast furnaces or at the steel works, to secure a representative sample of any lot of pig iron, and its selection offers little difficulty. But it is almost impossible to secure a representative sample of the scrap for analysis; hence, the scrap must be selected by inspection and with much care and judgment. For example, in order to produce a steel with a phosphorus and a sulphur content each under 0.05 per cent, such scrap as tubes, pipe, Bessemer rails, and castings from unknown sources must be avoided. The size of the scrap is also of some importance. Unless the proportion of pig iron is large, light scrap makes it difficult to get a full charge into the furnace. Besides, light scrap is excessively oxidized in melting, unless extraordinary precautions are taken in charging. This oxidation of the iron not only decreases the yield, but the resulting iron oxide also combines with the siliceous lining and rapidly fluxes out the bottom and banks of the furnace.

Manner of Charging—A few small and isolated furnaces still may be charged by hand, but in the main all the larger furnaces will be charged by machine. As to

the order in which the pig iron and scrap are charged, it is unimportant which is charged first if the scrap is heavy or makes up less than half of the total charge. But if the scrap is light, many prefer to charge it ahead of the pig iron which, being spread out on the top, tends to shield the scrap from direct contact with the flame and to prevent its being oxidized excessively. Others make a practice of charging part of the pig iron on the bottom and part on top of the scrap. The pig iron on top melts first and, trickling down over the scrap beneath, the silicon and carbon it contains react with iron and manganese oxides as soon as formed, thus insuring that the latter do not come in contact with the bottom or banks. The added protection afforded by a little pig iron on the bottom is obvious, and is a great advantage. It not only will increase the yield but also will save much repairing of the bottom. As soon as the liquid basic oxides of iron and manganese come in contact with the acid oxides of silicon in the lining, silicates of these oxides are formed and become slag. Then, not only are the banks rapidly eroded away, but also the iron and manganese oxides are neutralized and lose their power of oxidizing and neutralizing the metalloids in the pig iron.

Melting—As soon as the furnace has been charged, the fuel, which is usually partly or wholly turned off during the charging, is turned on full, and the flame is reversed every twenty to thirty minutes in order to raise the temperature and melt down the charge as quickly as possible. The heat usually requires little attention during the first part of this period, so the time generally is occupied in preparations for tapping. The steel ladle requires a new nozzle and a new stopper, and the ladle and cinder pits must be cleaned out—duties performed by men on the pouring floor. On the charging floor the second helper cleans and repairs the steel runner, sets it in place, and sees that it is thoroughly dried; procures, or has brought to the furnace, the ferromanganese, ferrosilicon, anthracite coal, and other deoxidizers and recarburizers required. From four to five hours are required to melt the charge. Toward the end of this period, the first helper, or melter, will keep a close watch upon the action of the heat. If the charge contained a larger proportion (over half) of pig iron, especially if the iron carries a high silicon and low manganese content, the slag formed will be very thick or viscous and, in order to hasten the process and increase the yield, it is necessary to add a little ore or lime. Great care must be used in adding lime, to prevent the banks from being destroyed. On the other hand, if the charge was composed chiefly of scrap, or the pig contained a low percentage of silicon, it may be advisable, in order to save the banks and bottom from excessive scorification, to add some sand, old bricks, or other siliceous material.

Adjusting Conditions After Melting—After the charge has melted completely, the heat, if handled properly, should and usually does contain about 20 to 40 points (hundredths of a per cent) more carbon than is required in the finished steel. Some melters prefer to have the heat melt with a carbon content of more than one per cent in all cases, but for soft steels this practice will not improve the steel and may cause additional labor.

If the charge was made up largely of a low-silicon pig iron and the heat was given proper attention during the melting period, practically all of the silicon and manganese will have been oxidized and neutralized, forming the slag. This slag, at first black in color, is normally made up of about 50 per cent of bases, principally iron and manganese oxides, and about 50 per cent of acids, principally silica. Such a slag tends to be

self adjusting in that the slag will tend to retard the elimination of silicon and carbon from the metal, if the silica content of the slag is increased, whereas if the iron and manganese contents be increased, silica will be absorbed from the furnace banks. Where the melting has been such that considerable oxidation may have occurred, the first slag may be sufficiently oxidizing to complete the elimination of carbon without the necessity of ore additions. This first slag tends to adjust itself as follows: As the temperature rises, the carbon in the bath reduces iron oxide in the slag, causing the slag to grow lighter in color. Also, the slag gains more manganese oxide so that the color changes are black to brown to greenish yellow to a light green color, tinted according to the manganese it contains. These color changes are still sometimes used to follow the course of the heat. However, more recently there have been developed slag fluidity measurements which, in conjunction with temperature measurements, give a more accurate picture of the condition of the slag. The measurement is based on the fact that the silica content of the slag greatly affects its fluidity and this fact can be used to determine the need for additions to the slag to cause it to be more or less oxidizing to the bath. The changes, which the slag tests indicate are necessary, may be made by lime or ore for more oxidizing conditions or by carbon and the regulation of fuel input when the oxidizing power of the slag is too great.

Working the Heat—The problem now before the melter is to reduce the carbon content as rapidly but controllably as possible, and at the same time get the metal heated up to a temperature that will permit the heat to be tapped and teemed satisfactorily. In order to determine the carbon content of the bath, a fracture test of the metal may be taken, but more often an instrument such as the "Carbometer," or "Carbanalyzer" or chemical analysis is used.

If the metal appears to be too cold, attempts are made to raise its temperature by burning more fuel and reversing the flame frequently, before any necessary ore is added. If the carbon is relatively high and the bath is quiet and sufficiently hot, a few lumps of ore are added to start a boil. After this ore has had time to react with carbon, more ore may be added, but this is not done so rapidly that a large excess of iron oxide is created in the slag, as this would result in erosion of the banks of the furnace and cause excessive slag volume.

If, by virtue of a large ore addition, the carbon content becomes lower than desired, it may be restored by adding pig iron to hold the carbon at the desired level until the tapping temperature is reached. Thus, by frequent sampling of the bath and treating it as required, the carbon content and the temperature will be so adjusted that the heat may be tapped. Low carbon heats, under 0.30 per cent carbon, may be normally worked down to a carbon content of 0.10 per cent to 0.12 per cent and recarburized to the desired composition by ladle additions, and some melters prefer to do this. However, especially in the case of higher carbon heats, it is customary to "block" the heat with ferro-silicon at the required carbon content. Since no phosphorus is removed in this process, the addition of large amounts of pig iron or spiegel for recarburizing is inadvisable, unless low-phosphorus pig iron is available, lest the phosphorus content be increased beyond the limit allowed. Therefore, medium- and high-carbon heats are usually "caught on the way down."

Finishing Acid Open-Hearth Steel—During the last hour the heat is in the furnace, little ore is added so that the carbon content may drop slowly and under con-

trol to the desired value for tapping. However, the slag must not become too depleted in iron oxide, for if the temperature is high, as in the case of low-carbon heats, some silica may be reduced and the silicon content of the steel will be increased. This condition will be indicated to the melter by a cessation of boiling and by an increase in the viscosity of the slag. It may be prevented by the addition of a small amount of limestone, which will decrease the viscosity of the slag and make available iron oxide by combining with the silica of the iron silicates. In Table 15—VI are shown the compositions of three finishing slags illustrative of the widest variations to be expected under normal conditions. Only the significant components are given.

Table 15—VI. Composition of Acid Open-Hearth Slags (Per Cent)

Constituent	Heat Number		
	1	2	3
SiO ₂	54.25	52.06	55.60
FeO	25.87	20.38	28.90
MnO	11.27	20.52	10.23
P ₂ O ₅	0.045	0.035	0.022
Al ₂ O ₃	3.28	3.07	4.16
CaO	5.37	4.35	0.70
MgO	0.12	Trace	0.12

When the temperature has been raised to the proper level and the slag is in the proper condition for tapping, the heat may be blocked or allowed to work slowly down to the required carbon content. At this time, additions to adjust the amount of alloying elements may be made. If the charge for the heat has been selected with care, the preliminary analysis, taken when the charge was completely melted, will have shown that the phosphorus and sulphur specifications will be met. However, if error or lack of satisfactory control of raw materials causes either the phosphorus or sulphur content, or both, to be too high, the heat must be rejected in the case of most steel foundries, or applied to an order with higher specifications for these elements, for there are no corrective measures that can be applied to reduce their contents.

Tapping—When ready, the heat is tapped by methods described for basic furnaces.

Additions to the Heat—In the acid open-hearth process, additions to the heat are made both in the furnace and in the ladle. In making high-carbon heats in which the carbon is caught on the way down, most of the additions, including ferromanganese and ferro-chromium, are added in the furnace, as well as unoxidizable additions such as copper, nickel, and molybdenum compounds. In making medium and low-carbon steels, adjustments in carbon content are made in the ladle, preferably with anthracite coal of about pea size. To facilitate the addition, the coal usually is contained in paper bags, each holding a known weight. Ferromanganese also may be added in the ladle, for which purpose it usually is crushed to pass a 2-inch screen. Additions made to the steel in the ladle also include such materials as ferrovanadium, ferrosilicon, ferrotitanium, and aluminum. All ladle additions should be completed before much slag begins to flow from the furnace. Additions to the heat during the pouring of ingots usually are confined to aluminum.

Teeming—The proper manufacture of an acid open-hearth steel heat leads to tapping temperatures of the order of 3000° F. For the manufacture of small and

intricate castings where many molds must be poured, such a temperature may be necessary at the start of the pour in order that the last molds may be filled. In pouring large castings, however, a temperature of this magnitude may be too high for what has been established as good practice. Pouring temperatures and rates will depend on mold sizes and the use for which the castings are intended. Small molds are usually filled by "hand shanking" whereby sufficient metal for one mold is taken from the large ladle in a small hand ladle for pouring. Larger molds are filled by the conventional ladle with stopper and nozzle.

The temperature and rate conditions mentioned above also apply in the pouring of ingots whose sizes may vary from 500 pounds for small forgings to as much as 400,000 pounds for very large forgings. The types of pouring employed are top pouring, bottom pouring, and basket or tun dish pouring, which already have been discussed in Section 5.

Preparation of the Furnace for the Next Charge—Efficient operation of the furnace requires that the next heat be charged with little delay after tapping. This is possible only if the furnace is kept in good condition from day to day and heat to heat; consequently, as soon as the heat is out of the furnace, the melter inspects the interior carefully. Steel remaining in low places on the hearth is removed by rabbling and holes are filled with bottom sand. All steel and slag are cleaned from the tapping hole, which then is closed in the same manner as described previously. While the taphole is being made up, the banks are repaired by shoveling sand on them and fritting it into place. This work requires about an hour if the banks can be rebuilt with one layer of sand. If the banks have been eroded badly, the sand must be applied in two coats and about forty-five minutes are required for each coat. As soon as this repair work is completed, charging of the furnace may begin.

SECTION 10

CHEMISTRY OF THE ACID OPEN-HEARTH FURNACE

Chemistry of Melting—The elimination of silicon, manganese, and carbon is effected in two stages corresponding to the melting and boiling of the heat. Analyses show that during melting nearly all of the silicon and manganese and a part of the carbon are oxidized. Since no ore is charged in the acid process, the only source of oxygen for oxidation during this stage is the furnace gases, which consist of the products of combustion of the fuel and any excess air which may have leaked into the furnace. Carbon dioxide, water vapor, and oxygen in the furnace atmosphere will oxidize iron rapidly above a red heat. The extent to which the oxidation will proceed will depend on the proportions of these gases and the gaseous products of the oxidation in the atmosphere, and the temperature. This action is confined to the surface until the fusion point is reached. When the melting point of the iron oxides is reached, they begin to flow off, thus leaving the metal beneath exposed to further oxidation. This action applies to the scrap only, because the melting point of the pig iron is about the same or lower than that of the oxide.

Purifying Reactions—The fact that the melting point of the pig iron is lower than that of the scale is a fortunate circumstance, for otherwise the liquid scale would trickle to the bottom of the furnace and there react with the silica hearth to form a slag. With the pig iron in a molten state on the bottom of the furnace, these oxides of iron react with the metalloids, removing them from the bath and giving up some of the iron contained in the iron oxides.

After the greater portion of the silicon and manganese has been oxidized, a small part of the phosphorus may be oxidized; but later, when the slag becomes higher in silica, the phosphorus compounds will be reduced, returning the phosphorus to the metal. Therefore, seldom more than traces of phosphorus can be removed from the charge; in fact, the phosphorus content may increase slightly by virtue of the removal of the other metalloids.

Elimination of Carbon During the Melting Period—If the melting could be effected instantaneously, very little carbon would be oxidized until after the silicon and manganese were eliminated. But since melting takes place gradually, a part of the pig iron coming in contact with the highly oxidizing basic slag, both being in the liquid state, loses its metalloids, including carbon, quickly; consequently the percentage of carbon in the

bath, immediately after melting is complete, will be considerably lower than the average of the charge. While the amount of carbon thus oxidized cannot be predicted accurately, melters are quite successful in properly proportioning pig iron and scrap to the condition of the furnace and method of charging, to have the heat melt near the carbon content desired for proper finishing.

Chemistry of the Boil—By the time the charge has melted completely, a substantial portion of the silicon and manganese will have been oxidized to form a slag. If the melting has been rapid, there may not have been enough oxidation of the scrap to start a boil, so ore must be added. On the other hand, if the melting has been slow, enough oxidation may have occurred for the boil to be completed without the use of ores. If the carbon content is low, the melter will first attempt to increase the temperature of the bath by increasing the fuel rate and reversing the flame more often, but if the carbon content is high, for example fifty points over the content desired, the feeding of ore may begin as soon as the temperature is sufficiently high. If the temperature is too low when ore is added, its action is delayed and is too violent when it occurs. The mechanism of metalloid elimination involves the same chemical reactions as those discussed under basic practice. The ore additions are made gradually in order to keep the boil under control and to protect the banks of the furnace.

Chapter 16

ELECTRIC FURNACE STEELMAKING

SECTION 1

DEVELOPMENT AND PRESENT STATUS OF ELECTRIC MELTING FURNACES

Numerous types of furnaces utilizing electric current as the source of heat have been developed by a large number of investigators, but relatively few types have survived as practical tools for steelmaking.

Methods of Electric Heating—Electric current can be used for heating in only two ways: (1) by utilizing the heat generated in electrical conductors by their inherent resistance to the flow of current; and (2) by utilizing the heat radiated by the electric arc.

Two general methods of heating by resistance are possible: (a) the indirect method in which the charge is heated by radiation and conduction from separate resistors through which the current is passed, and (b) the direct method in which the current is passed through the metal charge or bath itself. The indirect method of resistance heating for steel-melting operations is not practicable, for many reasons. The direct method in which high-voltage, low-amperage current is transformed to low-voltage, high-amperage current that

passes through the bath or charge is successfully employed. The bath in this latter method acts as the secondary circuit for the current which is generated from a primary circuit by induction; the method being known, therefore, as induction heating.

Likewise, arc heating may be applied in two general ways: (1) The arcs may be made between electrodes supported above the metal in the furnace, which thus is heated solely by radiation from the arc. This method is known as indirect-arc heating. (2) The arcs may be made between the electrodes and the metal. In this second method, known as direct-arc heating, the current must flow through the bath, so that the heat developed by the electrical resistance of the metal, though relatively small in amount, is added to that radiated from the arcs. This plan makes it possible to use two types of furnaces, namely, those with a non-conducting bottom, and those with a conducting bottom. The latter type has not been successful for practical steelmaking use.

Table 16—I. Classification of Electric Furnaces for Making Steel According to Principles of Heating Employed.

Indirect —The current is passed through a special resistor to generate heat, which is used to heat the charge by radiation, convection and conduction. Such furnaces are used for heat treating but not for melting steel.	
Resistance Furnaces	Using current from low voltage transformers. Not successful for melting steel.
Direct —The current passes through the material to be heated.	[Low frequency—Using a core transformer with the bath forming the secondary circuit.
	Induction High Frequency Coreless Induction. Current of high frequency is passed through a coil surrounding a crucible containing the charge.
	Direct Current Arc —Not used.
Indirect or Independent Arc —The bath is heated by an arc or arcs above it.	Single phase {rolling furnace with horizontal electrodes. furnaces for special purposes.
	Alternating Current Two phase {Straight arcs. } not used. Deflected arc.
	Three phase {Straight arcs. } not used. Repel-arc.
	Furnaces may be stationary, oscillating, or rolling.
Arc Furnaces	Series Arc. Current arcs from one electrode to the bath, passes through the bath, and arcs to another electrode. {Single phase Two phase Three phase
Direct Arc —The current arcs from electrode to bath.	
Combination Arc and Resistance —Use the arc and resistance of refractory bottom material for heating charge.	Single Arc. Current arcs from one electrode to the bath, passes through the bath, and out through an electrode in the bottom of the furnace. {Single phase Two phase Three phase



ALLOY STEEL MACH

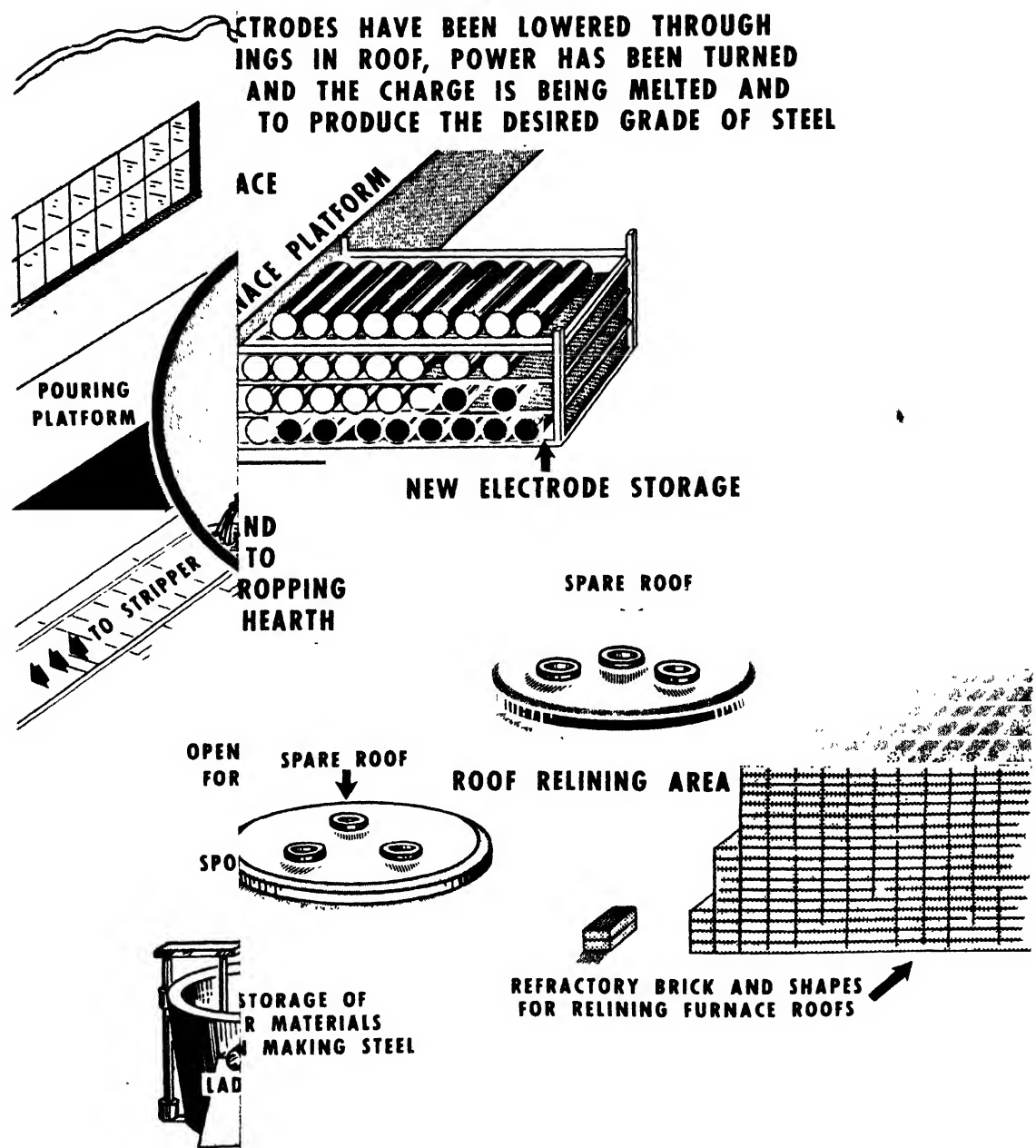
ELECTRIC ARC FURNACE

(THIS DRAWING IS ENTIRELY SCHEMATIC AND NOT TO SCALE)

SCHEMATIC DRAWING. THE VARIOUS PIECES OF EQUIPMENT ARE NOT DRAWN TO THE SAME SCALE. THIS DRAWING IS INTENDED ONLY TO SHOW THE MAJOR FEATURES OF A PLANT MAKING STEEL. THE ARRANGEMENT OF FURNACES, SOURCES OF RAW MATERIALS AND DISPOSITION OF THE SAME ARE NOT HERE FOR TRADE PURPOSES.

HEAT No. 3

ELECTRODES HAVE BEEN LOWERED THROUGH THE OPENINGS IN ROOF, POWER HAS BEEN TURNED ON AND THE CHARGE IS BEING MELTED AND STIRRED TO PRODUCE THE DESIRED GRADE OF STEEL



In furnaces with a non-conducting bottom, the path of the current is through one electrode and thence through the arc between the foot of the electrode and the bath, then through the bath and up through an arc between the bath and an adjacent electrode, completing the circuit through this second electrode. Path of the current through furnaces with conducting bottoms would be from the electrode or electrodes above the bath, through the arc into the bath and thence out of the furnace through an electrode forming part of the bottom in contact with the bath.

Table 16—I presents a listing of basic principles of electric heating which also will serve as a basis for classifying electric furnaces as to type. From among the many types listed therein, only two appear to have been able to stand against practical tests and competition as suitable for melting steel; these are the direct-arc furnace (series arc) originally developed by Heroult, and the high-frequency coreless induction furnace, which are discussed in Sections 2 and 5, respectively, of this chapter.

HISTORICAL DEVELOPMENT OF ELECTRIC MELTING FURNACES

It has been said that arc-type furnaces had their beginning in the discovery of the carbon arc by Sir Humphrey Davy in 1800, but it is more proper to say that the practical beginning of the art was the work of Sir William Siemens, who in 1878 constructed, operated and patented furnaces operating on both the direct-arc and indirect-arc principles.

At this early date, the availability of electric power was limited and its cost high; also, carbon electrodes of the quality required to carry sufficient current for steel melting had not been developed. Thus, the development of the electric melting furnace awaited the expansion of the electric-power industry and improvements in carbon electrodes.

Direct-Arc Furnaces—The first successful commercial direct-arc steelmaking furnace was placed in operation by Heroult in 1899, and the first shipment of electric steel was a carload of bars from Heroult's plant at La Praz to the firm of Schneider and Company at Creusot, France, on December 28, 1900. The Heroult patent, stated in simple terms, covered single-phase or multi-phase furnaces with the arcs in series through the metal bath.

This type of furnace, utilizing three-phase power, has been the most successful in the production of steel. During recent years, many large-capacity furnaces with 20-foot diameter shells holding an average charge of 185,000 pounds were placed in successful operation. During 1954, two furnaces designed for a maximum charge of 200 tons were installed; these furnaces have an inside diameter of 24½ feet and a hearth depth of 3 feet, 9 inches. Such units have been made possible by the increased availability of electric power, large graphite electrodes, and improved refractories, along with better furnace design and melting practices. These factors will be discussed in detail later. A typical large modern unit is shown in Figure 16—1.

Simultaneous with and subsequent to Heroult's success with the direct-arc principle, many investigators abroad directed their attention to developing furnaces employing the same principle. However, in this country, there were no developments along arc-furnace lines until the first Heroult furnace was installed in the plant of the Halcomb Steel Company, Syracuse, New York, which made its first heat on April 5, 1906. This was a single-phase, two-electrode, rectangular furnace of four tons capacity. Two years later a similar but smaller

furnace was installed at the Firth-Sterling Steel Company, McKeesport, Pennsylvania, and in 1909, a 15-ton, three-phase furnace was installed in the South Works of the Illinois Steel Company, now a part of the United States Steel Corporation, in Chicago, Illinois. The latter was, at that time, the largest electric steelmaking furnace in the world, and was the first round (instead of rectangular) furnace. It operated on 25-cycle power at 2200 volts and tapped the first heat on May 10, 1909.

The foregoing furnaces all were used for making steel for ingots. The first electric furnace for the production of steel for commercial castings was that of the Treadwell Engineering Company, Easton, Pennsylvania. It was a single-phase, two-electrode furnace of two tons capacity, designed along the lines of the Halcomb and Firth-Sterling furnaces. It was operated first in August, 1911.

About the same time, the General Electric Company began to experiment with the design of direct-arc electric furnaces, with the view of developing a market for electrical equipment required for their operation, and built three or four units. Other furnace designs followed: the Snyder, Ludlum, Vom Baur, Booth-Hall, Moore, Green, Swindell, and Volta, the last-named being of Canadian origin.

Indirect-Arc Furnaces—The first work on indirect-arc furnaces was done by Stassano, in Italy. His design consisted of a vertical, cylindrical shell, with three electrodes spaced 120 degrees apart and entering the furnace just above the bath. A furnace of this design was installed in the plant of the Clark Equipment Company, Buchanan, Michigan, in 1911.

About the beginning of World War I, Rennerfelt of Sweden developed an indirect-arc furnace with two horizontal electrodes and one vertical electrode, so connected electrically that the arc was "blown" down on the charge or bath, and the horizontal electrodes were arranged to tilt downward so that they could arc directly on the bath.

None of the indirect-arc furnaces came into very great use because maintenance was difficult and power consumption was high.

The Induction Furnace—Another type of electric melting furnace, used to a certain extent for melting high-grade alloys, is the high-frequency coreless induction furnace (Figure 16—2), which gradually replaced the crucible process in the production of complex, high-quality alloys used as tool steels. It is used also for remelting scrap from fine steels produced in arc furnaces, melting chrome-nickel alloys, and high-manganese scrap.

The induction furnace had its inception abroad and first was patented by Ferranti in Italy in 1877. This was a low-frequency furnace. It had no commercial application until Kjellin installed and operated one in Sweden. The first large installation of this type was made in 1914 at the plant of the American Iron and Steel Company in Lebanon, Pennsylvania, but was not successful. Some other low-frequency furnaces, however, have operated successfully, especially in making stainless steel.

A successful development using higher-frequency current is the coreless high-frequency induction furnace. The first coreless induction furnaces were built and installed by the Ajax Electrothermic Corporation, who also initiated the original researches by E. F. Northrup leading to the development of the furnace. For this reason, the furnace is often referred to as the Ajax-Northrup furnace.

The first coreless induction furnaces for the production of steel on a commercial scale were installed at Sheffield,



FIG. 16-1. A large, modern, three-phase Heroult electric-arc furnace.

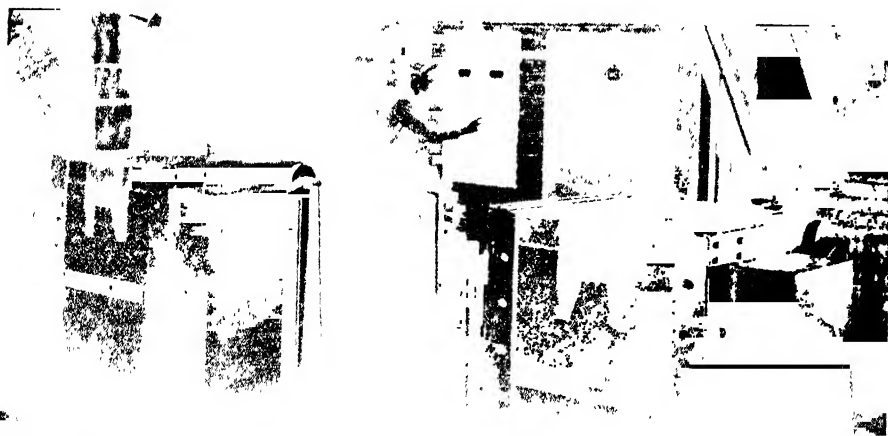


FIG. 16-2. Dual installation of coreless electric induction melting furnaces, with control panel in background. (Courtesy, Allis-Chalmers Mfg. Co.)

England, and began the regular production of steel in October, 1927.

The first commercial steel furnaces of this type in the United States were installed by the Heppenstall Forge and Knife Company, Pittsburgh, Pennsylvania, and were producing steel regularly in November, 1928. Each furnace had a holding capacity of 600 pounds and was served by a 150-kva motor-generator set transforming 60-cycle current to 960 cycles. The crucibles were monolithic, 6 inches in diameter, and 36 inches in depth. Time of an ordinary heat was 55 minutes. The coreless type of induction furnace normally operates on alternating current at a frequency of approximately 1000 cycles, produced by a motor-generator set of special design with the power transmitted over co-axial cables to the primary coil of the furnace. Recently, mercury-arc rectifiers, which have the advantage of higher efficiencies over motor-generator sets, have been developed to supply the high-frequency power. Some details of operation of this type of furnace are given in Section 5 of this chapter.

The primary coil of the coreless induction furnace is constructed of water-cooled copper tubing, positioned toward the inside of the furnace shell but covered with a layer of refractory material. The shell must be constructed of non-conducting materials, since the high-frequency power applied to the primary coil otherwise would induce current in the shell as well as the charge, and generate heat in the former. When operating, the high-frequency power applied to the primary coil creates a magnetic flux which passes through the charge, and the charge acts as the secondary winding of a transformer having a single turn. This induced current melts the charge by the heat developed due to the electrical resistance of the charge.

Commercial use of the high-frequency furnace for melting is not extensive in the steel industry. Those in use are small in size, ranging in capacity from about 500 pounds to 2,000 pounds. The largest one in the United States is of four-ton capacity and has for a power source a 1200-kilowatt motor-generator set.

Due to the time required to prepare the scrap and place it in the furnace, the usual installation consists of two furnaces for each generating unit. Thus, one heat can be melting while the next is being prepared. The finished heat can be disconnected from the power source and the furnace used as a ladle and carried by a crane to the molds for pouring, while the second furnace is connected to the power system.

Electric Reduction Furnaces—Electric reduction furnaces are used for smelting ores in the production of ferromanganese, ferrosilicon, ferrochromium, and so on. These furnaces differ from the steelmaking furnaces in that production is continuous, as in a blast furnace; the charge is placed in the furnace at the top and the molten product tapped near the bottom. They are designed either as single-phase or three-phase units.

In the single-phase units, a single carbon electrode is suspended in the center of the furnace, with the carbon bottom of the furnace serving as the second electrode. This is an example of a furnace with a conducting bottom. In the three-phase furnaces, the three electrodes are spaced as in the arc furnace and are buried in the charge. These units normally have an open top, and the charge either is shoveled in or introduced by chutes. The electrical load on this type of furnace differs from that on the arc furnace used for steelmaking. Since charging is continuous, the load is fairly steady, and the power factor can be maintained around 90 per cent. The voltages used are lower than in the arc furnaces, and the current density greater. These requirements necessitate

a careful design of the secondary circuits to keep reactance to a minimum. For some products, and on 25-cycle systems, units of electrical capacities up to 18,000 kva have been designed.

Furnaces of Special Design—There have been other types of furnaces designed that are worthy of mention. The Hering furnace was a conventional unit except that the bath was deeper than usual and the hearth contained one or more resistance tubes at the bottom which served as water-cooled electrodes. Molten metal was required for the first charge, and only two-thirds of the heat was tapped. The effect of the current flowing through the molten metal in the resistor tubes was to pinch off the column of metal and eject it from the tube. The furnace had no commercial application.

Another furnace of European design was a combination of open-hearth and electric-arc furnace principles. Still another was a combination of low-frequency induction furnace and pneumatic converter, the tuyeres of the latter being equipped so that a reducing gas might be introduced into the furnace.

Field of Application of Electric-Arc Furnaces—In the preceding section, it was pointed out that the electric-arc furnace for steelmaking first was introduced into the United States in 1906. The real expansion of this phase of the industry, however, has been more recent, with respect to both basic-lined and acid-lined electric-arc furnaces. The first development was due to the wide use of the SAE low-alloy steels. Then came the development and expanding use of higher alloy and stainless steels, which have been largely responsible for the more recent increasing use of basic-lined electric-arc furnaces in the steel industry.

The steel-casting industry found that acid-lined electric-arc furnaces were well qualified to meet their needs in regard to control of operations and quality of product. This has been the important factor in the growing use in foundries of this type of furnace.

Both the acid and basic processes for making steel in electric furnaces were used extensively during World War II, and production has continued at a high level. The basic process, especially, is in a good position to maintain a high production rate because of its ability to use both high-alloy steel scrap, lower grades of alloy scrap, and plain-carbon steel scrap in making steels that will meet rigid chemical, mechanical-property and cleanliness requirements.

The basic process, for which the electric-arc furnace utilizes a basic bottom of magnesite or dolomite and sidewalls and roof of either basic or silica brick, accounts for practically all of the electric-furnace steel produced in ingot form. Furnaces of this type usually are found in integrated steel works having installations for making steel by other processes also, where facilities already are available for stripping the ingots, heating them, and rolling or forging them to the desired size and shape. Such integrated plants make possible the utilization of the cold-melt process or one of the hot-metal methods such as duplexing, triplexing, or the refinement of molten open-hearth steel by special slags which can be prepared and removed more conveniently in the electric furnace than in the open hearth.

The acid electric furnace, with an acid bottom of ground ganister and silica-brick sidewalls and roof, seldom is used outside of steel foundries and forging shops. Straight-carbon, low-alloy and some high-alloy steels can be made by the acid process.

The importance of electric furnaces as steelmaking units in ingot-producing plants in the United States is indicated by the annual tonnages shown in Table 16—II, which also shows the increasing trend in production of

Table 16—II. Annual Production of Electric-Furnace Steels⁽¹⁾
(Steel Ingots and Steel for Castings—Net Tons)⁽²⁾

Year	Electric-Furnace Steel		Total All Processes, Including Electric		Percentage Represented by Electric-Furnace Steel	
	Capacity	Production	Capacity	Production	Capacity	Production
1954	10,448,640	5,436,054	124,330,410	88,311,652	8.4	6.2
1953	10,232,450	7,280,191	117,547,470	111,609,719	8.7	6.5
1952	8,232,850	6,797,923	108,587,670	93,168,039	7.5	7.3
1951	7,554,610	7,142,384	104,229,650	105,199,848	7.2	6.8
1950	6,871,290	6,039,008	99,392,800	96,836,075	6.9	6.2
1949	6,112,870	3,782,717	96,120,930	72,978,176	6.4	4.9
1948	5,396,750	5,057,141	94,233,460	88,640,470	5.7	5.7
1947	5,076,240	3,787,735	91,241,250	84,894,071	5.6	4.5
1946	5,500,290	2,563,024	91,890,560	66,602,724	6.0	3.8
1945	5,455,890	3,456,704	95,505,280	79,701,648	5.7	4.4
1944	5,350,880	4,237,699	93,652,290	89,641,600	5.7	4.7
1943	4,554,980	4,589,070	90,292,660	88,836,512	5.0	5.2
1942	3,737,510	3,974,540	88,569,970	86,031,931	4.2	4.6
1941	2,586,320	2,869,256	84,152,292	82,839,259	3.1	3.5
1940	1,882,630	1,700,006	81,619,496	66,982,686	2.3	2.5
1939	1,725,086	1,029,067	81,828,958	52,798,714	2.1	1.9
1938	1,490,858	565,627	80,185,638	31,751,990	1.9	1.8
1937	1,326,788	947,002	78,148,374	56,636,945	1.7	1.7
1936	1,147,221	865,150	78,164,300	53,499,999	1.5	1.1
1935	1,053,370	606,471	78,451,930	38,183,705	1.3	1.6

⁽¹⁾ From Annual Reports of American Iron and Steel Institute.

⁽²⁾ The figures include only that portion of the capacity and production of steel for castings used by foundries which were operated by companies also producing steel ingots.

electric-furnace steels during the twenty-year period from 1935 through 1954. It should be noted that these figures do not include steel made in electric furnaces by foundries that do not produce ingots. This increase was due to several factors, chief of which was that electric furnaces now are capable of producing a wide range of steels, and no longer are considered as suitable only for making a limited range of high-cost, high-quality steels of special compositions for extremely exacting uses. Originally, electric furnaces were accepted widely as an improvement over the crucible process for producing steels for tools, dies, watch springs, cutlery, and other special requirements, and gradually supplanted that process almost entirely. However, as experience was gained and costs decreased, the field for economical use of electric furnaces expanded greatly.

In addition, acid-lined electric-arc furnaces have sometimes replaced small open-hearth furnaces and pneumatic converters to produce steel for castings.

Although the production of electric-furnace steels during normal times represents only about six and one-half per cent of the total tonnage of ingot steel produced in this country, the electric-furnace products represent practically all of the stainless, constructional alloy, tool, and special alloy steels used in the chemical, automotive, aviation, machine-tool, transportation, food processing and many other important industries. While at the present time the electric-arc furnace cannot compete with the larger open-hearth furnace as to cost in the production of the commoner grades of steel, yet the electric furnace, if operated in a favorable scrap-producing area with favorable power rates, in many cases can produce ordinary carbon steels at costs comparable to, and sometimes less than, open-hearth costs.

General Comparison of Basic and Acid Electric-Arc-Furnace Processes—Following are factors in deciding

which of the two processes is best suited for the economic production of steel to meet demands with respect to grade of steel and its ultimate use. In this comparison, only the acid process involving complete oxidation of the bath, using a single, oxidizing slag, is considered.

1. Slags in the acid process are more siliceous than those in the basic process. Since acid slags do not react with the steel bath to remove phosphorus from the steel, the use of more expensive, carefully selected scrap and other raw materials of low phosphorus content are imperative. Conversely, basic slags do react with and retain phosphorus, and practically all types of scrap and raw materials can be used in the basic electric furnace. It may be mentioned here that sulphur cannot be removed in the acid process, but can be in the basic process, as will be discussed later.

2. Oxidizing action takes place in a shorter time in the acid process and the time required for "working the heat" also is shorter than in the basic process, mainly because of the selected scrap used in the acid furnace as compared with the wide variety of scrap used in the basic process.

3. In the acid process, because of the selected materials making up the charge, a one-slag process predominates, while a multiple-slag practice is used more widely with the basic process.

4. Iron loss is lower with the single-slag acid process than with the multiple-slag basic process because the oxidizing period is shorter, less metal is trapped in the much smaller volume of slag, and, since there is no slagging-off in the single-slag acid process, there is no loss of metal from this source.

The metallurgical differences in the two processes indicate that none of the advantages of one process over the other is sufficient to justify the selection of one or the other process without full consideration of all fac-

tors, such as flexibility and economies. Economy in production combined with adequate performance of finished product have been the determining factors. The result has been the previously mentioned prevalence of the acid process in the steel-castings industry and the basic process in making ingots for rolling.

Cold Scrap vs. Hot Metal—When the first electric melting furnaces were installed in the steel industry, electric power was not available in large amounts. Since the melting operation requires about three times the power that is required by the refining period, it was thought then that the cost of this extra electric power was too high to permit economical melting of cold charges. For this reason, the early installations at Halcomb and Illinois Steel were provided with a separate source of hot metal to provide a molten charge. The former had a tilting 30-ton basic open-hearth furnace and the latter a Bessemer converter department. Later, as more electric power became available, it was recognized that cold melting was feasible.

During World War II, in order to increase production, several installations were made where the practice of using hot-metal charges was revived. The largest installation had nine 70-ton electric furnaces operating in conjunction with tilting open-hearth furnaces to produce steel by a combination method. These door-charged furnaces now operate mainly on cold-melt practice, with some hot metal employed, when available, to aid charging. At the present time, there is no appreciable tonnage of steel being produced in electric furnaces using hot-metal charges. With the advent of new pneumatic processes (Chapter 14), it is possible that these processes may be used to prepare molten charges for finishing in electric furnaces to produce steel.

Advantages of the Electric-Arc Furnace—The increasing number of basic electric-arc furnace installations is evidence that, as steelmaking units, these furnaces provide certain advantages. Among the more important considerations for selecting this method over other steel production methods are the following:

Practically all of the known grades of steel can be produced in the basic electric-arc furnace. The products made include the plain carbon steels of rimmed, capped, semi-killed and killed types, low-alloy SAE-AISI constructional steels, high-manganese steels (up to 14 per cent), high-silicon steels (up to 5 per cent), aluminum steels (up to $4\frac{1}{2}$ per cent), the entire range of stainless steels, super-alloy steels for high-temperature applications, and high-speed and other alloy tool steels.

The basic electric-arc furnace may be selected as the more economical steel producing method when: (1) carbon and low-alloy steel production requirements are insufficient to justify use of the blast furnace—open hearth combination to produce steel, (2) facilities are installed in industrial areas of high steel-scrap availability but at a distance from natural sources of coke, limestone and high-grade iron ores, (3) the nature of subsequent processing is such that steel-production requirements are intermittent or molten metal must be supplied within controlled time limits.

The electric-arc furnace process is capable of producing killed-steel grades with very low residual phosphorus because: (1) the initial charge contains less phosphorus than the charges of steelmaking processes containing blast-furnace iron, and (2) the phosphorus-bearing oxidizing slags of the electric furnace heat may be removed prior to furnace deoxidation and thus avoid phosphorus reversion to the metal.

The greater desulfurizing power of basic electric-arc furnace reducing slags facilitates the production of steels with lower sulphur residuals.

The absence of an oxidizing heat source permits heats that have been deoxidized in the furnace to be held without becoming rapidly reoxidized; this feature, coupled with the ability to use reducing slags, makes it possible to produce steels containing fewer nonmetallic inclusions.

The production of steels alloyed with high percentages of oxidizable metallic elements can be produced more efficiently in the basic electric-arc furnace as such alloying elements or ferroalloys can be added in the furnace under reducing conditions at a high efficiency of recovery and in any amount required. Moreover, with proper slag manipulation, a relatively-high efficiency of recovery of oxidizable elements, such as chromium, manganese, etc., can also be obtained from alloy-steel scrap.

Because it can be tilted to pour off slag, the electric-arc furnace can be operated with slag volumes controlled to a minimum (2 to 4 per cent of the bath weight) and therefore the slag composition can be adjusted and controlled quickly by relatively small additions of slag-making, oxidizing or deoxidizing materials. Therefore, the steel may be treated under oxidizing, reducing or neutral slags, or any succession of such slags.

Under oxidizing slags, the electric furnace with a definite regulation of heat input provides good control of carbon content at melt-down and later while working the heat. Also, the removal of this oxidizing slag, with its comparatively high phosphorus content, limits the amount of phosphorus reversion that can occur by taking a large proportion of this element out of the field of action.

A reducing slag in the basic electric furnace, following the slagging off of an oxidizing slag, makes possible a high degree of refinement of the bath and minimizes loss of alloys by oxidation. Objectionable elements, such as sulphur, can be removed to a high degree after the oxidizing slag has been replaced by a reducing slag. The reducing slag also decreases the oxygen content of the bath to a low value; consequently, relatively few deoxidation products are formed when the final deoxidizers are added because less of the latter are needed. The deoxidation products that are formed have a better opportunity to rise through the steel bath and enter into the slag than would similar substances formed if large amounts of deoxidizers had to be added in the ladle. As a result, relatively few sulphide and oxide inclusions are present in electric-furnace steel, remembering, however, that limited circulation of the bath under this reducing slag necessitates a long refining time.

Disadvantages of the Electric-Arc Furnace—The disadvantages common to the operation of electric-arc furnaces are largely a matter of economics. The cost of auxiliary equipment, operating labor, power, electrodes, and refractories usually are higher than similar costs in the open-hearth and other processes. The costs exclusive of net metallic charge show the electric furnace to be at a disadvantage which can be overcome only partially by faster time of heat and greater efficiency. Individual open-hearth furnaces, in general, are of considerably larger capacity than electric furnaces; the possibility of overcoming the size differential in furnaces of the two types is limited by power problems. Larger electric-arc furnaces require higher voltages to transfer energy economically, and the longer arcs in larger furnaces can cause excessive damage to roof and lining. Peak loads during melt-down periods may increase the difficulties of the power plants which must maintain sufficient power for regular requirements during this same period.

While the foregoing is true with respect to *physical size* from the standpoint of the size of individual heats, the *production rate* of a standard 20-foot shell diameter electric furnace in terms of tons of finished steel produced per hour will compare favorably with 150-ton or larger open-hearth furnaces. Also, in 24-foot, 6-inch

diameter electric-arc furnaces recently installed, heats of nearly 200 tons have been made. Thus, the former advantage of the open-hearth furnace with regard to heat size is being overcome, and other factors (some of them economic and some metallurgical) now are the criteria for comparing the two methods of steelmaking.

SECTION 2

DIRECT-ARC ELECTRIC FURNACES AND AUXILIARIES

Mechanical Parts—All furnaces of the Heroult type are designed to tilt in two directions, the one for pouring and the other for slagging. Heroult furnaces are mounted on toothed rockers which rest on and intermesh with toothed rails. They are tilted by a motor-driven rack-and-pinion mechanism, usually placed underneath the furnace. On large furnaces where a slag-off operation is required, the tilting mechanism is designed to permit movement of a car-mounted slag pot through the foundations to the tapping pit.

Capacity of Furnaces—The inside diameter of the shell determines the capacity of the electric furnace. Many standard sizes have been developed to provide for the different practices and types of scrap used in various shops. Heroult furnaces now are being built by American Bridge Division of United States Steel Corporation in sizes of 7-foot shell diameter and up (Table 16—III). They are usually equipped with roof-moving mechanisms (described later) to permit top charging. Door-charge furnaces and furnaces of special design, including units equipped with austenitic-steel bottoms to permit induction stirring, are also built.

Table 16—III. Heroult Furnace Sizes, Capacities, and Transformer Ratings.

Inside Shell Diameter	Usual Charge (Pounds)	Maximum Rated Charge (Pounds)	Usual Transformer Capacity (kva)
7'0"	6000	8000	1500/1800
8'0"	8000	10,000	2000/2500
9'0"	12,000	15,000	3000/4000
10'0"	16,000	20,000	4000/5000
11'0"	24,000	28,000	5000/6000
12'0"	30,000	35,000	6000/7500
13'6"	40,000	50,000	7500/10,000
15'0"	70,000	80,000	10,000/12,000
17'0"	100,000	120,000	12,000/15,000
18'0"	120,000	150,000	15,000/17,500
19'0"	150,000	180,000	17,500/20,000
20'0"	170,000	200,000	20,000/25,000
22'0"	240,000	280,000	25,000/30,000
24'6"	360,000	400,000	30,000/35,000

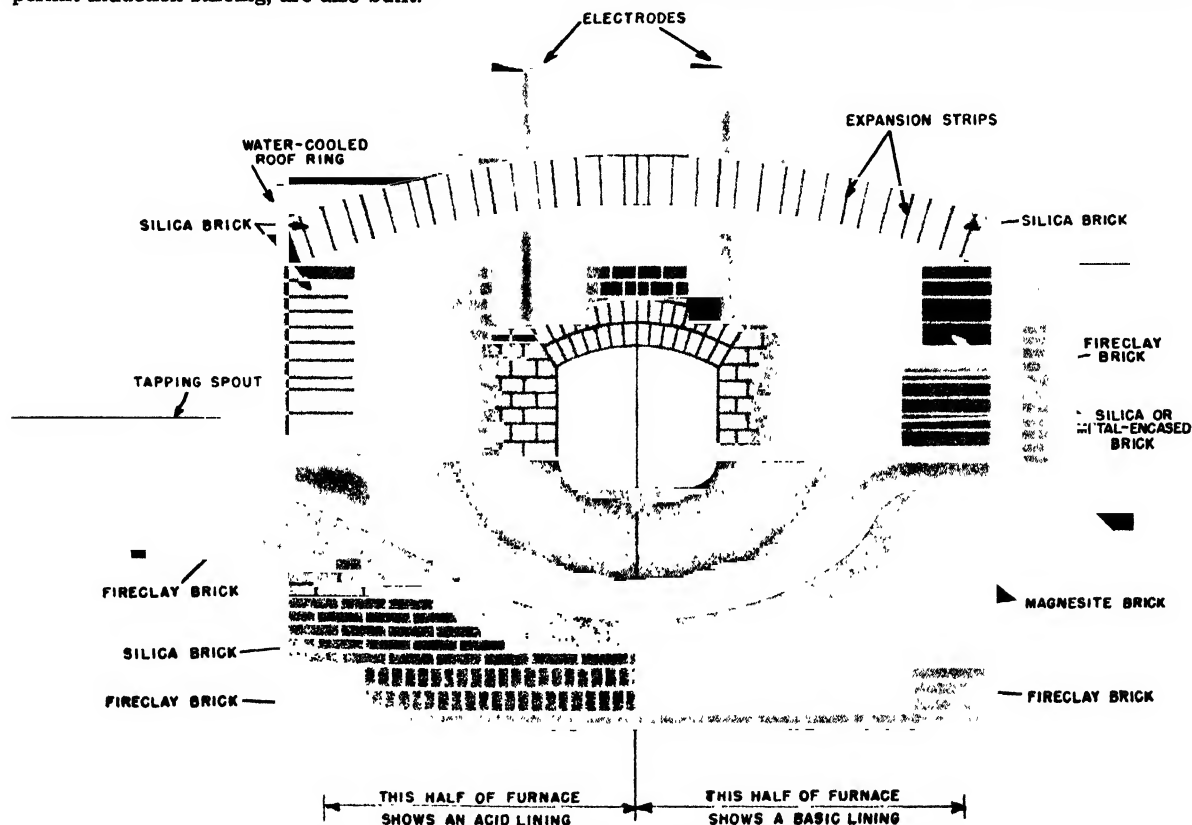


FIG. 16—3. Schematic cross-section of a Heroult electric-arc furnace, indicating typical refractories employed in (left) an acid lining and (right) a basic lining. Although only two electrodes are shown in this section, furnaces of this type (which operate on three-phase current) have three electrodes. (Courtesy, Harbison-Walker Refractories Company.)

The Furnace Shell and Lining—The shell of the modern electric-arc furnace is usually cylindrical in shape. It can be either riveted or welded, and is braced with structural-steel sections. As the development of the electric-steelmaking process has required the design of larger, more complex, higher-powered furnaces, problems have arisen which affect design, operation and maintenance. The shell has become a source of increasing trouble due to warping, burning and rupture. The shell on most existing large furnaces is an integral part of the furnace structure. Damage to the shell can weaken the structure and cause misalignment of operating parts mounted on it.

There are two methods for overcoming this difficulty on large furnaces:

(1) Build the shell as a unit structurally separate from the rest of the furnace structure; that is, to require the shell only to retain the refractories and charge while the operating parts of the furnace are supported on an independent structure unaffected by shell distortion.

(2) Eliminate rigid connections between the steel plates of the shell and the supporting yokes and buckstays, at least above the hearth line, to allow these parts to expand and contract independently of each other.

The shell of a cylindrical furnace has a flat bottom which can be laid first with several layers of clay, magnesite, or silica brick as required (Figure 16-3). As the side wall is constructed, the brick is set back to provide ledges for the granular material that will, after sintering in place, form the working bottom or hearth. Modern practice utilizes a much smaller amount of granular material than formerly was employed, and, with this design, it is possible to recover a large proportion of the bottom refractories when relining becomes necessary. The brick is placed easily around the door jambs, arches and in the door lining, as both jambs and doors have flat surfaces. The roof is a simple flat dome, and the water-cooled skewback type of

roof ring eliminates the necessity for special skew-shaped brick.

CHARGING METHODS

Charging methods for electric-arc furnaces fall into two classes—**top charging** and **door charging**.

Top-Charged Furnaces—There are two designs for modern top-charged furnaces: the **swing type** and the **gantry lift**.

In the swing type (Figure 16-4), the roof and the supporting structure for the electrode masts are lifted and swung to one side by motor-driven or hydraulic equipment.

The gantry-lift type, shown in Figure 16-5, has the electrode masts and the roof-raising mechanism built into a gantry crane that travels on rails along the charging floor. When the furnace is to be charged, the electrodes are raised to clear the shell, the roof is lifted and moved by the crane to one side of the furnace.

For medium-sized and large furnaces, top charging has the advantage of speed, as the entire charge can be placed in the furnace by drop-bottom buckets in a short time (Figure 16-5). In addition, very large pieces of scrap can be charged without being cut into charging-box sizes. However, scrap dropping from a considerable height results in a shock to the furnace bottom. To avoid this, a cushion of light scrap should be placed on the bottom by a magnet and this cushion used to absorb the shock.

Until recently, top-charge units of larger size were not extensively used. With heavy scrap, standard open-hearth machine charging offered and still offers an excellent method of placing the charge. However, within the last few years, the availability of heavy melting scrap has decreased, with considerable quantities of light scrap available. In utilizing top-charge furnaces, the furnace may be filled completely with light scrap which can be melted quickly by the use of transformers that can supply power at high rates to the furnace.



FIG. 16-4. A very large, modern Heroult electric-arc steelmaking furnace, with swing-type roof moved aside to permit top charging of the furnace by bucket. The rear electrode is partially hidden by the left front electrode.



FIG. 16-5. Drop-bottom charging bucket in position over the open shell of a gantry-type Heroult electric-arc steelmaking furnace.

Therefore, most of the new large furnaces being installed are top-charge units, with many old machine-charged units being converted to top charging. It is desirable with any large top-charge furnace installation to have open-hearth type of charging-machine facilities available for handling some heavy scrap and for making other additions.

Door-Charged Furnaces—Hand charging through the door is frequently utilized for very small furnaces, and, on somewhat larger units, charging with a chute is the method used. If larger furnaces are to be charged through the rear door, a charging machine is required. The standard open-hearth charging machine (described in Chapter 15) is most frequently employed.

Auxiliary Equipment—If the plant is designed for the use of hot metal, a 75 to 100-ton, four-girder, two-trolley crane is required for handling the ladles. This can serve also as the utility crane for the charging floor. If the plant is not to use hot metal, one or two 25-ton cranes with 10-ton auxiliary hoists will be needed in the charging bay. If top-charge furnaces are utilized, cranes in the charging bay should have ample capacity for han-

dling loaded charging buckets of whatever size is necessary.

Another item requiring space on the charging floor is equipment for drying any additions to the charge. This is necessary to prevent explosions in the furnace, with resultant blowing out of slag and damage to the furnace. It is also important that the moisture, both free and combined, be eliminated in order to reduce the possibility of absorption of hydrogen by the steel. The degree of dryness varies considerably in different shops due to the methods used. Some shops run a gas pipe into a pile of material and hope to obtain results with a gas flame. Other shops use large, flat pans set on blooms and heated by gas jets under the pans. A better method is to have one or two ovens heated with gas to about 1000° F, and place the material in charging boxes in these ovens. This method, however, has two disadvantages:

1. It is difficult to obtain uniform drying with the bulk load in the box, and
2. It requires a considerable amount of space on the charging floor.

The best method for drying is a small rotary kiln which does give uniform drying and does not require a large floor area.

Proper ventilation of the charging floor is of considerable importance. A recent development has been a furnace hood. This operates on the basic theory of dust control—confining the smoke and fumes at the source before they can disperse to adjacent working areas, and exhausting them through a duct to a point of collection and disposal. The specially designed, circular hood is attached to the furnace roof ring and permits operation and maintenance of the furnace without interference.

Pouring Facilities—The foregoing remarks have concerned the charging side of the furnace building. The pouring side has similar problems in track layout, cranes, and drying.

Since a large portion of the product will be alloy steel, and therefore, hot-topped ingots will be produced, provision must be made for preparing the molds. If it is to be done in the furnace building, track and floor space must be provided for preparing the molds for at least two heats. Since this requires considerable space, it is better for the work to be done in an adjacent building as, in addition to the floor space required, preparing the molds may interfere with crane service needed for other operations. If mold preparation is done on the pouring floor in a shop with a 200,000-ton annual capacity for steel making, a 25-ton crane is required in addition to two ladle cranes.

Cranes are rated in net tons of 2000 pounds and it should be noted that the capacities are based on the load on the hooks in accordance with the Association of Iron & Steel Engineers standard specifications, which make the load on the hooks the capacity rating. Previous to this time the weight carried by the drums was often used; and the weight of the cables, load beam, and hooks had to be subtracted to get the useful lifting capacity. It is the total ladle weight lifted and not the weight of hot metal that determines lifting capacity. The ladle, with its refractory lining, may be 40 per cent of the total weight, so that for 60 tons of liquid steel, a 100-ton crane is required.

After the heat has been poured into the molds, time must be allowed for the outer portions of the ingot to solidify and form a shell of adequate thickness or, in some cases, for the ingot to become completely solid before the mold can be removed by the stripping crane. On hot-topped molds, some plants provide a stationary stripper, located in the pouring aisle, to loosen the ingots. This device is installed on the floor with a base for holding one mold plus the ingot.

Ladle Drying and Preheating—Another requirement for the pouring floor is drying and preheating facilities for ladles, stoppers, and hot tops. When ladles are relined, the entire lining must be thoroughly dried and preheated, and after each heat the nozzle and replaced brick must be dried. This is necessary for three reasons:

- (a) To prevent explosions due to generation of steam.
- (b) To reduce thermal shock to the brick.
- (c) To reduce the chilling effect of the brick on the hot metal.

The usual method for drying ladles is to place a gas line with a crude burner in the ladle; this is fairly satisfactory for drying around a new nozzle.

Another method is to lay the ladle on its side, placing the open top against a brick wall and introducing the gas burner through a hole in the wall. This gives more uniform drying results.

Stopper Drying—The latest installations for drying the stopper-rod-assembly consist of a traveling-chain

conveyor passing through a gas-heated oven. The stopper rods are suspended vertically on this chain and pass slowly through the oven. The cycle is about 24 hours and when removed at the exit end, the stoppers and rods have been thoroughly and uniformly dried.

Hot-Top Drying—In most shops, hot tops are dried with heat from an open gas torch before they are placed on the molds. This is particularly true of the refractory-lined metal-encased type which are placed on metal stands for both preparation and drying between uses. It is presently proposed to completely mechanize the cleaning, coating, and drying of these tops through use of a conveyor line and suitable auxiliary equipment, all of which will result in more uniform preparation and fully-dried tops. The initial installation of such equipment has recently been completed in an open-hearth plant. At some plants, refractory hot tops may be dried separately in ovens where thorough drying at a uniform rate can be secured.

Electrodes—Sir Humphrey Davy, in his experimental work around 1800, used the current from a storage battery for the arc, and made his electrodes from wood, charcoal, and syrup of tar, molded under 100 pounds pressure.

In 1907, Dr. Heroult imported electrodes from Sweden for use in this country, because the largest electrode made in the United States at that time was only 12 inches in diameter. Importation was both slow and expensive, so he built a plant to produce carbon electrodes up to 24 inches in diameter. This plant supplied the electrodes for the large Heroult electric-arc steelmaking furnaces installed in what is now the South Works of the United States Steel Corporation, in Chicago, Illinois.

To the steel maker, there are two kinds of electrodes. Graphite (artificial crystalline carbon) electrodes (made from petroleum coke) are required in large electric steel furnaces, particularly for the manufacture of high-alloy steels. Carbon (amorphous) electrodes are widely used in small steel furnaces and in submerged-arc furnaces for the manufacture of ferroalloys, aluminum, calcium carbide, phosphorus, et cetera. Both carbon and graphite electrodes are used in electric furnaces because of their infusibility, insolubility, chemical inertness, electrical conductivity, mechanical strength and resistance to thermal shock.

Carbon electrodes in turn can be divided into two types; one, made of calcined petroleum coke, is used in aluminum reduction; the other, based upon calcined low-ash anthracite coal, is suitable for the other uses listed. Carbon and graphite electrodes are chemically essentially the same, but differ widely in their purity and electrical and physical properties.

In manufacture, both carbon and graphite electrodes start with mixtures of raw materials properly proportioned with a suitable bonding material, such as hot pitch or tar, and extruded or molded while still hot into "green" shapes. These are cooled, packed in powdered petroleum coke in furnaces, and baked at the desired temperatures. Carbon products are usually gas-fired in kilns to approximately 2000° F. Graphite products undergo this same gas-firing, followed by repack in Acheson electric-resistance furnaces, where they are heated to temperatures above 4000° F. This second treatment in the electric-resistance furnace results in the formation of graphite (crystalline carbon) and, through volatilization, removes most of the impurities. Electrodes of both types, after the foregoing processes, are cleaned, then bored and tapped—usually on both ends.

As an electrode is consumed in operation, a threaded nipple is inserted in the top and a new electrode is

Table 16—IV. Comparison of Carbon (Amorphous) and Graphite Electrodes

Properties	Graphite Electrodes	Large Carbon Electrodes
Specific resistance (ohm-inches)	0.0004	0.0017
Specific resistance (ohm-cm.) ..	0.0010	0.0043
Weight (lb. per cu. ft.)	96	99
Apparent density (grams per c.c.)	1.52	1.57
Tensile strength (lb. per sq. in.):		
Lengthwise	500	200
Crosswise	350	150
"Practical" oxidation point	900° F	700° F
Ash	0.50%	7.0%
Size ranges:		
Diameter (inches)	1½-30	8-40
Length (inches)	24-84	60-110
Weight (pounds)	1.6-2,960	160-7,100

screwed down tightly on the exposed end of the nipple. When carbon nipples are used, a carbon paste is used to improve electrical contact. Present trends are toward utilization of graphite nipples for joining carbon as well as graphite electrodes. Carbon and graphite electrodes vary widely in physical shape, dimensions and properties. These must be selected carefully, depending upon intended usage. A general comparison of the two types is shown in Table 16—IV. Where a single value is given it may be taken as average for a variety of sizes and shapes.

Carbon electrodes cost less than one-half as much per pound as graphite electrodes. However, the field of application for the carbon electrodes is limited by their capacity for carrying electric current, one of 20-inch diameter being rated at 17,000 amperes, while the 20-inch graphite electrode is rated at 34,600 amperes. For this reason, carbon electrodes seldom are used in furnaces with transformer capacities above 3000 kva. Figure 16—6 shows the relation between the current-carrying capacities of carbon and graphite electrodes.

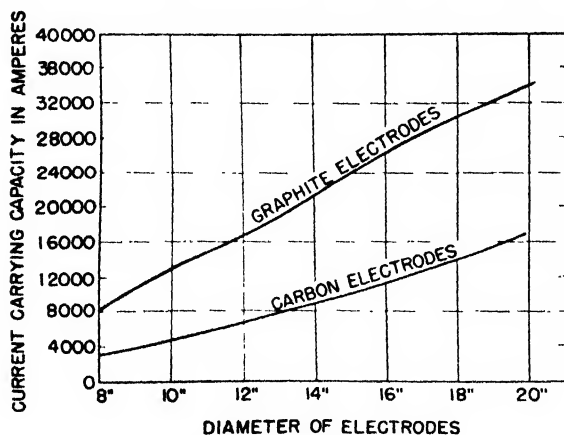


FIG. 16-6. Relationships between current-carrying capacities of graphite and carbon electrodes of different diameters, as used in modern electric-arc furnaces.

When installed on the furnace, the electrodes pass through circular openings in the roof, spaced at the corners of an equilateral triangle (see Figure 16—7). Water-cooled rings are placed over these openings and

around the electrodes to act as a seal. The electrodes are supported by water-cooled clamps at the ends of the horizontal arms that extend over the furnace from vertical standards or masts. These clamps form the electrical connection between the power supply and the electrodes, and their design is a very important detail of furnace construction. Electrode holders of the wedge type have been used for some time but now are being supplanted by so-called automatic electrode holders, usually of the spring-clamp air-release type which can be operated from furnace-platform level to adjust the electrodes (Figure 16—7).

On a typical arc furnace, the secondary leads extend in the form of bars through the transformer case. From this point, the shortest possible delta connection is made with heavy copper terminals placed generally in the transformer vault wall. The necessary secondary cables are utilized and placed in the form of a hollow rectangle to eliminate skin effect and give maximum conductivity. These cables connect to terminals on water-cooled bus tubes mounted on non-magnetic castings, remote from steel parts so as to minimize hysteresis and eddy current losses. The electrode arms, with horizontal section insulated at the rear, carry the bus tubes and electrode holders. The arms are usually in the form of an inverted-L cantilever mast. Each mast is capable of being raised or lowered independently to provide separate movement of any electrode.

The raising and lowering of the electrodes is accomplished through wire cables with motor-driven gears and winch units. It is the control of the motors driving these units that governs the successful operation of the furnace during the melting period.

Soderberg Electrodes—This is a continuous type of electrode made of a special paste that is baked by the heat of the furnace. A thin-steel cylindrical casing supported by two pieces of strip steel welded to the casing, one on either side, is filled with this paste. As the casing is lowered into the furnace, a new section of casing is welded to the top of the last section. As the bottom of the electrode is consumed in the furnace, the supporting strips are lowered and the electrodes slipped through the clamps. The heat of the conducting current plus the heat of the furnace bakes the paste on its downward travel.

This type of electrode has been built in sizes up to 27 inches by 66 inches, with a conducting capacity of 80,000 amperes. It is used on reduction furnaces where rapid movements of the electrodes are not required. Its application to arc furnaces, however, is doubtful because of the complicated design of the supporting structure plus the mechanism for quickly raising or lowering the electrodes for control of power during the melting operation.

The Power Transformer—Transformers for electric-arc furnaces are similar in design to large power transformers, except that they are designed for both delta (Δ) and wye (Y) connections on the primary side with several full-capacity taps. Because of widely fluctuating load, which at times approaches short-circuit conditions, special bracing of the windings and extra insulation between turns is required. This latter is needed to withstand the high-voltage surges set up by switching operations. In normal practice, there may be 125 switching operations per 24 hours, and the resulting surges often equal five times normal voltage. These transformers are designed to suit the furnace, and their specifications usually are written by the furnace manufacturer.

All transformers are rated in *kva capacity*. As discussed in Section 13 of Part 3 of Chapter 2, the char-

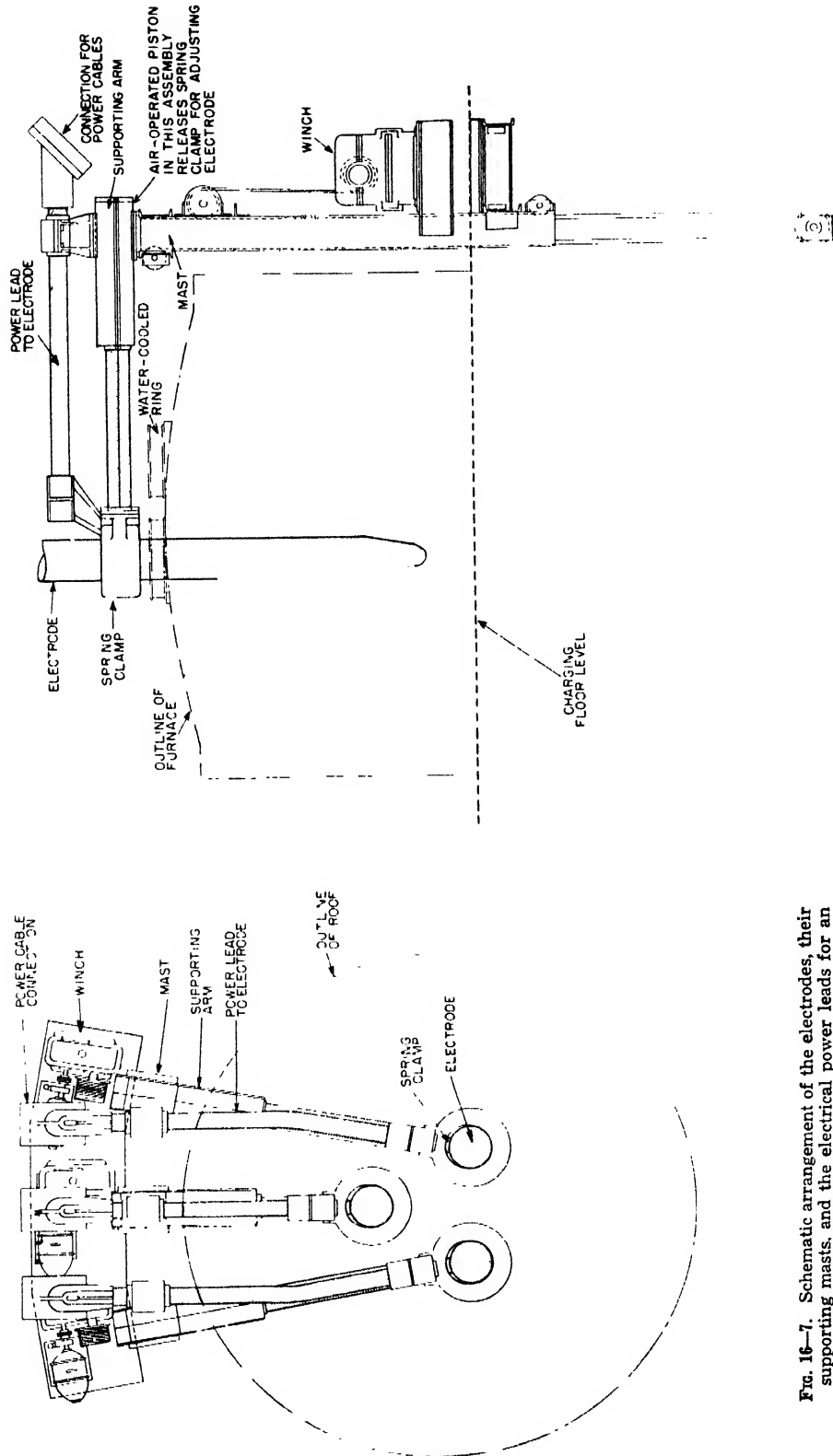


FIG. 16-7. Schematic arrangement of the electrodes, their supporting masts, and the electrical power leads for an electric-arc steelmaking furnace.

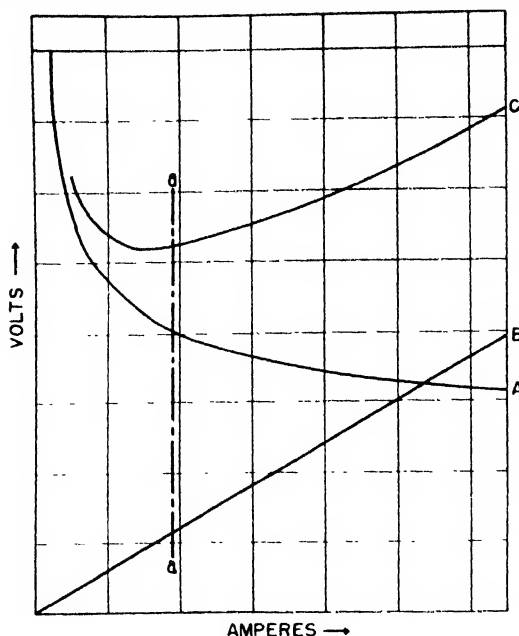


FIG. 16—8. Diagram depicting the volt-ampere characteristics of an arc with respect to its stabilizing element.

acteristics of an alternating-current circuit may result in the voltage and current being out of phase. This results in a loss of power, and the ratio of the useful power in the circuit to the apparent power supplied to it is called the power factor. The unit "kva," an abbreviation of "kilovolt-amperes," represents the product of the impressed voltage and the current in the circuit, and is a measure of the apparent power.

$$\text{kva} = \text{kilovolts} \times \text{amperes}$$

The useful power in the circuit is measured in kilowatts.

$$\text{kilowatts} = \text{kilovolts} \times \text{amperes} \times \text{power factor}$$

Kilowatts are equal to kva in the unusual case when the power factor is 1.0. The power factor is usually less than 1.0, so that the useful power (kilowatts) is usually less than the apparent power (kva).

Tap Changers—During the melt-down period in an electric-arc furnace, the delta connection on the primary winding is used as this gives higher secondary voltages. Each winding of the three-phase transformer has from four to six taps, which are terminated in a motor-driven tap changer located in a separate tank or control cubicle mounted on the side of the transformer. By changing from delta to wye connection, the voltages on the taps give secondary voltages in the proportion of 1.0 to 0.58. The mechanism of the tap changer is inter-locked with the main circuit breaker so that it is impossible to change taps under load.

Reactors—Due to the fact that, when starting to melt a heat, the resistance of the secondary circuit is very low, means must be provided to limit the current flow. This is done by adding reactance in the primary circuit. On small 60-cycle transformers (2,000 kva) the value of this external reactance may be 0.012 ohms, which value decreases to about 0.003 ohms on the 12,000-kva, 60-cycle size. On 60-cycle systems, sufficient reactance is obtained as inherent transformer reactance and reactance of leads to eliminate the use of any external reactance on transformers in excess of 7500-kva capac-

ity. On 25-cycle power supply, the limiting value is approximately 15,000 kva.

Where large transformers are used, it is important to maintain the total of the impedance of the circuit to a minimum, and secondary voltages of sufficient value must be utilized to permit introduction of the power at a desirable power factor.

Size of Transformers—The size of furnace transformer required is governed by several factors.

- (1) Whether hot metal or a 100 per cent cold charge is to be used.
- (2) Restrictions that may be imposed by the power supply.
- (3) The capability of the electrodes to carry the heating current and the ability of the charge to absorb it.

The rate at which a given charge will absorb heat is measured by the temperature gradient within the mass. This rate declines continuously as the average temperature of the charge rises and reaches zero at temperature equilibrium in the furnace chamber.

On arc furnaces up to and including 3000-kva capacity, voltage taps generally range from 250 volts, phase to phase, with transformer primary connected delta, down to approximately 90 volts with transformer primary connected wye. On larger furnaces with transformers of 20,000 kva and greater, secondary voltage taps may range from 550 volts with primary connected delta, down to 150 volts with primary connected wye.

The Secondary Circuit—An electric arc converts large quantities of electric power into heat in a space of small volume. This concentration of heat developed at a high temperature—about 6300° F (3500° C) for the carbon arc—is the initial consideration in the design of the furnace chamber. A peculiar electrical feature of the arc is that its voltage decreases with an increasing current as shown in Curve A (Figure 16-8). Hence, the arc is inherently unstable.

The condition for stability of an electric circuit with

a constant voltage is a positive volt-ampere characteristic, that is, an increase in voltage drop in the circuit simultaneous with an increase in current. To obtain this stability, something has to be added to the secondary circuit that will produce a positive volt-ampere characteristic and this something is reactance. Curve B (Figure 16—8) shows the curve for this part of the secondary circuit, and since this reactance is in series with the arc circuit, the two curves combine and form the total curve C. The dotted line a-a shows the lower limit of stability, and stable arc conditions are obtained by adjusting the length of the arc for some amount of current higher than the critical value indicated by the line a-a. In general, the circuit is stable with about 50 per cent reactance-volts drop in the arc circuit. This does not cause an energy loss, but does lower the power factor to about 87 per cent.

The arc-furnace circuit, when operated at a constant given voltage, supplies a constant power factor load during the phase of operations when that selected voltage is used. However, the reactance of the secondary circuit is what determines the value of the power factor.

The secondary circuit thus consists of a fixed reactance of supply lines, cable, transformers, bus and furnace leads, and a variable resistance consisting of the fixed resistance of these same circuit elements plus the variable resistance of the arc itself. The voltage applied to the circuit may be varied by adjustment of the transformer taps. The power at any given voltage is varied by changing the resistance of the arc. This may be done by raising or lowering the electrodes; this is accomplished by an automatic control that seeks, by positioning the electrodes, to find and maintain constantly the current value that has been selected by the furnace operator.

If the kilowatt and the kilovolt-ampere inputs are plotted for any given voltage with variations in the current input, the curves will be similar to those shown

on Figure 16—9. As the current increases, the kilovolt-ampere input increases in a straight line, since it is the product of kilovolts and amperes. The kilowatt input, however, increases only until a current is reached that will produce a power factor of 0.707 in the circuit. Further increases in current beyond this point cause a decrease in kilowatt input with increased kilovolt-ampere input and lower power factor.

Since a part of the energy input is dissipated in I²R (heat) losses in the transformer bus and leads, the actual kilowatt input to the furnace itself will be less than that shown on curve A (Figure 16—9) and will follow curve B, reaching a maximum at some circuit power factor higher than 0.707, depending upon the resistance of the other circuit elements. Since the losses to these other elements of the circuit vary as the square of the current, the result is a falling efficiency curve as shown (Figure 16—9).

It is perhaps unfortunate that current provides the most convenient factor for automatic control of the circuit. Furnace operators may thus be misled into associating heat input with amperes rather than kilowatts, and in their attempts to get high heat input into a furnace, use current settings beyond those that give the optimum kilowatt input, as defined above. Such practice results not only in high kilovolt-ampere demand and lower power factor, but also in the actual loss of steel production and increased incidental costs.

It is the custom in modern shops to determine this maximum current for each voltage setting and give this information to the melter. The following table has been worked out for a three-phase, 10,000-kva furnace transformer:

Volts	200	250	300
Optimum current (amperes)...	12,910	16,130	19,300
Maximum power (kw.)	3000	4700	6765
Approximate power factor (per cent)	74	74	74

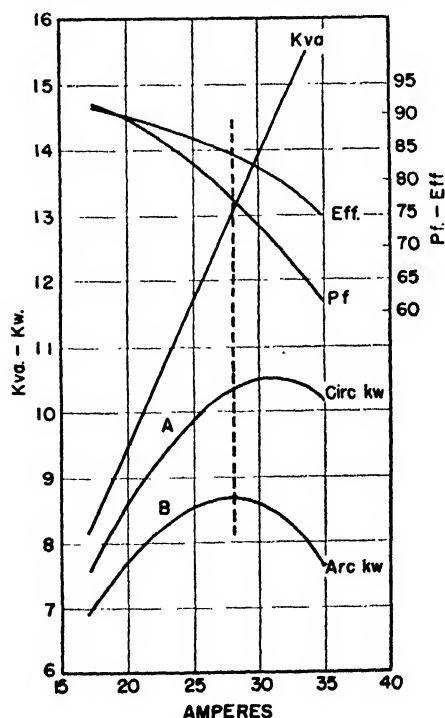


FIG. 16—9. Curves illustrating electrical characteristics of the electric arc-furnace circuit.

The Primary Power Supply with Circuit Breaker—

The type and size of circuit breaker to use for the primary circuit on large furnace transformers is a matter of controversy. Since the power supply is usually a high-capacity utility system, the required rupturing capacity of the breaker is often 1,000,000 kva. In actual operation, the breaker is opened each time a transformer tap is changed. There may be 100 such operations per day, and these high-rupturing-capacity breakers are not designed mechanically for such service. In addition, this type of breaker is designed primarily to interrupt heavy currents. Its operation is partially dependent on the magnetic field created by these heavy currents to quench the arc. When operating with only the magnetizing current of the transformer to break, this effect is not present and there is a tendency to burn the contacts. As a result of these factors, it is necessary to make both mechanical and electrical repairs to the breakers with each furnace relining.

There are several methods used to minimize circuit-breaker maintenance:

- (1) During the melt-down period there is in effect a series of short circuits on the transformer secondary circuit. If a breaker with a capacity to open the most severe of these short circuits is selected, the 1,000,000-kva rupturing-capacity size is not required, and a breaker with a lighter operating mechanism can be installed and the maintenance reduced. This breaker, however, must be inter-locked so that it will not function if there is a short circuit or ground within the transformer, and a back-up breaker must be in-

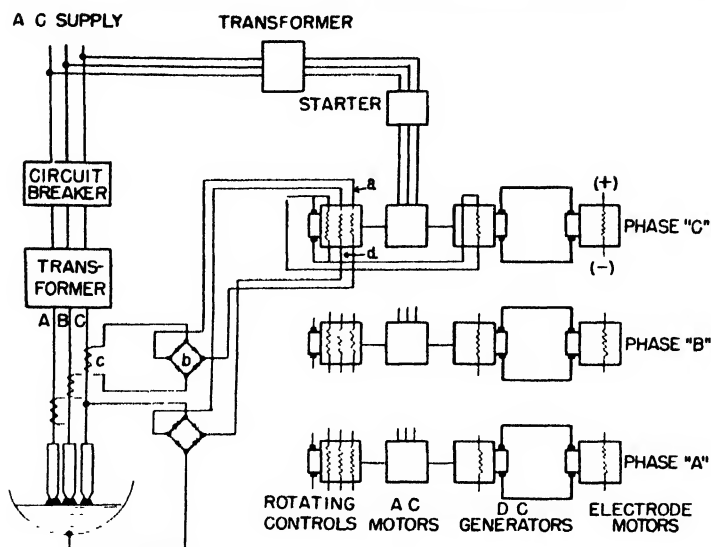


FIG. 16—10. Schematic elementary diagram of one type of regulating system for electric-arc furnaces. (Courtesy, Westinghouse Electric Corporation.)

stalled ahead of the furnace-shop power supply, set to operate under these conditions.

- (2) Compressed-air-operated breakers can be used for this work. The air required for their operation is supplied from a small compressor. This breaker has light moving parts and is designed for frequent operation. It has been very successful on installations up to 22 kilovolts and can be specified for the full rupturing capacity required by the power system.
- (3) Another type of switch is an air-operated disconnecting unit with sufficient interrupting capacity to open under electrode short circuits. A back-up breaker is required with this type of switch.

Control of Power Supply to the Charge—The early arc-furnace regulator was a simple current-operated device. The current transformers in each phase of the secondary circuits had secondaries which were connected to the regulating coils, and the pull of this current in the coil was balanced against a spring. Contacts on the plunger made connections with either the up or down set of contacts when the electrode current was higher or lower than the current setting. These contacts were relayed to the control of the electrode motors and caused them to operate in the proper direction as the current varied from some pre-determined value.

This control was unsatisfactory because the restraining force of the spring was constant. The variations of current during melt down are rapid, and the mechanical movement of the electrodes comparatively slow, so that this control responded to the average value of current variation. Furthermore, a three-phase system of currents in an electric furnace is very complex and the values of the currents in the regulator are interdependent, the value in any phase at a given time being determined by conditions in the other two phases, as well as by the length of the arc in that particular phase.

It is well known that the voltage drop across the arc will increase as the current decreases, and vice versa. Hence, if the varying pull of a coil or other regulating device, which is proportional to the current in the electrode, is matched against a restraining force that is pro-

portional to the voltage drop across the arc, the utmost sensitivity to changes in arc length can be obtained in the regulator.

The above is the principle used for the constant-potential, magnetic control systems used on most arc furnaces and, when combined with dynamic braking on the electrode-motor circuit, has been the standard method of control. This is known as the balanced-beam control.

Quite independently of the arc-furnace industry, new methods of regulation using rotating equipment with variable-voltage control were developed. When applied to electrode control, the several systems based on this principle balance currents in the electrode circuits against the voltage across the arcs and combines these with variable-voltage controls on the electrode motors.

The advantages claimed for these methods are as follows:

1. Variable-voltage control of the electrode motors results in motor torques directly proportional to the average unbalance between the current in the electrode and voltage across the arc, and gives an increased speed of response to these motors, resulting in smooth acceleration and deceleration.
2. When maximum unbalance occurs, the maximum permissible voltage can be applied to the motors causing high-speed withdrawal of the electrodes.

Figure 16—10 shows an elementary wiring diagram of one of these systems. The current-control field of the regulating element (a) is energized by a dry-type rectifier (b) which in turn is connected to a current transformer (c) in the electrode circuit. The potential-control field (d) is energized in a similar manner from the voltage drop between the electrode and the shell of the furnace.

When the breaker is closed, voltage is applied to the potential control field which causes the generator voltage to build up in the direction to lower the electrodes. When the first electrode strikes the metal charge, the voltage drops to zero causing the electrode motor to stop. When a second electrode strikes the metal, a current will flow causing the current-control field to become energized, which

acts to raise the electrode and establish an arc. As the arc is lengthened, its voltage increases and the current decreases until a balance is established between the potential and current-control fields.

Provision is made for manually controlling the electrodes individually, or automatically in a group. The operator sets the voltage tap on the furnace transformer and adjusts current input by a rheostat to the value desired.

3. Installations with this type of control, having smaller load fluctuations because of the increased speed of response, decrease electrode breakage and result in a decrease of kwh.-consumption per ton of ingots.
4. It is possible to build this control with the three generators driven by a single motor with a small flywheel, assembled on a common bed plate. Should the alternating-current power fail, there is enough momentum in the flywheel to keep the set spinning, and maintain voltage in the direct-current generators for a sufficient time to raise the electrodes out of the bath.

Operator's Control Panel—With each electric-arc furnace, it is customary to mount a control panel, usually in the wall of the transformer vault, with the instrument side of the panel flush with the outside wall. This panel contains all the instruments, rheostats and control switches for the operation of the furnace, with the exception of the control for the tilting mechanism. This master control station usually is located near the charging side of the furnace.

One of the control switches, with a red and a green indicating lamp on either side, closes and opens the main circuit breaker. Another operates the tap changer, but is so interlocked that it cannot be operated unless the main circuit breaker is open. If rotating-type control is used, the starting switch for the set, plus three switches to manually control each electrode motor, are needed.

There are three ammeters, one for each phase, and above each a rheostat. The value of the current that is to be maintained in each phase by the automatic control is set by the individual rheostats.

Above the ammeters is a wattmeter and a power-factor meter, which indicate the useful power being taken into the furnace and the power factor. There is some question of the accuracy of the power-factor reading, because of the complexity of the varying currents in all three phases, but it is a guide point and is required on all boards.

An integrating watthour meter, which adds up the kilowatt hours on the furnace, completes the normal complement of meters. Some operators install a graphic watthour meter to keep a record of the power changes and consumption at various times during the heat.

Power Requirements—Power requirements for actual melting are variable and depend upon the definition of when the charge is melted. Generally, it is assumed that the charge is melted when no further scrap remains and the metal has reached a temperature of 2850° F. Depending upon thickness of the lining and other variables that affect furnace efficiency, the kilowatt-hours required for actual melting vary from between 400 and 425 per ton.

The problem of determining the amount of power to be used during the meltdown period has to be based on several factors incident to the furnace installation and type of charge, but the maximum useful power that can be used is determined by the characteristics of the secondary circuit.

The arc itself is virtually a resistance drop, which is in series with a large reactance drop caused by the leads and transformer winding. The resistance drop is in phase with the secondary current, while the reactance drop is 90 degrees out of phase. The resultant relation is the vector sum of these two drops, and the cosine of the angle between the resultant current and the voltage is called the power factor. The useful power or kilowatts used to melt equals the product of the voltage drop times the resulting current times the power factor. When the ampere loading reaches a point where the power factor is 0.707, maximum power is going into the charge. This power factor (0.707) is that of the secondary circuit. Since the current and potential transformers that operate the power-factor meter are installed on the primary circuit of the power transformer, there is an additional reactive drop in these windings; so that maximum ampere input to the furnace occurs at from 0.75 to 0.80 power factor as read on the meter.

The maximum useful amperes for a given secondary voltage can be found by increasing the current until the kilowatt-meter reading reaches a maximum and then gives decreasing readings. The amperes at this value are the maximum to use, as any further increase not only will reduce the melting power but also heat up the leads and the transformer.

Following the meltdown period, the power requirements are reduced while the heat is worked and refined. This period depends on the composition of the steel, quality of scrap, furnace condition, and shop practice.

The rapidity of melting is a function of the amount of energy introduced in a given time. Theoretically, there is no limit to the amount of power that can be introduced to the furnace. However, if the energy introduced to the furnace is not absorbed by the charge, high gradient temperatures occur which will cause deterioration of the side walls and the roof. Thus, rapidity of melting depends entirely upon how rapidly energy can be introduced to the charge. As the energy introduced is largely in the form of radiation, it follows that light scrap with greater surface area can usually be melted more rapidly than heavy scrap. It is important, therefore, that the rate of power input does not exceed that rate of heat absorption by the charge. It is always advisable to utilize the minimum secondary voltage necessary to introduce the required amount of power at the proper power factor. On this basis, the arc length is at a minimum and damage to refractories is lessened.

Some melters start a meltdown with a lower voltage and amperage than they use after the melt has started. In a published record of a 50-ton furnace having the following transformer taps: 282, 242, 220, 199, 165 and 115 volts, good furnace results were obtained with these values:

	Volts	Amperes	Minutes
Start	199	25,000	15
	282	32,000	130
	242	25,000	15

In a shop using a 90,000-pound charge of alloy steel, the average kwh. per heat was 24,600 or 547 kwh. per ton for 200 heats. If 450 kwh. per ton were required for melting, then 97 were required for refining. The average time from tap to tap was 6 hours, 8 minutes, of which 37 minutes were required for charging. The average input during the meltdown period was 12,300 kva.

In connection with power input, the maximum input to the furnace is measured by two factors:

1. The rate at which a given charge will absorb the heat is a function of the temperature gradient

within the mass. This rate declines continuously as the average temperature of the charge rises and reaches zero at temperature equilibrium in the furnace chamber.

Inasmuch as the energy developed at the arc is largely radiant energy, the rate of heat absorption by the charge is largely dependent upon the area exposed to the radiation; hence, with large pieces of heavy scrap, the rate of heat absorption is considerably less than that which is afforded by the use of very light scrap with large area.

2. On small furnaces, the electrical circuit, including the reactor, is adjusted so as to give an average primary power factor during meltdown of approximately 87 per cent at full load. There are cases, however, where the companies supplying power object to the maximum swings in demand and insist on additional reactance in the primary circuit to

dampen these swings and reduce the power factor to 80 per cent or 82 per cent.

In Figure 16—9, the curve of power input plotted against current indicates that a relatively wide variation in current can occur without an appreciable change in useful power input. Thus, very little is gained in power input if a furnace is operated with a secondary power factor of less than 75 per cent. This means that the power factor measured on the primary side of the furnace transformer would normally be close to 80 per cent.

As a rule of thumb, it can be said that it is not economical to operate an arc furnace with a primary-side power factor under 80 per cent. It should be noted that the 0.707 power factor mentioned earlier as providing maximum power input to the charge is measured on the secondary side of the transformer.

SECTION 3

THE BASIC ELECTRIC-ARC FURNACE PROCESS

Before the location for an electric-furnace plant for the production of ingots by the basic process can be selected, its capacity and probable growth must be known, as an ample electric-power supply must be available. If it is assumed that the plant will have an initial annual capacity of 200,000 tons of ingots per year and a future capacity of 400,000 tons per year, a power supply of 50,000 to 60,000 kilowatts must be available.

This amount of power is not available in the generating stations of the steel plant and due to the highly fluctuating loads, poor power factor and load factor, it is not desirable on some utility systems. For these reasons, a source of power must be found before deciding on a plant location.

Installations have been made as additions to existing open-hearth shops. These involved low investment charges, as the electric furnace was installed either on the location of a dismantled open hearth or in an addition to the building. Hence, existing cranes, stock facilities, utilities, etc., could be used for the new installation.

Some of the objections to this type of layout are:

- (1) The chance of carbon pick up from the hot metal mixer (from flakes of graphite or kish floating in the air).
- (2) Difficulties in scrap segregation.
- (3) Congestion on the pouring floor.

Several plants were installed as complete steelmaking units which required a full complement of buildings, cranes and tracks. This arrangement is more costly but allows for segregation of materials, setting up a working force who are concerned only with making electric-furnace steels, and supervision is concentrated on the products.

Stocking and Charging Facilities—A most modern design for a basic-process electric-furnace plant calls for a lower level stockhouse located as an extension to the end of the furnace building or adjacent and parallel to the furnace building. At either of these two locations, drop-bottom charging buckets, ranging in capacity from 100 to 3300 cubic feet as required by furnace size, are loaded at ground level and subsequently moved by an overhead crane in the first instance or by transfer car and overhead crane in the second instance to the furnace floor, from which point they may be charged into the furnaces. Some older plants originally equipped with door-charged electric furnaces have two-level

stockhouses, the higher level being for the charging cars which can be loaded on the furnace level and the lower level used for storage bins.

In electric-furnace practice, the different lots of alloy scrap are kept in separate bins and, during normal times, between twenty and forty bins are required. When the use of virgin alloys is restricted, scrap must be segregated further and perhaps 65 or more kinds of scrap must be used, depending upon the grade of steel being made, as discussed later under "Utilization of Steel Scrap." Few stockhouses are designed for this practice and this may result in stocking out of doors.

Since the charge should be dry before it is placed in the furnace, a covered stockyard is advantageous. For a 200,000-ton plant, at least two 10-ton cranes equipped with lifting magnets are required in the stock yard. In those plants where the boxes are loaded on the lower level, they are placed on platens which are raised to the charging cars on the furnace floor. A 30-ton crane is required for this service. This layout calls for the loaded boxes to be carried to the scales and weighed before placing on the platen, which holds three or four boxes. This system has the advantage of easy, accurate weighing and conserves space in the stockyard. With this system of stocking, a study of crane movements with timing cycles is required as the operation can be bottlenecked by the cranes.

Tracks for both the incoming supplies and the charging cars are important factors in the plant layout. The daily supply of scrap for a 200,000-ton annual capacity shop will require a minimum of 11 railroad cars, in addition to those required for limestone, ore, brick, etc. In addition, storage tracks, located nearby, are required for these materials to provide for possible irregularity of railroad movements.

UTILIZATION OF STEEL SCRAP

Scrap Segregation—The necessity for conserving the valuable alloy content of steel scrap, to economize in the use of virgin alloys and to insure that only the elements desired are introduced in making the steel, makes it absolutely necessary to segregate or separate the available scrap into stock piles of identified grades. When the "product mix" (grades of steel produced) varies a great deal, the scrap classification by alloy content must be much more extensive than for a specialty plant producing the same grade continuously. One plant, making

various grades of steel by the basic process, including alloy and stainless steels, has found it necessary to segregate their scrap stock into 65 classifications.

The scrap stock may be revert or home scrap from the rolling mills and forge shops of the same plant where the electric furnace operates; it may be obtained from scrap dealers, from customers, from other steel producers, or from sister plants. To produce economically the wide range of steels common to modern practice requires careful selection of the scrap, and a scrap segregation and control plan is essential.

The term "segregation" as used here may be defined as the separation of a mass of mixed scrap into piles of individual compositions. Close adherence to a definite scrap-segregation program is essential if the greatest benefits are to be obtained with respect to alloy conservation and melting close to the desired composition. Consistent melting practice is an aid in maintaining mill schedules for the production and delivery of steel, and helps to obtain optimum tonnage from a given unit as well as to maintain steel quality at the high standards demanded by industry. Any even temporary disregard of the scrap-segregation program will lead to loss of alloys, and non-oxidizable elements such as nickel and copper may enter and remain in the bath and cause the heat to be scrapped or, at best, diverted to another order which originally did not require the use of so many valuable elements. For example, chromium can be oxidized from a heat ordered as nickel-molybdenum steel but it is a costly and wasteful process. On the other hand, nickel cannot be oxidized from a chromium-molybdenum heat; therefore, if it is present and the order restricts the nickel content to no nickel or very low limits, the heat must be diverted or scrapped.

Methods for Insuring Proper Segregation of Scrap—The segregation of home scrap is comparatively easy but the introduction of outside or foreign scrap imposes a problem. Several methods are available for testing the scrap to determine if it is what it is supposed to be. These include chemical analyses of selected samples, spectrographic analysis, and less costly and less accurate methods such as magnetic tests, to separate magnetic from non-magnetic scrap, and the "spark" test. The latter is made by holding a piece of scrap against a grinding wheel and observing the shower of sparks; it is possible for a trained observer to differentiate between the various kinds of steel scrap by noting the color of the spark, the length of the spark lines, and the characteristics of the ends of these lines. In addition, a couple of rough tests may be mentioned; for example, "spot tests" which permit rough estimates to be made of the amount of nickel or one of several other elements present by the use of a chemical solution or solutions applied to a clean surface of the sample to be tested. Another rough test is the "pellet" test by which a few elements may be detected by viewing oxidized particles from "spark" tests under a magnifying glass.

Physical Requirements of Scrap—The size of the scrap or its bulk density is of importance. The size may vary from ingots of various sizes to turnings from machine shops. Heavy scrap (ingots, ingot butts, crop ends) has considerable weight per unit volume, while the weight per unit volume for light scrap is low. Therefore, if there is too much light scrap in the charge for a given heat, the total volume of heavy and light scrap will exceed the volume of the furnace and part of the scrap cannot be charged until a portion of the charge is melted down. A charge made up entirely of heavy scrap is also objectionable, because it does not permit the shielding of roof and walls during the melt-down period to the same extent as a mixed charge of greater volume, and

results in decrease of refractory life. Another physical requirement, especially with heavy scrap, is that the pieces are not too long, so that the charge in the furnace may be kept relatively dense and not opened up by longer pieces which, as a rule, do not lie in a horizontal plane. If the scrap charge is made up of approximately 40 per cent heavy, 40 per cent medium, and 20 per cent light scrap, a density usually is obtained which reduces the number of "back charges" (the charging of additional scrap after the original scrap charge has melted down).

Selection of the Scrap Charge—With an efficient scrap-segregation plan in effect and with scrap available that has the proper physical characteristics, the charge should be made up as follows:

1. Grades of scrap must be selected which together contain the elements necessary to make the heat ordered. The charge may contain all or part of the specified elements, with care being taken to see that no scrap is included that contains elements which are not a part of the specification. This is imperative if the scrap contains an element which cannot be oxidized by regular practice.

2. For economical operation, each element contained in the scrap selected should approach by weight, as closely as possible, the number of pounds of that element required to meet the lower range of the chemical specifications (for that element) in the ordered heat.

3. Knowing the amount of the various elements contained in the combination of scrap selected for a heat, the total amount of virgin alloys necessary to make the heat are calculated. This calculation is based on the weight of the heat to be made (furnace capacity and amount specified by an order) and the specification. In making these calculations, allowance must be made for adjustments during the making of the heat; for example, losses in melting some alloys and the possible absorption of certain alloying elements from the bottom and banks of the furnace. In regular practice, a heat made to a low alloy specification occasionally finishes outside of the specified composition ranges when the heat is made immediately following a high alloy specification, such as a stainless grade of high-manganese steel, etc. The usual practice to overcome this difficulty is to make a so-called wash heat; i.e., a heat of medium alloy specifications, but containing the same elements, made following the high-alloy grades. The purpose of the "wash heats" is to have any absorption of elements from the furnace bottom and banks take place in a heat in which the increase in certain elements is not harmful and thus prevent high residuals in the low-alloy heats which will follow.

4. Efficient operation makes it necessary that the scrap selected for a given charge is not only satisfactory for that charge but that it be selected on the basis of the scrap available for efficient operation over a period of time. If only heavy or medium scrap were used for a few heats, it is probable that these particular heats would be speeded up. If, however, at the same time a large inventory of light scrap (turnings, punchings, etc.) were accumulating, excessive amounts of this light scrap would have to be charged in later heats, probably with back-charging, and the delays and damage to refractories would far off-set any gains in the few heats made with heavy and medium scrap.

5. It is probable that no part of the routine of charging the furnace is of more importance than the loading and weighing procedures. Unless the proper type of scrap is selected and correctly weighed, the final product probably will be of the wrong composition, because

all adjustments are based on what is assumed to be the correct weight of the scrap charged originally.

CHARGING, MELTING AND FINISHING

Charging the Furnace—The various types of charging equipment have been described earlier in this chapter. Regardless of the type of furnace, whether door or top-charged by hand, chute, bucket, or charging machine, an effort should be made to place the charge as accurately as possible.

The power is turned off and the roof and electrodes are moved out of the way in the case of top-charged furnaces. With door-charged furnaces, the electrodes are raised as high as possible to prevent breakage during charging, and the charging started. Light or medium scrap is charged in a thin layer on the bottom or hearth, because this type of scrap melts faster than large pieces when the metal directly under the electrodes melts and drips down through the charge and collects on the bottom of the furnace. Heavy scrap is charged in the area within or adjacent to the triangle or "delta" formed by the electrodes and must be charged in such a way that it will not shift during melting down and cause possible breakage of electrodes by falling against them. After this, light or medium scrap is usually piled high around the sides of the furnace to protect the roof and side walls from the arc during the melt-down at high power input.

Ferro-alloys, alloying oxides and virgin alloys which are not easily oxidized, can be and usually are charged in the furnace prior to melting down.

It is desirable to melt down with excess carbon in the bath in order that some carbon may be worked out by ore additions. If the metallic charge is too low in carbon, a recarburizer in the form of coke or scrap electrodes is charged with the scrap to allow for a carbon content of the bath at melt-down that will be 0.15 to 0.25 per cent higher than the carbon content of the finished steel.

Ore is added to lower the carbon content. Various operators may charge the ore with the scrap, or when the charge is partially melted, or when the charge is completely melted. The control of the boiling action caused by the addition of ore, cinder and roll scale to eliminate the excess carbon is one of the most important factors in the production of high-quality steel. The methods of charging the ore may be adapted to meet the conditions existing with a particular charge to give the best results in making the specified grade of steel.

Any general description of making steel in the basic electric-arc furnace must be broad enough to include double- and single-slag methods with other variations.

Considering as the basis for discussion the double-slag, cold charge practice, because it is relatively the most complicated, it can be divided into: (1) the melt-down and/or oxidizing period, (2) the slag-off period, (3) the reducing period, and (4) the tapping period.

Melt-Down and/or Oxidizing Period—When charging has been completed, the bank in front of the charging door is built up with refractory material (dolomite) to form a breast or dam to keep the molten metal from slopping out the door (furnaces may have more than one door if the size warrants). The door (or doors) is closed and the electrodes are lowered to about an inch above the scrap. The main circuit breaker is closed, an intermediate voltage is selected with proper current setting on the rheostats, and the arcs are struck under automatic control. After approximately 15 minutes (to allow the electrodes to bore into the scrap), maximum

voltage and current should be applied for the fastest possible melting of the scrap. The initial slow start is to shield the lining and roof from the heat of the arc, which lengthens as the voltage is increased.

The melting period in the basic electric furnace is the most expensive period in its operation because power and electrode consumption are at the highest rate during this period.

The electrodes melt the portion of the charge directly underneath them, and continue to bore through the metallic charge, forming a pool of molten metal on the hearth. From the time the electrodes bore through the scrap and form the pool of molten metal on the hearth, the charge is melted from the bottom up by radiation from the pool, by heat from the arc, and by the resistance offered to the current by the scrap. This continues until the charge is entirely melted.

From the time molten metal begins to form until the entire charge is in solution, oxidation occurs in varying degrees. During this period phosphorus, silicon, manganese, carbon, etc., are oxidized, the reactions being somewhat as follows:

1. (a) $2 \text{Fe} \cdot \text{P} + 8 \text{FeO} = 3 \text{FeO} \cdot \text{P}_2\text{O}_5 + 11 \text{Fe}$
 (b) $3 \text{FeO} \cdot \text{P}_2\text{O}_5 + 3 \text{CaO} = 3 \text{CaO} \cdot \text{P}_2\text{O}_5 + 3 \text{FeO}$
2. $\text{Si} + 2 \text{FeO} = \text{SiO}_2 + 2 \text{Fe}$
3. $\text{Mn} + \text{FeO} = \text{MnO} + \text{Fe}$
4. $\text{C} + \text{FeO} = \text{CO} + \text{Fe}$

Oxygen for these, as well as other oxidizing reactions, is obtained from (1) the oxides on the original scrap surface, (2) free oxygen in the furnace atmosphere, (3) oxidation of the scrap during melting, (4) calcination of limestone (if used), (5) oxidizing action of the gases in the furnace, (6) oxides of alloying elements added in the furnace, (7) ore, cinder and scale (if charged or added later), and (8) oxygen gas introduced into the molten-steel bath through a pipe or lance. The direct use of oxygen gas (Item 8) is extremely important in modern practice from the standpoint of removing carbon from the bath rapidly.

Oxidation practice must be varied with different grades of steel. For example, low-carbon steels and those in which the hydrogen content must be low, require a relatively high degree of oxidation. As the oxidation progresses, the temperature of the bath is raised to promote carbon removal and to increase the fluidity of the bath to the point where inclusions may rise through the molten metal to its surface and into the slag. The reaction of FeO with C forms CO gas, and this gas generated in the bath gives rise to the boil. Maximum cleanliness of the steel demands a high degree of oxidation and a hot, active bath. Oxygen blown into the bath is of great assistance in attaining these ends.

The reactions taking place in the bath of the basic electric-arc furnace during the oxidation period are similar to those in the basic open-hearth, except that the electric-furnace bath can be made hotter (hence, there is more chance of phosphorus reversion unless the slag is strongly basic). Also, there is no continuous supply of oxygen to the bath as from the open-hearth flame.

When the charge has been melted completely under the conditions described, a preliminary sample is taken for chemical analysis and any additional materials required are added to promote further oxidation and adjust the basicity of the slag. If the oxides with acid properties (SiO_2 , P_2O_5 , Fe_2O_3 , etc.) are in excess of the oxides having basic properties, (CaO , MnO , MgO , FeO , etc.), the slag is an acid slag, while a basic slag is one having an excess of basic oxides. If the total amount of

basic oxides balances that of the acid oxides, a neutral slag is formed. The lime-silica ratios (the so-called "V" ratios) are used generally in calculating the basicity of slag. If the "V" ratio is greater than 2.0, the slag is "basic;" from 1.0 to 2.0 it is semi-basic, and if less than 1.0 it is considered acid. In the process described, the slag test should show a lime-silica ratio of from 2.2 to 2.8. Several methods are used for the above calculation, three of which are as follows:

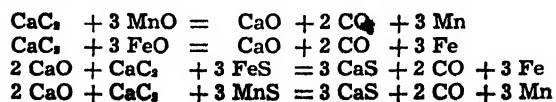
1. Lime-silica ratio = $\frac{\% \text{ CaO}}{\% \text{ SiO}_2}$
2. Corrected ratio = $\frac{\text{CaO} - 1.6 \text{ P}_2\text{O}_5}{\text{SiO}_2}$
3. Corrected ratio = $\frac{\text{CaO}}{\text{SiO}_2 + 0.634 \text{ P}_2\text{O}_5}$

Electric-furnace steel may be made by a single slag process; the slag is first oxidized as described and then made reducing by adding the proper materials during the refining period. The usual practice with the cold-melt process is to use a double-slag method, an oxidizing slag followed by a reducing slag. If desired, it is also possible to use a neutral slag in case it is necessary to maintain the bath without further refining.

Refining—In the double-slag method the original slag, with its oxidation products, is slagged off or removed from the surface of the bath by cutting off the electric power to the electrodes, raising the electrodes, back-tilting the furnace slightly, and then raking the slag out through the charging door with wooden or steel rabbles. The original slag must be removed thoroughly to prevent delay in making up the second slag and reversion of any elements from slag to metal.

The materials used in making up the second or reducing slag are burnt lime, fluorspar, and silica sand, with powdered coke to supply carbon for forming calcium carbide. Typical proportions are 5 to 8 parts of lime, $\frac{1}{2}$ to 2 parts of fluorspar, 1 to 2 parts of coke, and $\frac{1}{2}$ to 1 part of silica sand. The above amounts are premixed in proportions determined by experience. Sufficient sand and fluorspar should be used to flux the lime quickly. Coke usually is added after the slag becomes fluid. Very often, small amounts of crushed ferrosilicon also are added. In producing low-carbon grades of steel, under 0.12 per cent carbon, a lime-silica, a lime-alumina, or a modified carbide slag containing less coke, is used.

The object is to form as quickly as possible a strongly reducing slag containing calcium carbide, and maintain it through the refining period. The presence of carbide may be detected readily by the odor of acetylene generated when a slag sample is wet with water that reacts with CaC_2 to form this gas. Carbide cannot exist in the presence of oxides reducible by carbon; therefore, a carbide slag is free of such oxides. A carbide slag acts to return reducible oxides such as those of manganese, chromium, vanadium, tungsten, iron, etc., from the slag to the metal; consequently, such oxides may be added for direct reduction as soon as the carbide slag is formed. The slag also serves to reduce the oxides in the bath and facilitates the removal of sulphur as calcium sulphide. The reactions proceed according to the following chemical equations:



Desulphurization is aided by a high manganese content, by the addition of lime and fluorspar, by agitation of the bath, and by high temperature. If very low sulphur is required, sometimes two carbide slags may be necessary. The carbide slag should be completely shaped up 20 to 40 minutes after the first slag has been removed. The carbide content of the slag is kept to a relatively low figure for the lower carbon heats, and, for very low carbon heats, silicon can be substituted for coke in the slag to prevent carbon pick-up by the metal.

Lime-alumina slags are made from lime and aluminum in the form of shot and/or granules. Calcium aluminate is formed, which both desulphurizes and deoxidizes the bath. Some of the deoxidation of the bath is done by the metallic aluminum before it oxidizes and combines with CaO as calcium-aluminate.

The steel should not be held under the second slag any longer than is absolutely necessary. As soon as results of the last preliminary analyses are reported, the necessary additions are made to the bath for adjustment of the carbon and alloying element content. The additions of alloying elements must be made in quantities sufficiently small to prevent chilling of the bath. When all of the additions are in solution, the slag is again shaped up and ferrosilicon is added. Final additions of aluminum for grain-size control usually are made just before tapping. The aluminum is put into the furnace, tied on the end of long bars to insure getting it through the slag into the bath. About one-half of the aluminum usually is added to the stream of metal in the spout as the heat is being tapped into the ladle. The furnace is tapped usually from 10 to 15 minutes after the final shaping of the slag and additions of ferrosilicon and aluminum have been made.

Induction Stirring—In the last few years, there has been considerable interest in stirring of the steel bath by electric induction. The original development occurred in Sweden, and two large furnaces in the United States are now equipped with the Swedish type of inductor stirrer. At present, an installation is being made utilizing a rotating-type stirrer developed in the United States.

In the Swedish design, an inductor constructed of steel laminations, in which is imbedded a two-phase winding, is installed under the bottom of the furnace. In order to permit penetration of the magnetic flux and to minimize bottom heating, the carbon-steel bottom plates ordinarily used are replaced by austenitic stainless-steel plates, which are treated theoretically as an air gap. A special generator and exciter is provided to produce two-phase current at a frequency of approximately $\frac{1}{2}$ cycle per second. During operation, a magnetic flux is produced in the form of a moving field. This moving magnetic field, reacting with a field induced in the bath, causes the bath to flow at a low rate of speed across the bottom of the hearth. Thus, a stirring action is obtained.

With the type of stirrer developed in the United States, a two-pole electromagnetic rotor is revolved underneath the furnace by a motor-and-gear mechanism. The rotor is excited by current from either the shop supply or a special generator. This causes a movement of flux across the bottom of the furnace, the bottom plates of which are made of austenitic stainless steel. In turn, this induces a movement in the molten steel bath similar to that previously described.

It has been proved that with induction stirring the segregation of alloying elements is greatly minimized. The temperature of the bath is much more uniform. It has also been demonstrated that, due to the movement

of the steel, the interface contact between the slag and the steel is greatly improved, with less time required to obtain the necessary metallurgical reactions.

Tapping—In tapping a heat, the electrodes are raised to the maximum height after the power is shut off, the tap hole is opened and the furnace is tilted by a control mechanism so that the steel is drained from the furnace into a ladle set on the pit side of the furnace. The slag comes after the steel and serves as an insulating blanket during tapping. A clean, round tap-hole and a clean, smooth tapping spout reduce the possibility of having a ragged, easily oxidized stream when tapping into the ladle.

LADLE, MOLD AND POURING PRACTICES

Ladle Practices—After all the molten steel in the furnace has been drained into the ladle, the ladle crane moves the ladle to the pouring platform where the steel is poured into molds. Both the ladle and pouring practices are practically identical with open-hearth practice, except both the inner and outer linings of the ladles are constructed of first-quality fire brick and special precautions are taken in setting the nozzles and stoppers because of the higher tapping temperatures employed in electric-furnace practice and the closer control desired for most electric-furnace products. The steel may be re-ladled (i.e., poured from one ladle into another) before teeming to insure the proper mixing of the alloys and reduce pouring temperatures.

Mold Practice—When the steel is ready to be poured, it is teemed into carefully prepared molds. Molds of the big-end-up type are used generally for the killed electric-furnace steels, while rimmed steels are poured into the conventional big-end-down molds. Shapes and sizes of molds vary considerably, depending on the steel being produced and the size and shape of the product. Round, square, and rectangular molds with smooth or fluted inside surfaces may be used.

The molds are scraped carefully, cleaned, coated with mold coating, preheated (if necessary), and all dirt is siphoned out after the hot tops have been set—about 30 minutes to 45 minutes before the heat taps. Each shop has a variety of different mold coatings and mold practices that have been developed for the particular grades of steel being produced. For instance, steel has been poured into clean uncoated molds, as well as molds coated with tar, salt brine, lime slurry, resin smudge, acetylene smudge, molasses, lamp black, oil, graphite, aluminum paint, and various other coatings.

Pouring Practice—Either of two pouring practices may be employed, the direct or the indirect. The former practice is accomplished by raising the stopper rod and allowing the steel to run directly into the mold from the ladle. In the indirect method, the steel is allowed to run through a refractory funnel and runner (as in bottom pouring). Basket pouring, a modification of indirect pouring, utilizes a small intermediate ladle which is filled from the large ladle. When the small ladle is filled, the nozzle in its bottom is opened, allowing the steel to run into the mold or molds through one or more openings or nozzles in the bottom of the basket. The purpose of indirect pouring is to reduce the splash in the bottom of the mold by eliminating the high ferrostatic pressure in the ladle, with the view of reducing scabs and other defects on the ingot surface. The theory justifying basket pouring is based on the premise that nonmetallic inclusions in the steel rise rapidly through relatively short distances at higher temperatures; thus, in conjunction with tapping the heat hot, inclusions rise out of the steel in the lower area of the

ladle before that steel issues from the bottom to fill the basket, and also rise in the basket while the molds are being filled.

Regardless of the pouring practice, the molds are filled slowly at first until the pool of steel on the bottom of the molds is about 6 inches deep, and then a full stream is permitted. When a conventional brick hot-top is used, a full stream is maintained until the steel rises to the bottom of the hot-top, at which moment the stopper is lowered to shut off the stream to permit the steel to freeze at the junction between the mold and hot-top, after which more steel is poured to fill the hot-top. When using a hot-top which seals the junction, the stream is permitted to flow full until the hot-top has been filled.

If the junction does not seal, steel will leak out and run over the top of the mold. The blocks supporting the hot-top are knocked out immediately after the junction seals to prevent "hanger cracks" in the ingot. Burnt lime, straw, brick dust, or manufactured hot-top covering materials are used on top of the molten metal in the hot-top to hold the metal in a molten condition long enough to fill up the shrinkage cavity in the ingot.

The ingots usually remain at the pouring platform for a predetermined time, to permit solidification of the ingot, usually in the range 1 to 2 hours, depending on the size of the ingot. They are then sent to the stripper, enroute to the rolling mills or forge department.

Repairs to Furnace Bottom and Banks—After the furnace is tapped, it is tilted back to its stationary operating position, the doors opened to permit smoke and fumes to clear out, and the furnace is inspected for damaged areas on the banks, bottom, roof and around the tap hole, or any other location where repairs may be needed; also for furnace skulls. The reducing slag cuts the basic lining and it usually is necessary to make up the "cinder line" where it has been eroded by the slag. If the refractory lining requires repairs, the patching is done immediately to allow the material to be sintered into place by the heat of the furnace.

Electrode Adjustments—If the electrodes are not sufficiently long to finish the next heat, there will be a delay during the making of the heat while the necessary adjustments are being made to the electrodes. The usual practice, therefore, after making repairs to the furnace, is to run the electrodes down until they are a short distance from the bottom to enable the operator to decide how much longer the electrodes will last. Adjustments are made either by adding new sections to the tops of the present electrodes or by allowing the electrodes to slip down through the holder to the desired position as the holder is raised, provided there is sufficient electrode extending above the holder to permit this to be done.

SLAG CONTROL IN THE BASIC PROCESS

The preceding discussions on slags indicate that slag control is a very important factor in electric-furnace steel production. The electric-arc furnace permits the slags to be controlled to meet almost any desired characteristic, a fact that is the real basis of the flexibility of the arc furnace.

As pointed out, the function of the melt-down slag is to oxidize carbon and phosphorus out of the steel and to remove some sulphur and nonmetallic substances. The lime-silica ratio should be between 2.2 and 3.0. The iron-oxide content of the slag varies with the carbon in the steel at the end of the boil, and may range from 13.0 to 20.0 per cent for medium-carbon steels. A good basic, melt-down slag in the electric

furnace is similar to a good open-hearth slag. A typical composition of a melt-down slag is as follows:

Constituent	Per Cent
Lime, CaO	40.9
Silica, SiO ₂	13.4
Iron Oxide, FeO	14.8
Alumina, Al ₂ O ₃	3.5
Magnesia, MgO	8.2
Manganous Oxide, MnO	12.7
Phosphorus Pentoxide, P ₂ O ₅	0.6
Sulphur, S	0.1

Lime-silica ratio and iron oxide content can be estimated from the appearance of slag "pancakes," and a skilled operator can judge these values very closely. A "pancake" is a slag sample prepared by pouring molten slag into a small, flat iron dish possibly 4 inches in diameter and ½ inch deep. In solidifying, the "pancake" acquires visible markings characteristic of its composition.

The refining slag removes dissolved oxides from the bath, protects alloy additions from oxidation, and removes sulphur. This slag is gray to black in color, is highly carbidic, and disintegrates or slakes to powder upon cooling. A typical composition of such a chemically reducing carbidic refining slag is as follows:

Constituent	Per Cent
Lime, CaO	67.8
Silica, SiO ₂	22.4
Iron Oxide, FeO	0.5
Alumina, Al ₂ O ₃	0.5

Magnesia, MgO	6.9
Manganous Oxide, MnO	0.2
Sulphur, S	0.3
Calcium Carbide, CaC ₂	1.5
Chromium Oxide, Cr ₂ O ₃	0.4

It is essential that furnace doors and openings be closed tightly if a good reducing slag is to be maintained. If a heat is too hot, the carbidic property of a slag will be lost very quickly. The carbon pick-up from such slags is fairly high and close checking must be maintained.

With low-carbon steels (0.15 per cent carbon or less), the carbide slag is modified by using a higher lime and lower coke ratio. In making low-carbon heats of 0.08 per cent carbon or less, crushed ferrosilicon or aluminum is substituted for carbon in the slag, forming calcium silicate or calcium aluminate.

Following is a typical composition of a lime-silica slag with no carbon added:

Constituent	Per Cent
Lime, CaO	57.9
Silica, SiO ₂	27.5
Alumina, Al ₂ O ₃	3.7
Iron and Manganous Oxide, FeO and MnO ..	1.0
Magnesia, MgO	7.7
Chromium Oxide, Cr ₂ O ₃	0.3

The above slag is white and, when cooled, will disintegrate to a powder. For 0.12 per cent carbon and lower stainless-steel heats, lime-silica and lime-alumina slags are used because they are carbon-free and easy to manage.

SECTION 4

THE ACID ELECTRIC-ARC FURNACE PROCESS

The acid electric-furnace process is, as previously mentioned, employed chiefly for the production of steel for castings by the foundry industry. Four major variations of the acid process are used: (1) partial oxidation, (2) complete oxidation (with a single slag), (3) complete oxidation with silicon reduction, and (4) double-slag practice.

Partial oxidation practice is used chiefly to produce low-priced steel castings that do not require any acceptance tests other than superficial surface inspection, because it is the cheapest method of making steel for such castings. The double-slag process is employed where it is desirable to have positive control to keep the FeO content of the finishing slag to a low value (about 10 per cent). Silicon in the slag can be reduced to enter the metal in acid-electric practice, and this procedure is employed in European practice but is not generally followed in this country.

The great majority of all American steel foundries employ the complete-oxidation process, and this method will be taken as the basis for the ensuing discussion.

With the exception of selection of scrap (which must have a low phosphorus and sulphur content), the melting down of the charge in the acid electric furnace is similar to that of the basic electric furnace. As with the basic electric furnace, the electrodes melt the scrap and bore their way through to the hearth of the furnace. If the pool of molten metal formed on the bottom does not cover sufficient area to extend beyond the area covered by the electrodes, the arc will act on the sand in the hearth of the furnace. The conductivity of a non-metal increases with temperature; consequently, the furnace hearth is a fairly good conductor of electricity

if it is hot. This working on the hearth or boring a hole into the hearth by the electrodes is called "pulling bottom" and generally is indicated by the appearance of white smoke accompanied by bright yellow flames around the electrode ports. When this occurs, the electrodes should be raised out of the charge and enough clean scrap added and melted to form a pool extending out from under the electrodes. If enough heavy scrap is used and is packed compactly enough on the bottom of the furnace, there is very little danger of pulling bottom.

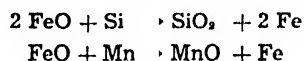
Working the Heat—As soon as the charge is melted or nearly all melted, it is time to start working the heat. A little iron ore and silica sand should be spread over the bath at this time. If, as is often the case, a high percentage of returned foundry scrap is charged, very little silica sand need be added, because the oxidation of silicon and manganese in the scrap will form almost enough slag. Some iron also will be oxidized in melting down, forming FeO which will contribute to the slag. Enough slag-making material should be added to each heat to form a layer covering the metal at least ½ of an inch thick.

Although such a thin layer of slag may cause some difficulties, it is never advisable to have more slag on the metal than necessary at any time, because this will slow down the deoxidation of the steel. If "slag" (the term often used for silica sand in acid furnace practice) is not added, the bath will take silica from the hearth.

Slag samples taken from the furnace at melt-down should have a glassy, black color, indicating a high iron-oxide (FeO) content, which is necessary if a boil is to be expected later. A sample of the metal should be

taken from the furnace at the same time and the carbon content determined. The carbon content after melt-down, as in the basic process, should be higher than the carbon desired in the finished steel after it has been killed by additions of silicon and manganese near the end of the heat. The excess carbon will be removed by the boil.

If the slag test taken from the furnace has a brown or greenish color instead of black, it is an indication that there is insufficient FeO present. Ore is then added shovel by shovel-full until a black slag results. If the charge is made up of a large percentage of returned scrap, the ore should be added before all of the scrap is melted. The FeO added in this way will be taken up by the bath and the oxygen will react with silicon and manganese in the melting metal to form silicon dioxide (SiO₂) and manganous oxide (MnO), respectively. Because these oxides are of lower specific gravity than the molten metal, they will rise to the top of the bath, and because of their chemical affinity for each other—one being an “acid” and the other a “base”—they will unite to form slag. The chemical reactions are as follows:



The silicon and manganese must be oxidized before the carbon in the molten bath can combine with more oxygen from the ore to form the “boil.” The best practice is to oxidize the silicon and manganese while the bath is comparatively cold because high temperatures favor the combination of carbon and oxygen.

After the bath is covered with a black or oxidizing slag and the carbon content is high enough, the temperature should be increased until the steel is hot enough to boil. The “boil” is the reaction between the carbon and oxygen dissolved in the steel, and is necessary in the manufacture of clean, high-quality steel. These elements combine, when a certain temperature is reached, to form carbon monoxide, a gas:



Enough ore and carbon should have been in the bath, either naturally or by additions, to maintain the boil for at least ten minutes. A sample of the metal is taken from the bath and the carbon determined by a fracture test or by some rapid analytical test.

The intensity of the boil diminishes gradually as the oxygen in the metal is reduced. During this period, the slag is changing color continually and when it reaches a pea-green stage and has thickened up enough for a spoon, stirred through the slag, to pick up small lumps of slag, the steel has been deoxidized as much as possible by the carbon reaction. A carbon test is then taken and the carbon adjusted to the desired content. Silicon and manganese should then be added as ferroalloys for de-oxidation and the heat tapped soon after they are melted completely and diffused through the bath.

The temperature at which the steel is tapped from the furnace depends largely on the size of the castings to be poured and the equipment for handling the molten metal. If poured into many small castings the steel must be very hot (possibly 3200° F by optical pyrometer), while it may be from 150° to 200° colder when tapped from the furnace if the steel is poured directly from a large ladle into a mold for a large casting.

The preceding discussion refers specifically to production of plain or carbon steel for casting. Alloy cast steels are coming more and more into use, and present a new problem. Fortunately, three of the alloys commonly used in steel-foundry practice, copper, nickel and molybdenum, can be added at any time without loss due to oxidation and subsequent absorption by the slag during the steelmaking process. If these alloys are not added with the cold charge, they should be added from 15 to 30 minutes before the heat is tapped to give ample time for their solution and uniform distribution throughout the bath of molten metal.

When the steel ordered calls for manganese in excess of 1.25 per cent, it is very difficult to hold it in the metal under an acid slag. It is, therefore, advisable at times to add lime to the slag before tapping, thus decreasing its acidity and ability to absorb manganese.

Chromium presents a problem, as it is easily oxidized. It usually is added to the bath after the steel has been deoxidized (just after the final silicon addition). It usually is added as ferrochromium containing about 68 per cent chromium, 6 per cent carbon, and the balance iron.

Alloys such as aluminum, titanium, zirconium, vanadium and boron are added in the ladle. The common practice is to add these alloys in paper sacks, thrown into the ladle as the steel is being tapped so that the sack hits the metal stream.

SECTION 5

INDUCTION ELECTRIC-FURNACE PROCESSES

In most steel plants using induction furnaces, the melting procedure is essentially a crucible or “dead-melt” process. The charge is selected carefully to produce the composition desired in the finished steel with a minimum of further additions except, possibly, small amounts of ferroalloys as final deoxidizers.

The charge may consist of a single lump of metal, a number of small pieces of selected steel scrap, or even turnings or other light scrap with which is mixed a moderate amount of larger pieces to provide initial conditions which are favorable to the generation of heat. The charge is collected in pans placed on the working platform and either is dropped into the furnace through the top opening by tipping the pan, or by raking the pieces out of the pan and into the furnace. If the charge consists of pieces of scrap of varying size, the larger pieces are charged first, and the smaller pieces are packed about them as closely as possible. Even with the closest packing it is sometimes necessary to add some of

the charge as the melting progresses. As a rule, no re-fining is attempted in acid-lined furnaces, and it is seldom tried in a basic-lined furnace. If the furnace is equipped with a tight cover over the crucible, very little oxidation occurs during melting. Such a cover also serves to prevent cooling by radiation of heat from the surface of the molten metal. Hence, from the standpoint of heat loss, the use of a slag covering to protect the metal is unnecessary. Slags are being used successfully both during melt-down and refining, in special cases.

Melting the Charge—As soon as the furnace is charged, the switches admitting the primary high-frequency current to the coil of the furnace are closed. Immediately, the rapidly changing magnetic field at high flux density generates heavy secondary currents in the charge which are converted into heat by the electrical resistance of the charge itself. This heat is developed mainly in the outer rim of the metal in the charge, but is carried quickly to the center by conduc-

tion. Soon a pool of molten metal forms in the bottom, causing the charge to sink and at this point, any of the charge remaining is added. The current exerts a strong motor effect upon the liquid metal, which accelerates the melting by washing the still-solid part of the charge with molten metal, thoroughly mixing the metal as it is melted. This motion of the metal continues after all the charge is melted, in eddies giving the bath a convex surface, and varies in intensity with the power input. As the convex surface is not a favorable condition for slag treatment, the power input always is decreased to flatten the convexity and reduce the circulation rate when refining under either an oxidizing or a reducing slag. The flow of the liquid metal, even though less vigorous, then becomes a favorable condition, accelerating the purification reactions by constantly bringing new metal into close contact with the slag. When no purifying is attempted, the chief metallurgical advantages of the process are attributable to the stirring action, which promotes the uniformity of the product, the control over the super-heat or temperature above that of the melting point, and the opportunity afforded by the conditions of the melt to control deoxidation through proper additions. As soon as the charge has "melted clear," and refining actions have ceased, any objectionable slag is skimmed off, and the deoxidizers or other necessary alloying elements are added. When these additions have melted and become diffused in the bath, the power input may be increased to bring the temperature of the metal up to the point most desirable for pouring. The current then is turned off and the furnace is tilted for pouring, either directly into ingot molds or into a ladle, the ladle being used with the larger furnaces. As soon as pouring has ceased, any slag adhering to the wall of the crucible is scraped out and the furnace is righted for charging again.

Advantages of Induction Melting—Induction furnaces are relatively low in cost, as compared to other types of melting units and, as a result, several furnaces for operation from a single frequency changer can be installed at little extra expense, and furnaces of various capacities can be used as required or individual furnaces can be retained for making melts of special alloys without danger of contamination or the necessity of making "wash heats." Among other advantages, there is very little heat radiated from the furnaces as they are water cooled, and there is practically no noise attending their operation.

This type of furnace requires about 650 kwh. of input power per ton of molten metal and heats are melted in about an hour to an hour and fifteen minutes.

For the melting of high-alloy chromium steels, there is quite an advantage in using the induction type of furnace because the highest temperature to which the scrap is subjected is that of the bath. In the arc furnace the temperature of the arc may be 7200° F (approximately 4000° C) which vaporizes the chromium in ex-

posed scrap, causing a loss of up to 15 per cent of this element. Since the capacities of induction furnaces have been limited to about three tons, the popular range being from 200 to 2000 pounds, this type of furnace is used only for small heats.

In remelting alloy-steel scrap in these furnaces, it is possible to make melts in less time than with other methods of melting and this minimizes loss of valuable alloying elements by oxidation. Heat is induced in the metal by eddy-currents due to circulating electrical currents of high magnitude. Where heat must be absorbed from radiant energy, as in other types of furnaces, there is a limit to the rate at which the metal charge can absorb heat.

Molten metal in an induction furnace is caused to circulate automatically by electromagnetic action. When alloy additions are made to a molten charge, the stirring action results in creating a homogeneous product in a minimum of time. Due to the turbulent action in the bath and the skim slag, it is possible to get accurate temperature readings with a radiation pyrometer and, by electronic control, hold the bath temperature within $\pm 5^\circ$ F. This is an advantage when pouring small castings, when thirty minutes or more are required to empty the furnace.

Up to a few years ago, the standard equipment used to supply power to these induction furnaces was a rotary motor-generator set, which supplied a frequency in the neighborhood of 1,000 cycles. Lately, mercury-arc frequency changers have been used which have several advantages; the efficiency is higher than that of motor-generator sets and the static device presents no problems in vibration or air ventilation, and the cost is kept to a minimum. Another feature of this type of melting is that the frequency output is determined by the output circuit. If the frequency characteristic of the circuit changes during operation, the mercury-arc inverter automatically will supply the frequency required by the melting circuit.

Special alloys can be produced by melting in a vacuum, or under pressure in an inert-gas atmosphere, by enclosing the entire coreless induction furnace and mold in an airtight container which can be evacuated or put under pressure. When melting is complete, the molten contents of the furnace are poured into the mold; this is accomplished entirely by control elements outside the container that make it possible to tilt the container for pouring the furnace without opening the container. After the metal solidifies, the container can be opened for emptying the mold and recharging the furnace. Section 6 of this chapter gives further details on vacuum and atmosphere melting.

Disadvantages of Induction Melting—Limited holding capacity and inability to control the composition of the melt except through the careful selection of scrap appear as the chief drawbacks of the induction furnace, from metallurgical and commercial viewpoints.

SECTION 6

VACUUM AND ATMOSPHERE MELTING

In recent years, methods have been developed for commercial melting of metals and alloys of certain types in vacuum or under controlled atmospheres. Vacuum-melting techniques are employed in the case of some steels to obtain improved physical and mechanical properties unobtainable in any other way. Some other metals, notably titanium, cannot be melted successfully at all except under such conditions.

Vacuum furnaces have been heated by the electric arc,

by electrical resistance, by gas, and by electrical induction. However, only electrical induction has been used on any sizeable scale for melting steels. This latter method employs a high-frequency coreless induction unit (see Section 5), enclosed in a container or tank which can be either evacuated or filled with an atmosphere of any desired composition and pressure. Provision is made by suitable electrical and mechanical mechanisms and controls for making additions to the

melt and for tilting the furnace after melting to pour its molten contents into the ingot mold which also is enclosed in the tank or container. Most of the vacuum furnaces in operation in the United States are of one-quarter and one-half ton capacity, but larger units melting up to three tons have been proved practicable.

The electrical frequency employed by the coreless induction furnaces for vacuum and atmosphere melting depends upon the capacity of the melting unit, as it does in the case of such furnaces used in ordinary melting processes. Most of the crucibles for vacuum melting have basic linings.

While vacuum melting often has been employed simply as a remelting operation for very pure materials, it is more generally useful in those applications where some refining also is accomplished. Oxygen, nitrogen and hydrogen can be removed from the molten metal in vacuum melting as well as carbon when alloys having very low carbon content are being produced (i.e., some of the stainless steels).

The control of pressure and composition of the gas over a melt makes it possible to deoxidize the melt with carbon or hydrogen, both of which produce gaseous deoxidation products, thus preventing the formation of solid non-metallic inclusions in the finished steel. Also, when melting in a vacuum, the absence of nitrogen from the atmosphere over the melt prevents formation of nitrides and carbonitrides that appear in many steels and high-temperature alloys melted under ordinary atmospheric conditions. The exclusion of oxygen by vacuum melting prevents oxidation losses and permits very close control of the composition of alloys containing easily-oxidized components.

The volatility of certain alloying elements such as chromium, aluminum and manganese may result in high losses of these elements if they are added to steel under a high vacuum. These losses may be minimized

by replacing the vacuum with an inert gas as the atmosphere over the melt during the period when such additions are being made.

The entire subject of vacuum melting of steels is still in the relatively early stages of development. As work continues with this technique, it will be possible to compare the characteristics of metals and alloys melted under reduced pressure or controlled atmosphere with those of materials melted in air by ordinary methods. Vacuum and atmosphere melting of steels and other ferrous alloys will grow in proportion to the need for metals possessing whatever special properties may be developed by these techniques.

Chapter 17

DUPLEX AND TRIPLEX STEELMAKING PROCESSES

SECTION 1

DUPLEX PROCESSES

Definition—The term *duplex process* may be applied to any combination of two processes for manufacturing steel but, in the United States at least, custom has restricted the unmodified term to mean only a combination of the acid Bessemer converter and the basic open-hearth processes, in which the latter plays the part of a finishing process. Briefly, the duplex process consists of blowing molten pig iron in the Bessemer converter until the silicon, manganese and most of the carbon have been oxidized, and then transferring this semi-finished metal to a basic open-hearth furnace where, through the agencies of iron oxide and lime, the phosphorus and the remainder of the carbon are oxidized and lowered to desired limits. The steel is then finished, recarburized and deoxidized as in the usual open-hearth practice.

Tilting Open-Hearth Furnaces—Open-hearth furnaces of the tilting type are especially adapted to the duplex process and the major portion of the tonnage of duplex-process steels is made in such furnaces. This type of open-hearth furnace was originally designed for purposes other than duplexing. The first tilt-

ing open-hearth furnace was placed in operation in 1889 at the Steelton Plant of the former Pennsylvania Steel Company by H. H. Campbell. This antedated the introduction of the duplex process by more than ten years.

The Campbell tilting furnace was introduced as a means of readily using high percentages of pig iron and avoiding some of the operating difficulties then inherent in stationary furnaces. The furnace proper rested on rollers arranged in a circular path, providing rotation on its longitudinal axis. With this arrangement, fuel could be fired even when the furnace was in tipped positions. The advantages claimed by Campbell for the tilting feature were the ease of removal of large slag volumes when high-pig-iron charges were used; the elimination of delays and difficulties in maintaining tapholes, the taphole being located above the level of the bath; the greater ease in repairing bottoms or hearths because the furnace could be drained by tilting; and finally, the greater ease of maintaining the backwalls.

The Campbell furnace was shortly followed by the Wellman tilting furnace, in which the furnace proper

FIG. 17-1. A tilting basic open-hearth furnace of 200 tons capacity, in normal operating position.

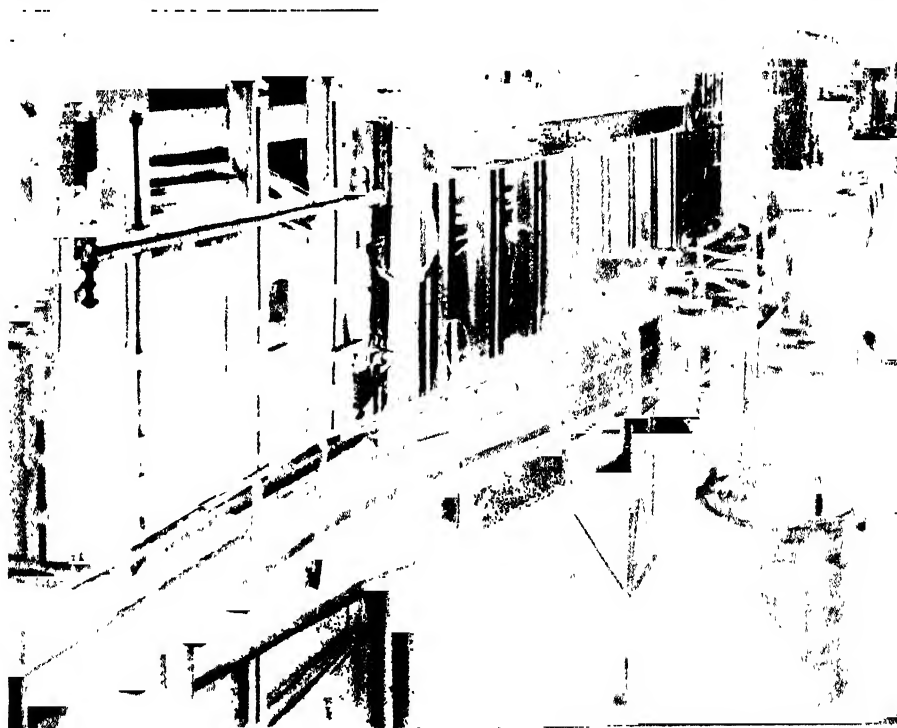




FIG. 17-2. This illustration shows the 200-ton tilting basic open-hearth furnace of Figure 17-1 in tapping position.

rolled forward on a horizontal track, necessitating fuel and air shut-off when the furnace was in tipped position.

Figure 17-1 shows a typical modern tilting open-hearth furnace of the Campbell type in normal operating position; Figure 17-2 shows the same furnace in tilted position during tapping.

DUPLEX PLANT LAYOUT AND EQUIPMENT

Description of Duplex Plant—In a plant to make steel only by the duplex process, Bessemer capacity should balance tilting-furnace capacity, and the converters should be in close proximity to the tilting furnaces to facilitate transfer of metal from one to the other. An exact balance is difficult to plan in advance, for the time the steel is held in the open-hearth furnace varies with the grade of steel being made. For this reason, it is better to arrange for the production of some Bessemer steel along with the duplex production in order to utilize the equipment more fully. Also, the relative number of furnaces necessary to maintain a workable balance depends somewhat upon the tonnage to be produced. For example, one 20-ton converter serving a 200-ton tilting furnace would be an unsatisfactory combination, as interruptions in the operation of either would delay the other. But a combination in which three 25-ton converters serve three 200-ton tilting furnaces gives a well-balanced operating unit for the continuous production of steel.

In addition to the required number of furnaces and converters for the steel production desired, it is generally considered necessary to have at least two hot-metal mixers. Adequate facilities for transfer of iron to the converter and of blown metal from the converters to the open-hearth floor must be provided in order to prevent delays in furnace charging.

In general, all other facilities provided for a scrap-melting shop, including facilities for ladle repair, mold preparation and slag disposal, must be provided. The requirements for charging machines, boxes and buggies

are not so heavy as in a scrap shop. On the pit side, pouring facilities must be particularly well organized in order to tap, pour and move the heats on the rapid schedules attained in duplex operation.

In addition to the auxiliary equipment necessary for an open-hearth shop, much of the apparatus necessary for a Bessemer shop must also be provided. In this connection blowing equipment, vessel-tilting machinery, vessel and vessel bottom-repair equipment are the most important.

The application of the principles of the duplex process to the making of steel has resulted in various modifications. Some of the most common examples will be described in the following sections, using figures based on practices in typical plants.

A. CONTINUOUS PROCESS MAINTAINING A DOMINANT POOL

This description will cover a process developed in the South for the manufacture of steel in basic-lined tilting open-hearth furnaces, using hot metal produced in blast furnaces charged with a blended iron ore of relatively high phosphorus content. Typical contents of various constituents of the fractions of such a blended ore may be as follows:

Constituent	Percentage Content		
	Coarse Fraction	Medium Fraction	Sinter Fraction
Moisture	3.40	5.23	1.50
Iron (Fe)	43.23	49.10	58.33
Silica (SiO_2)	11.00	6.92	6.57
Alumina (Al_2O_3)	2.61	2.23	2.58
Lime (CaO)	9.50	5.72	4.98
Manganese (Mn)	0.13	0.09	0.09
Phosphorus (P)	0.25	0.19	0.19

Hot metal produced from ore of this composition by the blast furnaces may be expected to have a composition falling within the following specification ranges:

Element	Per Cent
Silicon	0.70 - 1.00
Sulphur	0.075 maximum
Phosphorus	0.40 - 0.60
Manganese	0.13 - 0.30

The basic tilting open-hearth practice developed to use high-phosphorus hot metal of this composition for steel-making requires the following essential steps: the maintenance of a **dominant pool** of metal in a tilting basic open-hearth furnace, adding approximately 80 per cent of the next succeeding heat in the form of blown metal from the acid-Bessemer converters, impregnating the basic finishing slag of the previous heat with the phosphorus contained by this blown metal, and then flushing the phosphorus-laden slag from the furnace.

One open-hearth plant employing this process consists of two shops with five and four tilting furnaces, respectively. Between these two shops is a converting department with one 1250-ton and one 650-ton mixer, together with three 20-ton acid-Bessemer converters. The location of the converting department lends itself to an operation with a minimum of delay in delivering blown metal to the open-hearth furnaces. In case of a delay in one shop, the blown metal generally can be delivered to one of the furnaces in the other. Blown metal is poured out of the converters on the open-hearth charging-floor level and is transported in 25-ton ladles mounted in trunnions on ladle cars. Metal is poured into an open-hearth furnace by tilting the ladle in its trunnions on the car.

The tilting open-hearth furnaces are of the Campbell type in which the furnace is mounted on rocker arms and rollers and can be tilted forward and backward about a horizontal axis. The axis of rotation of each of these furnaces is coincident with the center-line of the ports and rotation of a furnace does not interfere with firing. The framework is of much stronger construction than that for ordinary open-hearth furnaces in order to resist twisting stresses and vibration which would be very harmful to the brickwork.

Only that section of a furnace comprising the hearth, sidewalls, and roof is made tilting; all the ports and flues are stationary, and, together with the checkerwork, are of the same construction as in stationary furnaces. The clearance between the movable and stationary parts of the ports is kept very small by the use of water-cooled joints, in order to keep air and gas leakage to a minimum. On the pouring side, these furnaces have but one opening, a tapping hole located above the slag line and provided with a lip or spout for directing the stream of molten metal into the steel ladle. As in the case of stationary furnaces, doors for introducing the materials into the furnace are located on the front side. Slag notches are also located in the middle doors on the front side.

Preparing the Furnace for Charging—The process may be said to be continuous. When it is necessary to make repairs, the tilting furnace is thoroughly drained, the bottom and slag lines are made up, the ports are cleaned and repaired, and everything is made ready for the resumption of standard production. Of course, during operation the front and backwall must be attended to and such minor repairs made as are found necessary. Before any iron is blown in the converters, the preparation of a slag is begun by charging and melting down lime and oxides. Considerable care is given by the melter to the preparation of a good slag, for, as in all open-hearth work, the success of the process depends upon the slag.

Except for the first heats, there are from 25 to 35 tons of each 175-ton heat of steel retained in the furnace at each tapping; this forms what is referred to as the "dominant pool."

Cycle of Operations—When a ladle is filled with finished steel tapped from one heat, the steel in the ladle is covered with a blanket of slag which flows out of the taphole as the furnace is rolling back to an upright position. Most of the slag is retained in the furnace. When a furnace is completely upright, the slag level is below the taphole. As soon as tapping is completed, approximately 80 per cent of the next succeeding heat is immediately added to the furnace in the form of full-blown metal which usually consists of five "pots" or ladles full of blown metal totaling approximately 100 tons. The blown metal, being washed through the highly-basic finishing slag of the previous heat, is rapidly dephosphorized. This slag is then immediately poured off the washed metal over the foreplate into slag pots on cars on narrow-gauge tracks at basement level. The slag-making constituents, for refining the washed metal that is the starting point for the new heat, is then charged; these usually consist of burnt lime and oxides as required. From 1 to 1½ hours are required for the charged burnt lime and oxides to melt and form a suitable slag. When the slag is in proper condition, two pots or ladles of partially-blown high-carbon metal are added to supply the carbon necessary for working the heat. The refining period from time of the addition of the last partially-blown or high-carbon pot requires from one hour and thirty minutes to two hours and thirty minutes, depending upon the type of steel being produced.

In the blowing operation silicon, carbon and manganese are all practically removed when blowing a full-blown, low carbon or "soft" pot. The phosphorus content, due to the loss of carbon, silicon, manganese and some iron during blowing, is increased from an average of 0.50 per cent in the iron to around 0.55 per cent in the blown metal. The greater part of the silicon, along with some of the carbon and manganese are removed in blowing a high-carbon pot. A definite range of ferrous-oxide content of the slag is necessary for effective dephosphorization of the blown metal, and this is maintained by suitable additions to the bath of burnt lime, mill scale, ore and ferromanganese or ferrosilicon.

With this operation it is possible to produce steel ranging from low-metalloid grades to carbon tool steels.

Advantages of the Process—Considerable benefit is derived from the use of full-blown metal charges, eliminating contamination of the bath by undesirable alloys normally present in scrap. This is especially true when producing a low-metalloid heat.

One of the benefits derived from this process is a minimizing of reaction between slag and steel during tapping; more of this reaction occurs at tap in conventional stationary open-hearth practice. This advantage is made possible by plugging the taphole of the tilting furnace with bagging which holds until the slag rises above the hole while the furnace is being tilted. The only slag removed is that amount coming out after the ladle is filled and the furnace is being rolled back to an upright position. This provides a relatively thin blanket to prevent the steel in the ladle from freezing.

The process provides uniformity from heat to heat through the duplication of desired heat cycles. During all periods the fuel is not checked and little loss in temperature occurs. This is advantageous as regards the useful life of the roof and other brickwork of the furnace.

This process can be flexible in its use of charged materials. When, due to shortages of hot metal, increased percentages of scrap are required, the additional scrap may be charged in small amounts into several furnaces or may be confined to one furnace, although increasing the percentage of scrap charged does lengthen the heat cycle. The tilting furnace facilitates bottom repair in that the furnace may be tilted forward or backward to drain a hole which would have to be rabbled out in the stationary type of furnace.

Slag Composition—Composition of a typical refining slag and composition of the slag after the addition of the full-blown "soft" pots is as follows:

Constituent	Refining Slag (%)	Before Slagging Over Foreplate (%)
Ferrous oxide (FeO)	14.00	4.50
Ferric oxide (Fe ₂ O ₃)	8.60	5.50
Silica (SiO ₂)	9.75	12.50
Lime (CaO)	48.00	49.00
Magnesia (MgO)	4.00	4.00
Manganous oxide (MnO)	2.50	1.50
Phosphoric Acid (P ₂ O ₅)	8.00 - 13.00	10.00 - 20.00

The high-phosphorus slag obtained after the addition of the full-blown "soft" pots is diluted to some extent by the "kick-off" or flush slag resulting from the reaction at the time of addition of the partially-blown high-carbon pots of metal.

Deficiencies of the Dominant Pool Method—The most valid criticisms of the dominant pool method are that, (1) the tapping slag which remains in the furnace after a heat contains a high phosphorus content, (2) the pool of metal remaining in the furnace is likely to become too highly oxidized before the furnace is filled up again, (3) the fluxing materials for the next heat are charged on top of and float on the remaining pool, failing to give the beneficial effects that can be derived from a prolonged boil from lime charged on the bottom, (4) frequent examination of the furnace bottom is impossible with resulting long "bottom delays."

B. THE DRY BOTTOM OR SINGLE HEAT DUPLEX PROCESS

This duplex method is commonly used in northern United States at present, except where special conditions dictate continuation of the dominant pool method. It is a continuous operation with no week-end bottom-repair period. Several different practices of preparing the charge are used and local conditions are the factors that determine which practice shall be employed. The method is so flexible that all of the practices in use produce steel of satisfactory quality and quantity; consequently, only one dry-bottom practice is described.

The Charge—After the previous heat has been tapped, the furnace is turned into its normal operating position. The melter foreman examines the bottom and, if no repairs are necessary, orders the next charge from the Bessemer steel blower. The amount of limestone charged is determined by the grade of steel to be made. If a heat of rimmed steel is to be produced, 16,800 pounds of raw limestone will be charged for a 145 ton heat. If the heat is to be semikilled or killed, this amount is reduced to 11,200 pounds. As soon as the melter foreman has ordered the charge, approximately one-half of the limestone is charged on the furnace bottom and the furnace crew proceed with the slag line repairs to the front, back, and ends.

To produce an open-hearth heat of 145 tons, approximately 335,000 pounds of liquid pig iron are ordered for the converters. Two heats (blows) of 45,000 and 50,000 pounds each are blown simultaneously in the Bessemer converters and when they reach a point in the blow where the carbon content is between 0.10 and 0.20 per cent they are turned down and poured into one ladle. This ladle of metal is carried by an overhead crane and poured through a spout into the open-hearth furnace. After the first ladle of metal has been poured into the furnace, the remainder of the limestone is charged. The balance of the blown metal follows, there being three ladles in all.

After the last heat has been blown in the converters, liquid basic pig iron in amounts ranging from 35,000 to 47,000 pounds is brought from the mixers and charged into the open-hearth furnace. The unblown basic hot metal is used because its manganese content is higher than blown Bessemer hot metal, thus assuring a residual manganese in the charge comparable to that encountered in any regular open-hearth heat. Varying amounts of this metal are used to provide the initial carbon content of the bath necessary for different grades of steel. The amounts of blown metal, therefore, must obviously be adjusted so that the complete charge will total approximately 310,000 pounds.

There are three reasons for stopping the blow when the carbon content reaches 0.10 and 0.20 per cent; first, this eliminates the possibility of over-oxidation of the metal; second, the nitrogen content of the steel will be lower than in full blown metal; and third, it assists in producing a slag in the acid converter that is sufficiently thick (viscous) to minimize its carry-over to the basic open-hearth furnace.

A summary of the charge is as follows:

	Rimmed Steel (Pounds)	Semi-killed or Killed Steel (Pounds)
Limestone	8,400	7,000
Blown Metal	90,000	90,000
Limestone	8,400	4,200
Blown Metal	90,000	90,000
Blown Metal	90,000	90,000
Basic Hot Metal	40,000	40,000
Total Limestone	16,800	11,200
Total Blown Metal	270,000	270,000
Total Hot Metal	40,000	40,000

The difference between the 335,000 pounds of metal ordered from the mixers and the 310,000 pounds in the open-hearth charge is, of course, that portion lost during the blowing process.

Working the Heat—Approximately thirty minutes after the furnace has been completely charged, a test is taken and analyzed for carbon and manganese. This analysis indicates to the furnace crew the amounts of lump ore necessary to reduce the carbon to the desired point. All heats are charged to provide an initial carbon content about 40 "points" (0.40 per cent) above the tapping carbon, with the exception of certain grades for extra-deep-drawing applications. On these latter grades a much higher initial carbon is desired and provided because the increased ore additions necessary for the carbon reduction promote an active boil of the bath for a longer period of time and thus reduce the nitrogen content of the steel. The normal nitrogen content of steels made by the liquid-charge method is 0.006 to 0.008 per cent, while in standard basic open-hearth steels the range is 0.004 to 0.006 per cent. The additional bath activity described above, however, re-

duces the nitrogen content to the 0.004 to 0.006 per cent range, and it is this slight reduction in nitrogen content that permits steel made by the liquid-metal process to compete favorably on applications demanding a high degree of ductility and relative insensitivity to strain aging.

During the early working period of the heat, the lime boil takes place until the limestone charged on the bottom is thoroughly calcined and rises to the surface to form the slag.

When the bath has reached the necessary temperature, additions of lump ore are begun and proceed periodically throughout the working period as demanded. During this period chemical analyses are made for phosphorus and sulphur, and carbometer tests are taken to indicate to the furnace crew the extent of carbon elimination.

Pancake slag tests are also taken and additional burnt limestone or mill roll scale is added for the purpose of providing the proper slag consistency.

The heat is "worked down" in the same manner as any basic open-hearth heat, and when the desired carbon content is reached, a ladle is set on the tapping side of the furnace. The furnace is then tilted to permit the slag to rise above the tap hole and the metal to flow into the ladle.

Recarburizing material is usually kept to a minimum, the carbon being "caught on the way down." Deoxidizing materials, such as ferromanganese, silicon, aluminum, titanium, etc., are added to the ladle the same as in any other open-hearth heat.

The Slag.—The slag volume for steel made by this duplex process is less than on regular open-hearth steel, running approximately 6.5 per cent of the weight of the finished heat. The limestone charged is normally 90 to 95 pounds per ton of steel; this is possible because of the very low silicon content of the charge. No difficulty is experienced in the removal of phosphorus, except when a considerable amount of Bessemer slag is allowed to enter the furnace with the charge.

The average composition in per cent of tapping slag of a steel tapped at 0.09 per cent carbon and 0.17 per cent residual manganese follows:

SiO ₂	11.58 %	MnO	8.86
Al ₂ O ₃	1.16	CaO	41.50
TiO ₂	0.29	MgO	6.23
FeO	18.55	P ₂ O ₅	3.16
Fe ₂ O ₃	5.57	S	0.17

Heat time averages approximately four hours tap to tap.

C. COMBINATION SCRAP AND BLOWN METAL METHOD

This type of operation may be used in tilting-furnace shops, but is particularly adapted for use in plants that have stationary furnaces, yet desire to take advantage of the faster-melting duplex practice. Although many variations of the method are possible, depending upon local operating and economic conditions, the system described is one that has been very successful from a production standpoint in one of the eastern plants.

The practice employed is to charge the limestone, ore, scrap and hot metal as for a regular high-iron charge. After the slag flush is finished, the full-blown Bessemer metal is added in amounts of about 16 or 32 per cent of the total charge, and a second quantity of slag is flushed off. Satisfactory slag flushes are obtained when the blown metal is either 16 or 32 per cent of the total charge. The amount of ore charged is regulated to specific conditions, such as type of scrap, age of furnace, and carbon desired at melt down. The limestone requirements for the blown-metal heats are regulated by the amount of silica in the charge in the same manner as for regular high-iron charges. The fuel requirements for 16 and 32 per cent blown-metal heats are about 10 and 20 per cent, respectively, below those of high-iron-charge heats made in the same shop. The melting times for these two types of heats are approximately 10 and 25 per cent less than the corresponding high-iron-charge heats, but approximately 100 per cent longer than liquid-metal heats made by dry-bottom practice.

As soon as the melt-down is complete, the working of the heat and slag adjustments take place just as in dry-bottom duplex or scrap and hot-metal heats. When the analysis shows that the heat is ready to tap, additions are made as in other heats produced in a stationary furnace, the furnace is tapped out clean and the steel is deoxidized in the usual manner.

Table 17—I indicates the time-saving and corresponding increased production possible with this type of operation.

The quality of the steel made with this type of charge corresponds favorably to that of regular basic open-hearth steel.

ADVANTAGES AND DISADVANTAGES OF THE DUPLEX PROCESSES

In the northern district of the United States, the chief advantages of the duplex process are the increased

Table 17—I. Comparison of Heat Times and Production Rates for High-Iron, 16% Blown-Metal, and 32% Blown-Metal Charges.

Type of Charge	No. of Heats in Avg.	Time of Heats				Improvement in Chg.-Tap Time			Avg. Tons per 24 Hrs.
		Chg.-Tap		Tap-Tap		hrs.	min.	percent	
		hrs.	min.	hrs.	min.				
High Iron	726	10	53	12	22	326.4
16% Blown	291	9	41	11	18	1	12	11.0	356.0
32% Blown	79	8	12	9	48	2	41	24.7	417.0

Type of Charge

High Iron
16% Blown
32% Blown

Percentage Increase In Production per 24 Hours

9.05%
27.70%

tonnage which it produces in a given time, and its freedom from dependence on purchased scrap. In the southern district, the duplex process also provides a method of manufacturing high quality steel from high phosphorus pig iron, and, in addition, produces a highly basic slag containing a high percentage of citric-acid-soluble P_2O_5 which is valuable as a soil conditioner. Thus, while the product is similar in quality and of the same grades as basic open-hearth steel, the time of the open-hearth operation is shortened by more than half. Whereas one open-hearth furnace will turn out an average of about fifteen heats in a week of melting scrap and hot metal or scrap and ore heats, the same furnace operated as a duplexing unit will produce about forty heats in the same period. This shortening of the time of heats saves fuel, and this factor, together with the elimination of the silicon in the converter slag, tends to prolong the life of the open-hearth furnace. The process does not require the use of scrap, an advantage when scrap is scarce and high in price.

The duplex process is capable of making any type of heat which can be made in a basic open-hearth furnace, but because of the ability of a scrap shop to recover alloys from scrap, most alloy steels are made in shops charging scrap rather than in duplex shops. On the other hand, where heats of low residual-alloy content are desired, a duplex shop is at a distinct advantage as compared with a scrap and hot-metal shop. In respect to flexibility, a duplex plant is in a far better position than one equipped only for scrap and hot metal because of the possibility of charging either scrap or blown metal, whichever is cheaper at the time. Naturally, a shop designed for duplex operation which changed over to scrap melting in order to take advantage of cheap scrap, would fail to melt the tonnage attainable on duplex operation.

In recent years, steel produced by the duplex process, according to the improved practices discussed earlier, has been applied to an ever-increasing number of uses for which basic open-hearth steel formerly was thought to be better suited. As knowledge increased as to the effects of relatively small amounts of various constituents of steel on its properties, it was established that the relatively higher nitrogen content of duplex steel (as compared with basic open-hearth steel) was the principal factor influencing its properties and resulting in certain disadvantages of that earlier duplex steel. The chief effect of the higher nitrogen content of the earlier duplex steel was to increase its susceptibility to strain aging. When these facts were ascertained, duplex steel-

making practices were altered and improved in a way which would consistently produce steel having a controlled nitrogen content but little greater than that of basic open-hearth steel, as indicated by the following figures showing the ranges of nitrogen content common to present-day Bessemer, duplex, and basic open-hearth steels:

Type of Steel	Average Nitrogen Content
Bessemer	0.012 to 0.020%
Duplex	0.005 to 0.008
Basic open-hearth	0.004 to 0.006

Because the chemical composition of duplex steel made by the improved practices approaches that of basic open-hearth steel, and can be controlled closely, it is understandable that the new duplex steel now can be applied to many purposes for which it once was considered less suitable than open-hearth steel.

The duplex process is now employed to produce many high-quality grades of steel for such applications as forged automotive crankshafts, seamless tubes, rolled sections for automotive-wheel rims, and other products ranging from high-grade thin flat-rolled materials to railroad rails. It may be noted that the slightly higher nitrogen content of duplex steels makes them somewhat harder than open-hearth steels of otherwise similar composition; this may be an actual advantage in a few cases, for example, in the manufacture of low-phosphorus steel for tin plate that must meet high-temper specifications.

The high-quality raw materials provided for successful duplex practice, coupled with the present satisfactory performance of the product, leads to the belief that further expansion will be made in application of duplex steel. The present status of basic open-hearth practice, wherein the charge material contains ever-increasing amounts of sulphur and unwanted alloys from scrap, may result in the wider use of the duplex process.

The principal disadvantages of the duplex process are the initial cost of providing a plant with both open-hearth and Bessemer facilities, and lower ingot yield brought about through the combined Bessemer and open-hearth conversion losses. The double conversion cost of producing steel by this method would also appear to place it in an unfavorable competitive position. Based on its higher rate of production, however, the seriousness of the initial cost is offset to a great extent.

SECTION 2

TRIPLEX PROCESSES

The term "triplexing" refers to a combination steel-making practice in which the same steel is processed successively in Bessemer, open-hearth and electric furnace with the usual aim being to produce a steel of electric-furnace quality.

Just as there are theoretically many possible duplexing combinations, as outlined earlier, so there are even more possible combinations of both acid and basic furnaces for the manufacture of triplex steel. However, only a limited number of these combined processes has been used for any length of time. The excessive maintenance cost of three furnaces, high cost of handling molten metal, and the excessive heat losses in handling have prevented the use of the process except under highly specialized conditions. Also, electric-furnace

steels have been produced by charging blown metal direct from an acid converter into a basic electric furnace to produce electric-furnace steel of good quality at a high rate. The knowledge that this is possible serves to illustrate the wastefulness of triplexing when, as in most cases, the same end can be accomplished by a duplex process.

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Chapter 18

CASTINGS - STEEL AND IRON

SECTION 1

STEEL CASTINGS

Casting Compared with Other Forms of Shaping Steel—The process of making steel castings consists simply of pouring metal into a mold which is of the desired shape, dimensionally accurate and of sufficient stability to permit the metal to solidify in the exact shape of the mold cavity. Intricate and complicated castings of practically any desired shape or size, and for almost any particular application, can be made in this manner. The versatility of the foundry trade commands recognition throughout the industrial world, and design engineers in every field of endeavor have come to rely upon the utility of steel castings.

Mechanically, steel castings are considered inferior to wrought-steel products, and it is true that wrought steels do exhibit higher mechanical properties than cast steels, especially when the former are tested in the direction of rolling or forging. Moreover, cast structures, unless designed in strict conformity with the natural characteristics of metal solidification, sometimes contain internal defects or surface imperfections which may seriously affect serviceability and otherwise render them less dependable than a wrought-steel product. However, the casting of steel is the most direct method of producing a given shape and, for this reason, the method provides the basis for a key industry.

Tonnagewise, the steel-casting or steel-foundry industry would appear to be of minor importance, approximately two per cent of the total steel production per year being in the form of castings. However, few industries enjoy so prominent a role in the industrial and domestic development of the world as the foundry industry. Many articles, especially turbine shells, valve bodies, and machine parts are made by casting, not by choice, but because they cannot be made as readily by other processes. For example, valve bodies, exhaust manifolds, pump casings, and turbine diaphragms could be fabricated by other processes or from wrought products only with extreme difficulty and at a serious economic disadvantage. Also, cast structures are generally considered to be more rigid than their wrought counterparts.

Steel castings vary in weight from a few ounces to hundreds of tons and cover a multitude of designs and services. Latch castings for airplane cowls weigh less than four ounces, while one of the largest castings ever produced was a 240-ton casting for an armor press of 35,000 tons capacity. Castings for steel-mill service alone cover a very large field and include housings, gears, charging boxes, guides, blast-furnace bells, hoppers and cinder pots. Rolls for certain types of rolling mills are made of cast steel; because of the specialized foundry techniques employed in roll casting, the manufacture of rolling-mill rolls by casting is discussed separately and at length in Chapter 23. The transportation industry re-

lies upon castings for railroad and marine use, such as couplings, journal boxes, bolsters, frames, brake shoes, cylinders, housings, valves, crankshafts, engine beds and steam chests. These are but a few of the examples in everyday use and hundreds of other applications could be mentioned serving the chemical, petroleum, mining, excavating, agricultural, ceramic and construction industries.

Composition and Mechanical Properties of Cast Steels—The development of the mechanical properties of cast steels depends almost wholly upon heat treatment, and since the depth of hardening in carbon steels is limited, it follows that alloys in varying proportions must be added when high strength is required. This is especially true of castings with heavy metal sections. Nickel, chromium, manganese, molybdenum, and vanadium are alloys commonly added to cast steels. These are added according to the manufacturers' past experiences and heat-treating facilities. Table 18—I shows typical compositions and results of tension tests for some heat-treated cast steels.

MAKING STEEL FOR CASTINGS

Five types of furnaces are or have been employed by foundries for melting steel. These are: (1) the direct-arc electric furnace; (2) the open-hearth furnace; (3) the pneumatic converter; (4) the electric-induction furnace; and (5) the crucible furnace. The pneumatic converter and the crucible processes, formerly of importance, have now been supplanted almost entirely by electric-furnace melting. The size and type of castings being made dictate the type of furnace. Foundries making small castings weighing from 1 to 500 pounds favor the electric process. Producers of large castings weighing from 500 pounds up to 25 tons or more use open-hearth furnaces.

Electric furnaces in foundries vary in size and the normal weight of charge runs from 250 pounds to 25 tons. Open-hearth furnaces in foundry practice usually have capacities between 10 tons and 100 tons. The present trend in melting equipment seems to be in favor of the electric process and many of the open-hearth foundries are converting to electric-furnace practice. The reason for this is the greater flexibility of the electric furnace over the open hearth, as well as various economic considerations.

In general, furnace charges consist of purchased scrap in the form of billets, shearings, flashings, punchings, plates and turnings; also, scrap from the foundry itself in the form of gates, heads, and scrapped castings. High-grade pig iron, low in phosphorus and sulphur, is used in the open-hearth furnace charges, and frequently in the electric-furnace charges when a high-carbon melt

is required. Ferroalloys and slag-making materials are used in about the same proportions in the melt as for steel-ingot production. However, the casting-from-melt yield is considerably lower than that generally experienced with ingots. For every ton of steel castings produced, at least three tons of raw materials are consumed, including scrap steel, pig iron, fuel oil, limestone, sand, organic and clay binders, and miscellaneous materials. This includes the mold-making materials.

Fifty per cent of the steel-casting tonnage is melted in open-hearth furnaces, approximately half being made in basic-lined furnaces. The basic furnace offers one distinct advantage in permitting partial removal of sulphur and phosphorus, thereby making possible the use of less costly scrap. However, in localities where scrap low in sulphur and phosphorus is abundant, the acid open-hearth process commonly is used for various reasons, some of which are largely a matter of beliefs. For example, it is claimed that acid open-hearth steel has less flakes and cooling cracks, and that it has fewer internal cracks or ruptures in heavy sections. Neither of these claims has been proved definitely. Records indicate, however, that less melting skill is required with the acid process, that refractory costs are lower, melting time is faster and that, in pouring castings, acid open-hearth steel has greater fluidity than basic open-hearth steel.

Much has been written concerning the operation of open-hearth furnaces, and elsewhere in this book the subject is treated in some detail. Suffice it to say that the melting process in foundry practice is very similar to that practiced in the steel mills, the chief difference being that greater amounts of deoxidizers are required in foundry practice to produce sound castings, free from porosity. It is necessary to add silicon and manganese in excess of 0.30 per cent and 0.60 per cent, respectively, for deoxidation purposes. Also, aluminum and alloys of calcium, manganese, silicon and zirconium commonly are added to insure complete deoxidation, removal of dissolved gases such as hydrogen and nitrogen, and, finally, for grain-size control.

Electric furnaces offer several distinct advantages over the open-hearth furnaces in the foundry. The modern arc furnace, with its top-charge mechanism, improved switchgear and automatic control, is an extremely flexible unit and rapidly is replacing the open-hearth furnace in the foundry. Electric furnaces permit a wider range of scrap selection than the open hearth, can be charged quickly and without the aid of a charging machine, operation requires fewer men, and the power costs are less than fuel costs in many localities. The furnaces can be shut down and allowed to cool to room temperature with less damage. The types of steels that can be produced in electric furnaces are unlimited. Highly-alloyed steels can be made as well as the plain carbon steels and electric furnaces are more flexible in the size and type of heats. This facilitates foundry planning and affects customer relations by the ability to make quicker deliveries.

Electric furnaces are lined with acid or basic refractories as available scrap or product characteristics may warrant. High-manganese steels are always melted in basic-lined furnaces because the slag from such steels is very destructive to acid refractories. Where low-sulphur and low-phosphorus scrap is not available, basic slags must be used to remove these elements, hence basic-lined furnaces are used. However, the major part of the electric-steel-casting tonnage is made by the acid-electric-furnace process.

The generally accepted method for making steel in acid electric furnaces is to melt hot, then by the applica-

tion of either iron ore or oxygen, bring the bath to a violent boil or "blow," following which the heat is tapped before the action subsides completely. The metal generally is deoxidized in the ladle by the addition of relatively large amounts of deoxidizers.

There is greater variation in the melting operation of basic furnaces than acid furnaces. However, the three principal methods employed in the basic furnace are: (1) the **dead-melt process**, in which a reducing slag is made up as quickly as possible and maintained throughout the heat; (2) the **double-slag process** in which the first oxidizing slag is removed and replaced with a white, lime finishing slag; and (3) the **single-slag method** with an oxidizing slag which may or may not be made reducing before tapping. Each method has its own specific advantages, depending upon the type of steel being made.

MOLDING FOR CASTING STEEL

Patterns and Molds for Steel Castings—The construction of the **pattern** is perhaps the most important single factor in the production of a casting. Not only must the pattern be dimensionally accurate, but full consideration must be given to making it meet the requirements of the foundry equipment and technique. The patterns for steel castings are usually made $\frac{3}{16}$ to $\frac{1}{4}$ of an inch per foot larger than the dimensions shown on the drawing to compensate for metal shrinkage, because steel castings cooling from the liquid state to room temperature contract approximately $\frac{1}{4}$ inch per foot, depending upon the chemical composition of the steel and the size and design of the casting. It follows, therefore, that to make dimensionally accurate castings provision must be made for metal shrinkage. Patternmakers make such adjustments by **shrink rules** or **patternmakers' rules** which are graduated to compensate for the necessary shrink allowance. Other details of pattern construction, such as allowance for **draft** to facilitate removal of the pattern from the molding sand, **padding** for feed purposes, avoidance of sharp changes in metal sections, and elimination of sharp edges, corners and reentrant angles, all require full consideration.

There are several types or classes of patterns, each fulfilling a specific need. Patterns may be made of wood or metal, as required, and used in conjunction with hand-molding or machine-molding methods, depending upon the number of castings to be made and the degree of precision required. Descriptions of patterns and molding machines and procedures will be found in the books listed at the end of this chapter.

The materials used for making molds vary to a great extent, not only from foundry to foundry, but within the same foundry. The size and type of castings, the composition and temperature of the metal, pouring methods and foundry technique, sand mixing and reclaiming facilities, and location of the foundry exercise a profound influence upon the type of molding material. Some foundries prefer to make molds of crude or **bank sand**. For special work calcined ganister or **Chamotte** is sometimes favored.

Patterns have been issued for many molding media, such as those used in the **Fischer process** (calcined aluminous-clay grog), or in the **Randupson process** (sand and cement). Highly refractory silica sands of known particle size, mixed with weighed or measured quantities of various types of clays, resins, dextrins, vegetable oils, and water to develop desired molding characteristics are used. **Plaster, plastic and wax patterns** have been used for special purposes, such as making an experimental casting. Wax patterns are used ex-

Table 18-L Compositions and Mechanical Properties of Some Cast Steels.
COMPOSITIONS

Type of Steel	C (%)	Mn (%)	Si (%)	Ni (%)	Cr (%)	V (%)	Mo (%)	Cu (%)
Low-Carbon	0.12	0.32	0.25					
	0.19	0.60	0.44					
	0.19	0.60	0.44					
	0.11	0.60	0.30					
	0.17	0.67	0.23					
Medium-Carbon	0.21	0.55	0.45					
	0.25	0.57	0.41					
	0.28	0.63	0.47					
	0.28	0.62	0.40		0.10*			
	0.27	0.69	0.26					
	0.31	0.75	0.42					
	0.31	0.75	0.42					
	0.42	0.69	0.43					
	0.46	0.73	0.28					
	0.50	0.59	0.54					
High-Carbon	0.56	0.62	0.47					
	0.84	0.73	0.44					
2% Nickel	0.15-0.35	0.60-1.00		2.00				
3% Nickel	0.20-0.45	0.50-0.80		2.50-3.50				
Nickel-Manganese	0.20-0.40	0.90-1.50		0.75-1.50				
Nickel-Vanadium	0.20-0.35	0.60-1.10		1.40-1.75		0.80-0.15		
Nickel-Molybdenum	0.20-0.35	0.60-1.00		1.25-2.00			0.25-0.40	
Nickel-Chromium-Molybdenum	0.30-0.50	0.60-0.90		1.50-3.00	0.45-1.00		0.20-0.40	
Medium-Manganese	0.30-0.40	1.35-1.55	0.25-0.50					
Manganese-Molybdenum	0.20-0.40	1.10-1.50	0.25-0.50					
Low-Chromium	0.25-0.40	0.65-0.85			0.60-0.90		0.25-0.50	
2%-3% Chromium	0.25-0.40	0.65-0.85			2.00-3.00			
4%-6% Chromium	0.10-0.30	0.60-1.00			4.00-6.00			
Chromium-Molybdenum	0.25-0.50	0.65-0.85			0.50-1.50		0.20-0.50	
Carbon-Molybdenum	0.25-0.40	0.60-0.80					0.30-0.50	
Vanadium	0.25-0.40	0.65-0.90	0.30-0.50			0.15-0.25		
Copper	0.15-0.40	0.70-1.00	0.30-0.50					1.00-2.00
Hadfield Manganese	1.00-1.40	10.00-14.00	0.30-1.00					

*Residual element.

Table 18-1 (Continued)
MECHANICAL PROPERTIES

Type of Steel	Heat Treatment	Tensile Strength (Lb./Sq. In.)	Yield Strength (Lb./Sq. In.)	Elongation (%)	Reduction of Area (%)	Brinell Hardness Number	Impact** Properties (Ft.-Lb.)	Source
Low-Carbon	Annealed	51,000	26,000	36.2	66.3	Korber & Pomp Lorig Lorig
	Annealed	71,500	46,500	34.0	58.0	139	24 ^a	
	Normalized and water quenched and tempered at 1250°F.	74,500	48,000	32.0	53.1	142	20 ^a	
Medium-Carbon	Annealed	62,500	35,500	36.5	59.6	Korber & Pomp
	Annealed	64,000	38,000	28.5	40.2	
	Annealed	77,100	40,900	27.0	42.5	
High-Carbon	Annealed	77,950	40,600	32.0	51.9	Phillips Lorig Lorig
	Annealed	80,450	42,950	29.0	42.9	
	Water quenched and tempered at 1050°F.	101,250	84,350	18.0	43.7	207	...	
2% Nickel	Normalized	76,000	41,500	28.0	44.8	156	14 ^a	Hall-Nissen-Taylor Melmouth Hall-Nissen-Taylor
	Annealed	77,000	43,500	28.7	44.5	134	20 ^a	
	Normalized	83,500	53,000	29.3	51.9	146	...	
Nickel-Manganese	Annealed	77,000	...	22.0	23.0	Internat'l. Nickel Co.
	Annealed	93,000	...	22.0	33.6	
	Annealed	84,000	...	19.8	24.5	
Nickel-Vanadium	Annealed	91,250	56,300	16.0	21.0	Internat'l. Nickel Co.
	Annealed	114,750	60,150	8.0	12.7	
	Normalized and tempered	80,500	50,000	28.0	59.0	
Nickel-Molybdenum	Normalized and tempered	97,100	60,700	26.0	53.0	Internat'l. Nickel Co.
	Fully annealed	85,500	53,500	29.0	55.0	
	Normalized and tempered	100,000	63,500	23.5	46.5	
Nickel-Chromium-Molybdenum	Normalized and tempered	89,600	56,150	26.5	51.5	Internat'l. Nickel Co.
	Normalized and tempered	96,500	69,800	26.0	55.0	
	Normalized and tempered	105,000	79,000	26.0	51.0	
Manganese-Molybdenum	Normalized and tempered	83,500	57,500	27.0	52.0	Internat'l. Nickel Co.
	Normalized and tempered	88,000	55,000	25.0	48.0	
	Normalized and tempered	104,000	81,300	21.0	46.5	
Low-Chromium	Water quenched and tempered	149,000	135,000	12.0	32.0	Industry
	Normalized and tempered	88,850	51,200	27.5	58.1	187	...	
	Quenched and tempered	98,800	61,450	22.0	44.0	202	15 ^a	
2% Chromium	Normalized and tempered	85,550	56,600	29.0	56.4	196	...	Industry
	Normalized and tempered	119,400	95,050	21.0	51.4	235	40 ^b	
	Normalized and tempered	84,000	51,000	33.0	63.0	187	26 ^b	
4% Chromium	Normalized and tempered	99,500	58,400	16.0	29.0	Industry
	Normalized and tempered	118,000	94,000	23.0	51.0	
	Normalized and tempered	122,500	97,500	15.0	38.0	
Carbon-Molybdenum	Normalized and tempered	86,300	53,850	23.4	39.8	Industry
	Normalized and tempered	120,000	79,000	19.0	35.6	241	...	
	Normalized and tempered	77,500	48,000	30.0	56.2	168	...	
Vanadium	Normalized and tempered	90,000	60,000	20.5	33.3	Industry
	Normalized and tempered	79,000	46,250	28.0	55.0	
	Normalized and tempered	92,400	58,600	23.5	42.4	
Copper	Normalized and tempered	67,000	49,500	32.5	53.9	Greenridge-Lorig
	Normalized and tempered	96,250	61,750	22.0	35.0	119	47 ^b	
	Quenched	125,000	35,000	50.0	40.0	183	...	
Hadfield element	At room temperature.
Residual element	Charpy.

*Izod.

**Charpy.

**At room temperature.

*Residual element.

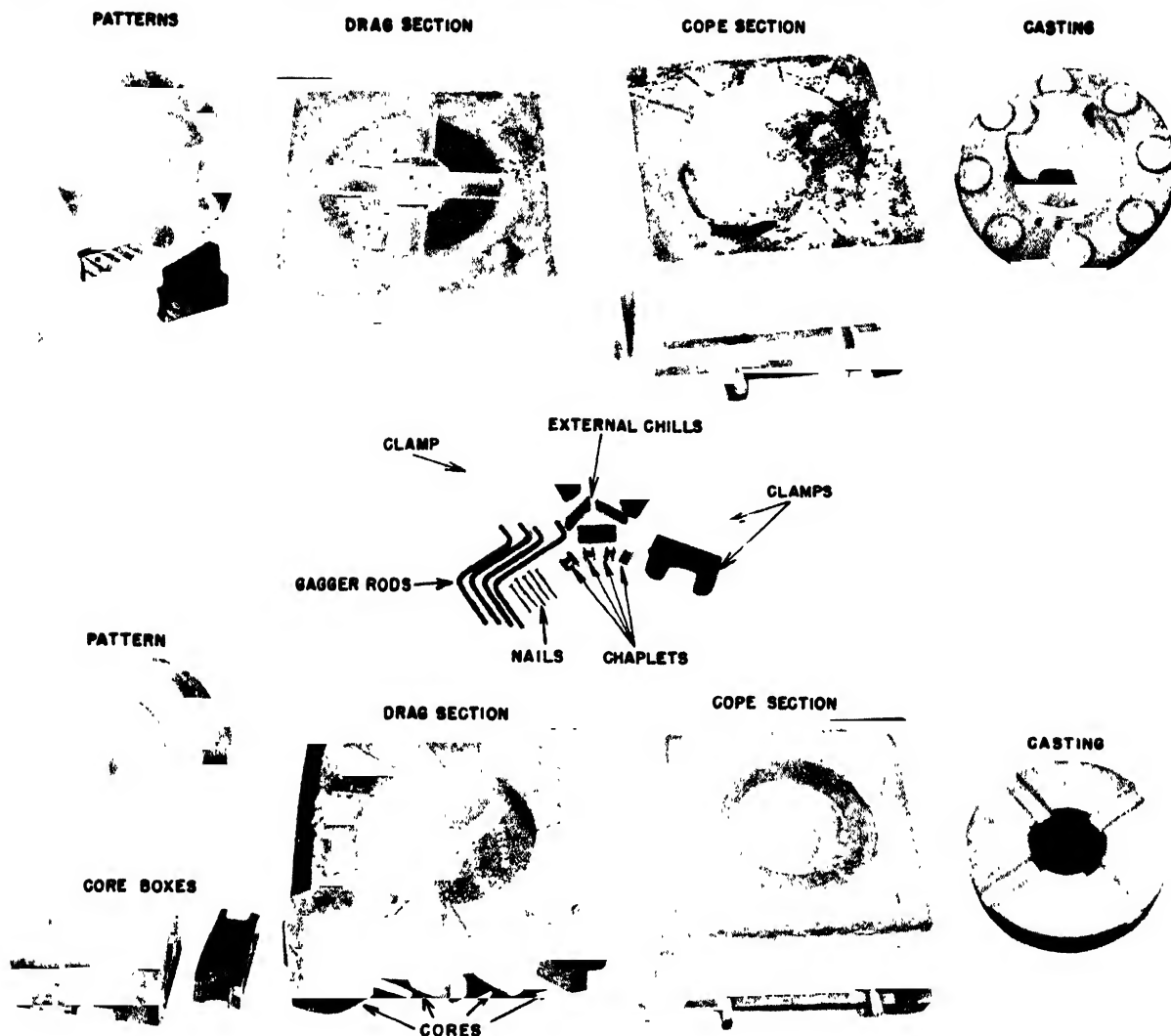


FIG. 18-1. Parts and accessories for preparing typical molds for two relatively small ferrous castings.

tensively for castings weighing a few ounces and a special operation known as the "lost wax" or "precision casting" process is based upon the use of such patterns; this process is described in more detail later in this chapter.

Other molding materials include **mold and core washes** which are sprayed, or swabbed, on the mold surfaces to make a smooth mold and resist metal penetration, **gaggers** or reinforcing rods which are placed around the pattern in the sand to add strength to the mold, and **chills** of various types are used to promote directional solidification and reduce the effect of severe temperature gradients caused by sharp changes in metal sections of the casting. **External chills** are placed directly on the pattern and the chill is flush with the mold wall when the pattern is removed from the sand. **Internal chills** are placed in the mold after the pattern has been removed. Internal chills are often more effective than external chills, but they must be positioned with the utmost discretion. **Chaplets** and stem anchors are used within the mold cavity to support an internal core. **Nails** frequently are inserted in the mold surface to prevent metal penetration and sand erosion. Rolled-steel,

cast-steel or wooden frames, commonly called **flasks**, are used to hold the molding sand around the pattern.

Flasks are made in two sections—the lower half is called the **drag** section; the upper half is called the **cope** section. Occasionally, there is need for a third section which is placed between the cope and the drag. This is known as the **cheek** section. The bottom of the flask is termed the **bottom board** or **bottom plate**. It is important that the cope and drag sections match properly. With small or medium flasks, this is accomplished by hardened pins placed on the outside of the drag section; with larger flasks, this matching is performed with guides, peep-sights, match blocks and many other ingenious devices. The flask sections and the bottom board are held together by "C" clamps and wooden or steel wedges. Figure 18-1 shows patterns, cope and drag sections of two different molds, and accessories required for one of the molds, and the finished castings produced with this equipment.

Making the Mold—The size and shape of the casting are the controlling factors in deciding how the molten steel should be introduced into the mold and where to locate gates, risers and vents. A flask is selected which

is sufficiently larger than the pattern to provide room between the pattern and the flask wall to accommodate at least several inches of sand and the gate system. The flask also must be sufficiently large to permit placement of the **feedheads** or **risers** which are attached to the pattern and leave spaces in the finished mold to serve as reservoirs for molten metal that supply extra metal to feed the voids formed by shrinkage as the metal cools and passes from the liquid to the solid state.

In preparing the mold for a typical steel casting of relatively small size, the bottom board or bottom plate is clamped securely to the flange of the drag section of the mold as shown in Figure 18—1. The pattern is set on the bottom plate in such a position that sand may be rammed over the top of the pattern. The amount of sand must be sufficient to prevent metal runoff, to develop the required strength to resist ferrostatic pressure, and to permit handling of the mold. **Facing sand** is riddled or sifted over the surface of the pattern to a depth of one inch and packed in any pockets and around the corners of the pattern. **Heap-sand**, which is nothing more than used facing and core sands, sometimes re-bonded, is rammed into the flask to a depth of four inches. Pneumatic air hammers may be used for this purpose, and many foundries use sandslingers. More heap-sand is added to a depth of six to eight inches, or until the flask is full and, after ramming, the excess sand is removed by a straight-edge. This operation is known as **striking off**. A bottom plate is clamped over the top of the flask and the entire flask section is rotated or turned 180°. The first bottom board is now removed and the cope section of the pattern is placed on the drag section. The two parts of the pattern are matched properly by dowel pins fitted into holes located in the face of the **parting line**, joint line, or split sections of the pattern. The cope section of the flask, which is reinforced with cross bars, is set on the drag section and fitted properly, as described. The cross bars in the cope section also serve to support gagger rods which reinforce the sand, thereby preventing possible distortion of the mold cavity by pressure of the metal entering the mold.

A finely ground sand, known as **parting sand**, is dusted on the face of the drag section to prevent cohesion of the sand in the cope section with the sand in the drag section. **Riser patterns** are placed at the desired locations on the casting pattern and a **sprue stick** or **gate tile** is placed upright on the sand surface of the drag near the point where the metal is to enter the mold cavity. Facing sand is then riddled over the pattern and packed firmly by hand. The gagger rods are placed and heap sand rammed into the flask in the same manner as described for the drag section. After ramming is complete, the riser patterns and sprue stick, if used, are removed. The cope section is removed from the drag and the cope and drag patterns **drawn** from the sand by **lifting screws**. Pneumatic vibrators are sometimes attached to the patterns for the purpose of freeing the pattern from the sand. The mold is smoothed off and patched, and rough corners and edges are rounded. The gate is cut in the drag section from the base of the sprue stick to the mold cavity. Cores, and chaplets, if required, are properly placed as shown by core prints and markings on the pattern. Provision for elimination of mold and core gases is made by jabbing a rod through the mold wall or by scratching vents across the drag section of the mold at the parting line. Internal chills, if required, are placed at this time. External chills are placed on the pattern surface prior to adding the facing sand.

If the casting is to be what is known as a **green sand casting**, the cope section of the mold is placed on the drag section and clamped securely to the bottom board.

A runner cup, a sand mold having an internal shape similar to a funnel, is placed directly over the gate, and the mold is ready to pour.

If the casting is to be what is known as a **dry sand casting**, the mold cavity is sprayed or swabbed with a mold wash, and placed in a mold oven before the cores are set. Operating temperatures of **mold drying ovens** vary from 300-800° F and the time of drying may vary from 4 hours to 72 hours, depending upon the type of molding sand, the size of the mold, and the drying characteristics of the oven. After drying, cores and chaplets, if required, are placed and asbestos rope or putty is placed on the joint surface for a seal. The cope section then is fitted properly on the drag section and, after placing the runner cup, the mold is ready to receive the molten metal.

Machine Molding—Fundamentally, machine molding methods differ little from the process previously described for the manual or floor molding operation, the chief difference being that the ramming of the sand and the removal of the pattern from the sand are performed by machine. Also, there are details of pattern construction, such as integral gates and, frequently, attached risers, which eliminate some of the work formerly done by hand. However, the placement of cores, patching and finishing of the mold still remain as the chief function of the molder.

A relatively new molding method known as "shell molding" lends itself to the production of molds by machine methods. This method of molding is described briefly in Section 2 of this chapter. It is especially adapted to production of large numbers of small repetitive castings.

Cored Molds for Hollow Castings—Many castings are designed with overhanging flanges, ribs, bolt holes, bosses and hollowed-out sections. Such castings, because of their irregular shape, cannot be produced simply by making a mold, and the foundryman must resort to the use of cores to meet the demands of the design. A core is nothing more than a solid shape made of sand. Sand is rammed either by hand or machine, or blown into a **core box**. When the core box is filled with sand and the excess sand "struck off" with a straight edge, it is turned over and the box lifted from the core thus formed. The ramming operation includes placement of reinforcing rods, or wire, for strengthening purposes. Vent rods are jabbed through the core to permit escape of gases. Sometimes wax wire is rammed in with the sand and, upon baking, the wax wire melts, thereby forming a passageway through which mold gases escape. As soon as the core is vented and sprayed with core wash, it is placed in an oven to bake. The baked core should be hard, strong, and smooth, but it should be sufficiently collapsible at high temperature in order not to cause the casting to "hot tear," i.e., pull apart due to contraction as the metal cools. Appendages attached to the molder's pattern permit the cores to be placed at the proper locations in the mold. These appendages are known as **coreprints** and have the same dimensions as the inside dimensions of the core box. The core prints are of sufficient length to provide a good bearing surface to support the core. It is sometimes necessary to tie the cores in the cope section of the mold or to use chaplets and stem anchors to provide additional support.

Cores are made by hand and by various types of molding machines, including sandlinger and rollover machines. For high production work, core blowers are used extensively. Core baking and drying operations require considerable attention and much use is being made of dielectric heating methods, as well as auto-

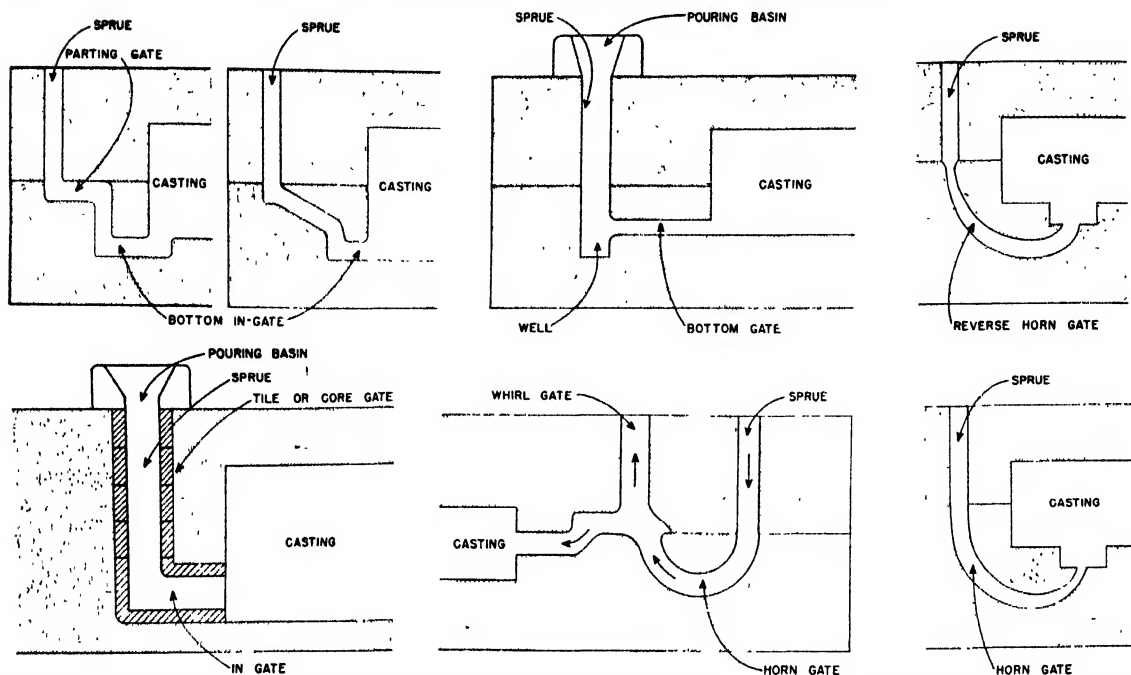


FIG. 18-2. Schematic vertical sections through various molds, illustrating types of gates frequently employed (After Briggs).

matically controlled oil- and gas-fired ovens. Perhaps one of the greatest problems in the foundry is the selection and development of core sands to fill the specific needs of the various types of castings. Such problems are so varied and of such importance that specialists are employed in many of the foundries to study sands and sand compositions.

Gates, Risers and Vents—The gate, as mentioned previously, is the channel or passageway through which the metal flows in filling the mold. The height, the cross-sectional area, and the shape of the gate, as well as the point at which it enters the mold cavity, are all important factors in any gate system. Many defects in steel castings, such as imbedded sand, cracks, shrinkage cavities, core failures, and misrun castings are attributed directly to poor gating practice. A gate, in order to function properly, must permit the flow of the metal into the mold with the least amount of turbulence. It must carry sufficient volume of metal with enough pressure to fill the mold quickly, and it must be arranged to permit proper distribution of temperature gradients.

There are four general types of gates, (1) the bottom gate, (2) the parting gate, (3) the top gate, and (4) the step gate which is a combination of the three aforementioned types. The bottom gate is used most commonly for large floor-molded castings. The parting gate is favored for smaller work and is used on practically all machine-molded castings. There are many modifications of each of the four types, such as horn gates, pencil or finger gates, swirl or whirl gates, skimmer gates, shower gates, ring gates and strainer gates. Figure 18-2 shows types frequently used.

There are two types of risers; (1) the open riser, and (2) the blind riser. Open risers are attached to the surface of the casting at some location on the cope side and extend through the sand to the surface of the mold. The size of the riser depends upon the size of the section to be fed and upon the design of the casting. In general, the cross section of the neck of the riser should be equal to the section thickness of the casting at the point of at-

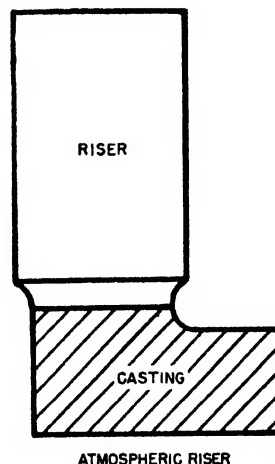


FIG. 18-3. Schematic representation of an open or atmospheric type feedhead or riser (After Briggs).

tachment. The body of the riser should be slightly larger than the neck and the height at least 1.5 times its diameter or width (see Figure 18-3).

Blind risers usually are attached to the drag side of the casting and are covered completely with sand. The neck thickness should be $1\frac{1}{2}$ to 2 times the sectional thickness of the casting at the point of attachment. The diameter of the riser body should be 2 to $2\frac{1}{2}$ times the sectional thickness at the point of attachment. The height of the riser should be about 1.5 times the diameter and the topmost surface rounded off or dome shaped. In order to insure proper feeding action of blind risers, a sand core or carbon rod should be inserted in the top section. Also a vent or pop riser should extend from the top of the riser to the surface of the mold. The use of the core in blind risers is patented and is known as the

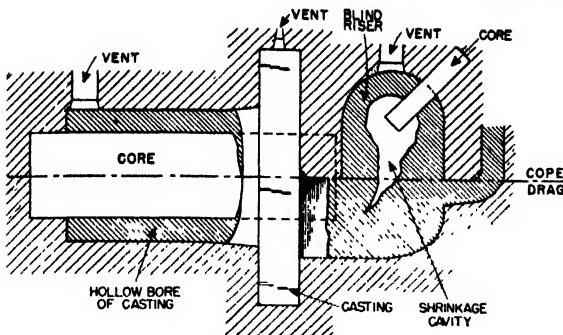


FIG. 18-4. Vertical section of a mold employing a Williams core in conjunction with a blind riser (After Briggs).

Williams method. The core serves to admit atmospheric pressure, thereby increasing feed efficiently. Figure 18-4 is a sketch of a Williams core on a blind riser.

Risers placed on steel castings present a serious problem to foundrymen from the standpoint of cleaning, because all risers must be removed prior to shipment. For this reason the Washburn core or necked-down riser is finding considerable use in conjunction with both open risers and blind risers. The necking core is nothing more than a core through which a small opening connects the riser to the casting. The thickness of the core and the size of the opening depend upon the size of the riser required. The necking core has the advantage that risers can be removed easily by sledge hammers, whereas oxyacetylene torches are necessary to remove conventional risers. Figure 18-5 illustrates use of a necking core with a blind riser.

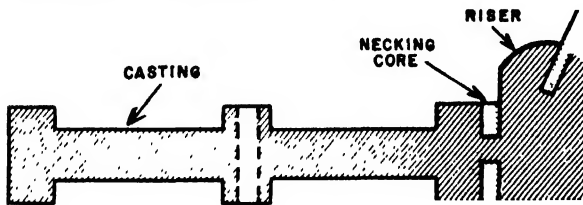


FIG. 18-5. Washburn necking core used to neck down the riser to facilitate removal, applied to a blind riser using a Williams core. (After Briggs).

During the teeming process many mold gases are formed by volatilization and burning of sand binders and by the expansion of the cool air within the mold cavity. Provision for the exhaust of such gases is made by jabbing wires through the mold to the surface or by the use of very small risers. The small holes thus formed are called vents and usually are located over points of the mold cavity where trapped gas might be expected, such as pockets and high points, or at flanges where a blind riser is used in place of an open riser. The vent rod preferably should be flat in shape rather than round. Some foundrymen contend that the practice of venting is unnecessary and results in excessive cleaning costs. However, the practice still is carried on in most foundries.

STEEL CASTING AND FINISHING OPERATIONS

Casting—There are two classes of castings, static castings, and centrifugal castings. The discussion, so far, has dealt only with static castings, i.e., castings which depend upon proper molding practice, atmospheric

pressure and gravity to form castings free from internal shrinkage cavities and other defects. The second class, centrifugal castings, as the name implies, makes use of centrifugal action to perform the function of gravity in static casting. There are two types of centrifugal castings—the horizontal type and the vertical type. In the horizontal type, the mold rotates on a horizontal axis. This method is used principally for making long, concentric, hollow castings with uniform wall thickness, where risers are not required. Tubing, pipe, gun barrels, bushings, sleeves, etc., are typical castings made by this process. The vertical method, employing rotation about a vertical axis, is essentially a pressure-casting method and depends, in part at least, on gravity, atmospheric pressure, and proper location of riser or risers to produce sound castings. Gears, piston rings, impellers, propellers, turbine diaphragms, etc., with sections too thin to be cast by static methods or where exceptional feeding problems arise which cannot be surmounted in any other way, are made by the vertical centrifugal process. There are several advantages to be gained from centrifugal casting methods: (1) castings are said to be sounder and have fewer inclusions, (2) fewer cleaning problems, and (3) a 10 to 30 per cent increase in yield.

Shaking Out, Cleaning, Finishing and Testing—After the castings have been poured and sufficient time allowed for solidification and cooling, they are shaken out, i.e., the castings are removed from the flask and sand. This operation is performed by placing the entire flask on a vibrating grid called a shakeout machine. The clamps are removed from the flask sections and the vibration of the machine causes the sand to fall loose and free of the castings. The castings are then removed to the cleaning room where they are subjected to blast cleaning or put through a tumbling barrel, where the abrasive action of sand particles or metallic shot impinging upon the walls of the castings loosens and removes adhering sand. Risers and gates are removed by oxyacetylene torch, electric arc, abrasive or friction saws, or by sledge hammers. Fins and surface defects found during preliminary inspection are removed as far as possible by chipping and burning. Within recent years, the powder-injection acetylene torch has been applied successfully to the removal of burnt-on sand. The riser necks, gate stubs, and slight surface imperfections are dressed by grinding wheels of various types and sizes. Repairs by welding also are performed sometimes, depending upon the carbon and alloy content of the steel. Castings containing more than 0.30 per cent carbon or of equivalent hardness that require repairing by welding are welded prior to heat treatment. After heat treatment, the castings are subjected to final surface-finishing operations—chipping, grinding, or sand blasting—as required. Finally, they are inspected for surface defects and dimensional accuracy. This inspection may include examinations utilizing X-ray, gamma-ray, magnetic-particle, fluorescent-penetrant, or ultrasonic methods.

HEAT TREATMENT OF STEEL CASTINGS

In the "as cast" or "green" state, steel castings are relatively brittle and possess poor mechanical properties. It follows, therefore, that in order to render them serviceable they must be subjected to a heat treatment which will refine the grain and break up the dendritic structure, relieve internal stresses, and develop the desired physical and mechanical properties. The heat treatment applied to steel castings depends upon the chemical composition, section size, design, grain size,

and the desired mechanical properties. Because of the variations in size, section and design of castings, their heat treatment requires considerable care. However, steel castings respond to heat treatment similarly to wrought steel products and, for all practical purposes, the same rules apply, with care being exercised in placing castings in the furnace in such a way that there is minimum of danger of warpage, distortion or cracks which may cause them to be unfit for service. The common heat treatments applied to steel castings are as follows:

Annealing—The castings are placed in a furnace and heated slowly to a temperature slightly above the A_c point, usually 1650°F for carbon steels. The castings are held at that temperature 1 hour per inch of thickness of the heaviest section and cooled slowly in the furnace. Such treatment relieves casting stresses, refines the grain, and serves to eliminate the dendritic structure. Annealing raises the tensile and yield strength and increases ductility. It also improves machinability, especially of high-carbon steels.

Normalizing—The normalizing treatment is similar to the annealing process, except that the castings are removed from the furnace at the end of the soaking or holding time and allowed to cool in still air. The normalizing treatment produces a harder steel with higher yield and tensile strength than the annealed product, with ductility value approximately the same. However, internal stresses are not removed to the same extent as in the annealing process. Double normalizing is often employed to produce a more uniform grain structure and such treatment improves the ductility. To remove internal stresses or to reduce the hardness of normalized steels, a draw or tempering treatment is used frequently, i.e., heating the casting to a temperature below the critical range and allowing it to cool in the furnace. Tempering temperatures for steel castings range from 500 to 1250°F .

Quenching and Tempering—The quenching and tempering treatment is confined principally to high-carbon and alloy-steel castings where high strength and resistance to impact and/or abrasion is required. The general practice is to anneal or normalize the castings, reheat and quench. The tempering treatment should follow immediately, because internal stresses set up by the quench may cause the castings to crack. Sometimes, it is necessary to use a time quench to eliminate cracking, in which case the casting is immersed in the quenching bath for a predetermined time interval, removed, and subjected to the tempering treatment immediately. Such procedure must be controlled closely; otherwise, wide variations in hardness values may result. The end-quench hardenability test is of great value in determining quenching and tempering procedure for steel castings.

Some alloy steels, particularly straight manganese steel in the range of 1.00 per cent up to 2.00 per cent, are susceptible to temper brittleness. Such steels have low impact and ductility values when cooled slowly from the tempering temperature. To avoid this, it is sometimes necessary to cool quickly from the tempering temperature by quenching or air cooling.

Flame Hardening—It is sometimes desirable that castings be differentially hardened, i.e., parts of the casting subject to extreme wear or abrasion should be harder than another part of the same casting which requires a machining operation. A pinion gear, for example, should have hard, wear-resistant teeth with a machinable bore. In such cases, the castings first are annealed or normalized and then only the surfaces to be hardened are heated to the hardening temperature by

a torch or induction-heating apparatus. The heated parts are quenched in water. A time quench often is employed to prevent critical internal stresses. Sections can be hardened up to $\frac{1}{4}$ inch in depth by the flame hardening process.

HEAT- AND CORROSION-RESISTANT STEEL CASTINGS

Highly Alloyed Steels—Normally, high-alloy steel castings contain at least 12.0 per cent of alloying elements, such as chromium, nickel, cobalt, copper, molybdenum and tungsten, although 4 to 6 per cent chromium steel is generally considered in this class. Various alloy combinations have been developed by consumers and manufacturers for specific applications where conditions of heat and corrosion are critical. There is no clear line of demarcation between heat-resistant castings and corrosion-resistant castings since corrosive media, in the form of fumes and exhaust vapors, usually accompany high temperatures. Moreover, the so-called stainless or corrosion-resistant castings possess excellent mechanical properties at high temperatures. It is not uncommon, therefore, to see castings of a composition designed to resist corrosion being used for heat-resistant applications, and vice versa.

Typical Applications—Heat- and corrosion-resistant castings serve a very broad field. Mine pumps, impellers, fans, valves, valve trim, tubing, sleeve nuts, return bends, agitators, retorts, stills, vessels, digesters, thimbles, nozzles, and centrifuges, are but a few of the castings being used in chemical plants for corrosive applications. The explosives manufacturers, oil and petroleum refineries, paper mill plants, nylon and fabric manufacturers, food and medicinal producers, and the mining and ore-concentrating industries use many corrosion-resistant castings. Heat-resisting alloy castings are used in practically every installation where temperatures exceed 1200°F or where alternate heating and cooling cycles are employed; typical examples are chain conveyors, blower tubes, carburizing trays, recuperators, heat exchangers, hearth plates, grate bars, dampers, retorts, rolls, furnace doors, skid bars, carrier blades, muffles, safety sleeves, fractionating towers, normalizing shafts, guides, piercer points, and tube supports.

In the past, the optimum chemical compositions were determined largely by trial and error methods, but in recent years scientific studies and tests conducted by the Alloy Casting Institute and other interested parties have done much toward the standardization and proper use of the various grades of high-alloy steels. Table 18—II shows the compositions of some of the important grades of heat- and corrosion-resistant steels used for castings.

Melting—Steels for heat- and corrosion-resistant castings are melted in arc-type electric furnaces and high-frequency electric induction furnaces. The direct-arc electric furnace is used more extensively, but modern developments in induction melting equipment make it an extremely desirable unit for melting alloy steels for metallurgical, as well as operating, reasons. Furnace linings may be acid or basic and, in the case of the induction furnace, the lining may be neutral. The basic-lined furnace has the advantage of greater alloy efficiency and makes possible the use of chrome ore and nickel oxide for chromium and nickel additions. Also, it is possible to oxidize and reduce heats containing alloy scrap without serious loss of expensive alloys. On the other hand, acid furnaces require less skill to operate, they are faster, and the refractory costs are lower. Melting costs of induction furnaces compare

Table 18—II. Standard Designations and Chemical Composition Ranges of Heat- and Corrosion-Resistant Steels for Castings (Per cent)*

CORROSION RESISTANT								
Alloy Casting Institute Type	C (Max.)	Mn (Max.)	Si (Max.)	P (Max.)	S (Max.)	Cr	Ni	Other
CA-15	0.15	1.00	1.50	0.04	0.04	11.5-14.0	1 max.	Mo: 0.5 max.
CA-40	0.20-0.40	1.00	1.50	0.04	0.04	11.5-14.0	1 max.	Mo: 0.5 max.
CB-30	0.30	1.00	1.00	0.04	0.04	18.0-22.0	2 max.	0.90-1.20 Cu optional
CC-50	0.50	1.00	1.00	0.04	0.04	26.0-30.0	4 max.
CE-30	0.30	1.50	2.00	0.04	0.04	26.0-30.0	8.0-11.0
CF- 8	0.08	1.50	2.00	0.04	0.04	18.0-21.0	8.0-11.0
CF-20	0.20	1.50	2.00	0.04	0.04	18.0-21.0	8.0-11.0
CF- 8M	0.08	1.50	1.50	0.04	0.04	18.0-21.0	9.0-12.0	Mo: 2.0-3.0
CF-12M	0.12	1.50	1.50	0.04	0.04	18.0-21.0	9.0-12.0	Mo: 2.0-3.0
CF- 8C	0.08	1.50	2.00	0.04	0.04	18.0-21.0	9.0-12.0	Co: 8xC Min., 1.00 max.
CF-16F	0.16	1.50	2.00	(a)	(a)	18.0-21.0	9.0-12.0	(a)
CG-12	0.12	1.50	2.00	0.04	0.04	20.0-23.0	10.0-13.0
CH-10	0.10	1.50	2.00	0.04	0.04	22.0-26.0	12.0-15.0
CH-20	0.20	1.50	2.00	0.04	0.04	22.0-26.0	12.0-15.0
CK-20	0.20	1.50	2.00	0.04	0.04	23.0-27.0	19.0-22.0
CN-7M Cu	0.07	1.50	(b)	0.04	0.04	18.0-22.0	21.0-31.0	May contain Mo and Cu.

HEAT RESISTANT

Alloy Casting Institute Type	C	Mn (Max.)	Si (Max.)	P (Max.)	S (Max.)	Cr	Ni	Mo (Max.)
HC	0.50 max.	1.00	2.00	0.04	0.04	26.0-30.0	4.0 max.	0.5
HD	0.50 max.	1.50	2.00	0.04	0.04	26.0-30.0	4.0- 7.0	0.5
HE	0.20-0.50	2.00	2.00	0.04	0.04	26.0-30.0	8.0-11.0	0.5
HF	0.20-0.40	2.00	2.00	0.04	0.04	18.0-23.0	8.0-12.0	0.5
HH	0.20-0.50	2.00	2.00	0.04	0.04	24.0-28.0	11.0-14.0	0.5
HI	0.20-0.50	2.00	2.00	0.04	0.04	26.0-30.0	14.0-16.0	0.5
HK	0.20-0.60	2.00	3.00	0.04	0.04	24.0-28.0	18.0-22.0	0.5
HL	0.20-0.60	2.00	3.00	0.04	0.04	28.0-32.0	18.0-22.0	0.5
HT	0.35-0.75	2.00	2.50	0.04	0.04	13.0-17.0	33.0-37.0	0.5
HU	0.35-0.75	2.00	2.50	0.04	0.04	17.0-21.0	37.0-41.0	0.5
HW	0.35-0.75	2.00	2.50	0.04	0.04	10.0-14.0	58.0-62.0	0.5
HX	0.35-0.75	2.00	2.50	0.04	0.04	15.0-19.0	64.0-68.0	0.5

* Based on: "Steel Castings Handbook," Steel Founders' Society of America (1950).

(a) For free-machining properties, suitable combinations of Se, P, Mo and S may be used.

(b) Silicon content varies for different proprietary alloys.

favorably with acid- or basic-lined arc furnaces, and chemical composition and metal temperatures can be controlled within closer limits. Moreover, alloy recoveries are said to be higher. The induction furnace is limited to the dead-melt process but with present developments in flushing inert gases, such as argon and nitrogen, through the bath to remove dissolved gases, this does not pose a serious disadvantage.

Regardless of the type of melting unit, strict metallurgical control is necessary to produce high-alloy steels. Care must be exercised in the purchase of ferroalloys and other raw materials, and scrap must be segregated according to composition and used with utmost discretion. Because most furnaces used for melting alloy steels for foundry use are small, seldom exceeding three tons' capacity, heats are made rapidly and time is not available to run control analyses. For this reason, complete control of furnace operation and all raw materials and scrap is necessary.

Casting—High-alloy castings are made in static molds and centrifugal molds of the horizontal or vertical type. By far the greater tonnage is produced in static molds by conventional molding methods. However, a large number of horizontal centrifugal castings are made in the form of tubing, retorts, rolls and rollers. For work of this type, the centrifugal method is of great advantage since casting yield approaches 90 per cent, no cores are required, and the molds may be made of sand or, in some instances, cast iron sprayed with a highly refractory material. The vertical centrifugal method of casting is used only when static methods fail and the castings cannot be made any other way. Impellers containing thin vanes, for example, or castings designed in such a manner that they cannot be properly fed, are made by the vertical centrifugal method.

Molding—Fundamentally, the molding procedure for high-alloy castings is the same as that employed for carbon-steel castings. However, greater skill and more

precision is required on the part of the molder because excess stock on castings necessitates expensive machining operations. Every effort must be expended to produce clean, smooth, sound castings. The molding sand must be rigidly controlled and the particle size and bonding must be such that details of the pattern are reproduced accurately. The risers and gates should be located so as to avoid internal defects and center-line shrinkage, for such defects in castings exposed to high temperatures or corrosive media cause premature failures. The use of internal chills should be avoided, but if the casting problem cannot be surmounted in any other way, the chill material should be of the same chemical composition as the metal in the casting. Likewise, stem anchors and chaplets should be used sparingly and with care. Chill nails should never be used on high-alloy castings because contamination of the surface of the casting by the steel nail head may initiate corrosion and ultimately cause failure.

Finishing Operations—Cleaning and finishing operations for high-alloy steel castings differ but slightly from ordinary steel castings. Shot blasts or other abrasive cleaning devices must contain high-alloy shot or what is called high-alloy sand, since steel shot causes the castings to corrode. Risers and gates are removed by burning with an electric arc, although abrasive or friction saws are sometimes used for this purpose. The riser and gate pads are removed completely by grinding and all surface imperfections must be smoothed off by grinding. Frequently, alloy castings are ground to template. Casting repair by welding is limited strictly to minor defects and when welding is permitted, a rod of the same chemical composition as the casting generally is used. Heat treatments vary according to compositions, and range from carbide-solution treatments for corrosion-resistant castings such as 18 Cr-8 Ni, to precipitation-hardening treatments for the high-chromium, low-nickel-molybdenum steels. Steels containing 11.5 to 14.0 per cent chromium respond to the conventional hardening and tempering treatments.

Heat-resistant alloys, such as 24 Cr-12 Ni, are sometimes treated to improve machinability, in which case they are heated to approximately 1600° F and slowly cooled. The purpose of the treatment is to precipitate carbides.

Before shipment, castings are checked thoroughly for dimensional inaccuracies and surface imperfections and finally are cleaned in a sand blast or similar equipment.

Methods of Sampling and Testing—Sampling and preparation of samples for chemical tests have become standardized fairly well in all foundries. In general, test specimens for chemical analyses are poured from the ladle at about the mid-point of the teeming operation so that the sample is representative of the entire heat. The sample is cleaned and drilled; the first ¼ inch of drillings is thrown away as insurance against contamination. The chemical composition is then determined from subsequent drillings according to proven analytical procedures. Heat numbers are stamped on the castings and test specimens, thereby making possible the identification of the chemical composition of the castings. Most foundry laboratories are equipped to determine completely and accurately the compositions of steels, ferroalloys, sands, and various raw materials.

Mechanical properties of cast steels are determined from test coupons or test lugs cast integrally with the castings. At times it is necessary, by reason of the size or design of the casting, to make separate test coupons which are identified properly with the castings they represent by heat numbers stamped on the castings and coupons. The test bars always are heat treated with the castings, machined and tested in accordance with standard mechanical-testing procedure. Test specimens cut from castings sometimes show inferior mechanical properties when compared to standard coupon specimens. This is due principally to the mass effect, although in some instances it may be attributed to improper foundry technique, such as inadequate feeding action. Generally, results of mechanical tests as performed in the foundries may be considered indicative of the quality of the cast-

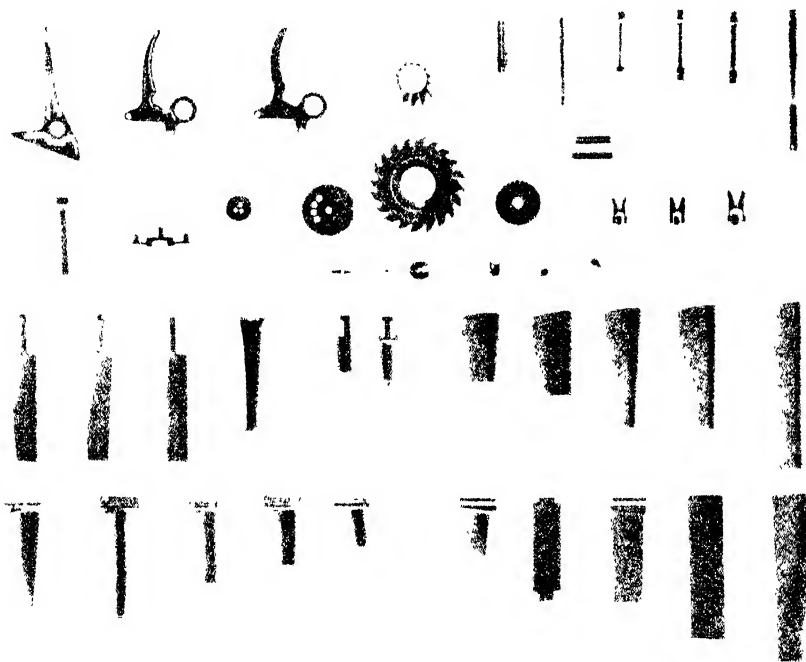


FIG. 18-6. Typical products produced by the lost wax or investment molding process for precision casting of materials not readily machinable that cannot be formed by hot or cold working. (Courtesy, Westinghouse Electric Corporation.)

ings, but it should be recognized that excellent mechanical properties shown by tests do not preclude the possibility of poor mechanical properties in a casting.

Many foundries make use of modern non-destructive testing equipment, such as X-ray and gamma-ray apparatus. Also, an ultrasonic method for detecting internal defects and magnetic particle testing for detecting surface imperfections, such as cracks and small sand inclusions, is used extensively. In the case of high-alloy steels of the austenitic type which are non-magnetic, a method which makes use of a fluorescent penetrant and black light, is used to indicate cracks. Destructive tests in which castings are machined to complete destruction are employed frequently to determine the internal soundness of castings. Such tests are used as control measures and serve to establish foundry technique for subsequent castings. The microscope has been used in foundries for many years as a control tool to establish heat-treating practices and for other related metallurgical investigations.

PRECISION STEEL CASTINGS

The lost wax or investment molding process of making castings is often referred to as "precision casting." Castings weighing one pound or less can be made to tolerances of 0.005 to 0.010 inch. Larger castings weigh-

ing as much as 10 pounds have been made successfully by this method. The process is restricted largely to high-alloy steel parts which cannot be machined readily and which cannot be formed by hot or cold working methods. Such parts include valve parts; turbine blades, buckets and nozzles; molds and dies for the plastic and ceramic industries; small gears for timers; hobs, milling cutters, magnets, jewelry, surgical and dental tools, etc. Figure 18-6 shows some typical precision castings.

The process consists of making a pattern of free-machining steel, around which a mold is formed of an alloy of low melting point. The mold thus formed is used for making wax patterns. The wax patterns are then used to make up molds similar to those used in the conventional molding process, except that risers are not required and the molding sand is bonded with an ethyl silicate, making a very hard and highly refractory mold when dried. The wax patterns within the mold are not lifted out of the sand but are removed by melting with steam or hot air; hence, the name "lost wax." The molds then are inverted over small induction furnaces containing the metal to be cast and clamped to the furnace. The furnace is rotated 180 degrees and the molten metal forced into the molds by air pressure, by centrifugal force, or by vacuum. The castings finally are cleaned and finished just as in conventional casting practice.

SECTION 2

IRON CASTINGS

Pig Iron for Castings—Castings are of innumerable kinds and uses, roughly grouped as **chilled-iron castings**, **gray-iron castings**, **alloyed-iron castings**, **malleable castings** and **nodular-iron castings**. In general, castings are made by mixing and melting together different grades of pig iron; different grades of pig iron and foundry scrap; different grades of pig iron, foundry scrap and steel scrap; or different grades of pig iron, foundry scrap, steel scrap and ferroalloys or other metals. In rare instances, molten iron direct from the blast furnace is run to a mixer, then to an electric furnace in which its composition is adjusted. In all remelting operations, the pig iron undergoes some change. Hence, physical properties of the pig iron itself are held subordinate to chemical composition in iron for remelting. However, by selecting iron of different grades and controlling the rate of cooling, the widest variations in mechanical properties may be obtained, from extreme hardness and brittleness with low impact resistance to extreme softness with, however, considerable strength and enhanced toughness. Thus, without the use of alloys and by selecting different malleable and foundry pig irons and controlling the rate of cooling, the following ranges in properties may be obtained:

Brinell Hardness Number	100 to 500
Tensile Strength (Lbs. per sq. in.)	10,000 to 60,000
Deflection (Transverse, in.)	0.04 to 0.36
Modulus of Elasticity (Lbs. per sq. in.)	12,000,000 to 29,000,000

Besides these properties, representing hardness, strength and elasticity, other important properties in iron for castings are listed as follows:

a. **Fluidity**, at time of casting, which depends upon composition and temperature. The melting and freezing points of pig iron and cast irons of eutectic and

hypoeutectic composition, for a given phosphorus content, vary inversely with the carbon and silicon, from 1990° F (1088° C) with 4.40 per cent carbon and 0.6 per cent silicon to 2280° F (1250° C) with 3.56 per cent carbon and 2.40 per cent silicon, the melting point varying inversely with the combined carbon for a given low (under one per cent) silicon content.

b. **Shrinkage**, which is the net result of contractions and expansions in cooling to atmospheric temperature, i.e., through the point where solidification begins, through the solidification range, and in cooling from this range to atmospheric temperature. Shrinkage depends upon the temperature of the iron as it enters the mold, the composition of the iron, the rate of cooling, and subsequent heat treatments. and it varies from $\frac{1}{32}$ to $\frac{1}{8}$ in. per ft.

The net volume change on cooling from the liquid state is incurred stepwise and is complicated by structural and other changes that take place, somewhat in the following sequence and at the indicated approximate temperatures.

1. Contraction in the liquid state—Tapping temperature to solidification (say 2200° F, corresponding to 1205° C).
2. Contraction, liquid-to-solid, solidification—Temperature nearly constant at freezing, largest change.
3. Contraction of austenite and ledeburite—2200° F (1205° C) to 2050° F (1120° C).
4. Expansion due to graphitization—2050° F (1120° C) to 1950° F (1065° C)—large.
5. Contraction due to cooling—1950° F (1065° C) to 1325° F (720° C).
6. Expansion—austenite to pearlite, gamma to alpha iron—1325° F (720° C).
7. Contraction due to cooling—1325° F (720° C) to room temperature.

- c. **Growth** is the tendency of castings to increase in volume after repeated heatings to temperatures between 850 and 1650° F, (455 and 900° C) in the absence of stress. It is usually measured in linear units and expressed in per cent. It varies from essentially zero to several per cent (there is an example of a reported extreme of 50 per cent) according to the time of heating or number of heating cycles and type of iron. For ordinary castings, 5 per cent is extreme, while a growth of 0.002 inch per linear inch is common, and depends upon composition and prior heat treatment. The causes of growth are believed to be (1) graphitization and (2) penetration of active gases into the discontinuities of the coarser-grained irons.
- d. **Creep** is the tendency to increase in length under stress (tension) at elevated temperatures above 700° F; the stress being but a fraction of that required to break a specimen of the material in a short-time tension test at room temperature. As it is expressed in different ways, one example is cited for an ordinary gray iron. In short-time tension tests, the breaking loads were 34,200 lb. per sq. in. at 70° F, and 35,100 lb. per sq. in. at 700° F; at a testing temperature of 700° F the test broke in 9 days under a load of 18,000 lb. per sq. in., and in 90 days under a load of 9,000 lb. per sq. in. Creep is lessened by certain alloy additions.
- e. **Porosity, density, and closeness of grain** are designations referring primarily to macroscopic structure, particularly of heavier sections, as revealed by fractured surfaces. A porous structure is conducive to weakness, and is indicative of a condition bordering on unsoundness, which term is applied when the structure shows more clearly visible blowholes or gas pockets, or small cavities due to bleeding or shrinkage. The term "density" is sometimes used loosely to designate grain size rather than mass per unit weight; thus, iron showing a fine-grained fracture is said to be a "dense" iron. Porosity may be due to segregation of low-melting constituents, to minute slaglike inclusions, or oxides that react with carbon to form gas while the casting is solidifying.
- f. **Machinability** (the capability of being machined) may be viewed from various standpoints. Most frequently it is considered to be the characteristic (of the metal being machined) which causes more or less wear on the cutting tool; less frequently the definition involves the power required for the machining operation; often it refers to the degree of smoothness obtainable on the machined surface. With due precautions to keep the casting free of sand or dirt, machinability is controlled through the composition and rate of cooling, but often must be sacrificed for some more essential property, such as strength or toughness.
- g. **Graphitization**, a phenomenon common to pig iron and cast iron, refers to the decomposition of iron carbide or, in any event, the rejection of elemental carbon in a casting after solidification has taken place, the carbon being liberated in the form of graphite which is usually found existing as minute, flaky particles disseminated throughout the casting. This property is controlled basically through composition of the iron, and is very important in castings, particularly malleable castings in which, upon reheating, carbon is rejected as nodules. As will be shown later, the formation of graphite is promoted by slow cooling, hindered or prevented by rapid cooling, and, if suppressed, can be brought about by subsequent heat treatment.

The effects of the different elements upon the properties of pig iron and cast iron will now be discussed.

IRON COMPOSITION vs. PROPERTIES

Forms of Carbon in Pig Iron—The factors influencing the carbon content of pig iron were discussed in Chapter 12. In considering the effect of temperature on molten iron, it is necessary to keep in mind that the effective temperatures are inevitably above a certain minimum of about 2085° F (1140° C), marking the freezing point of the iron-carbon eutectic. Referring to the iron-carbon diagram (see Index) and Figure 18-7, it will be observed that iron at 2370° F (1300° C) may absorb about 5.0 per cent carbon. Being a saturated solution, cementite (Fe_3C , 93.33 per cent Fe and 6.67 per cent C) crystallizes and separates as the liquid cools to 2065° to 2085° F (1130° to 1140° C), at which temperature we have the eutectic (lowest freezing) liquid, called ledeburite, containing 4.3 per cent carbon. Upon freezing, this eutectic liquid becomes a two-phase solid, one phase of which is composed of primary austenite (47.7 per cent) containing 1.7 per cent carbon in solution (equivalent to 12.1 per cent Fe_3C) and the other (52.3 per cent) of cementite. In the freezing of the eutectic, no drop in temperature occurs until heat equivalent to the heat of fusion of this eutectic is abstracted. Then, if the cooling is extremely rapid, practically all the carbon remains in these combined forms and the metal is extremely hard and brittle. But the saturated liquid solution may, in rapid cooling, produce a supersaturated solid solution. As another complication, the iron changes its allotropic form at about 1310° F (710° C) losing its power to hold carbon in solution, and if then the cooling is slow, some of the already formed cementite breaks down, or decomposes, to precipitate free carbon, which assumes the graphitic form, and usually is distributed as tiny flakes throughout the metal. The remainder of the carbon remains as combined carbon, part of which may be present as free or "proeutectoid" cementite, and some as pearlite, a fine lamellar aggregate of iron and cementite (so-called from its resemblance to mother-of-pearl). This last component is a microscopically laminated structure composed of alternate layers of nearly pure iron, called ferrite, and cementite, in the proportion of about 7 parts ferrite to 1 part cementite (about 0.80 per cent carbon). The proportion is by no means constant. The presence of pearlite in cast iron usually strengthens it without producing too great an embrittling effect.

Influence of Silicon—The silicon content of the iron is second only to its carbon content in regard to its effectiveness as a means for controlling the properties of the castings. As it is increased above 3.5 per cent, it makes the iron matrix more and more brittle, forming silvery iron, so that it is generally held to amounts between 0.5 and 3.0 per cent. Since it tends to throw carbon out of solution, as explained in the discussion of carbon in pig iron in Chapter 12, it is used as a "softener" in gray-iron castings, as a graphitizing agent in malleable castings or castings to be heat treated, and to regulate depth of chill in chill castings. With sulphur under 0.05 per cent, even one per cent silicon makes it difficult to obtain a chill, (an external zone of hard cementitic iron without appreciable graphite). Below one per cent, the chilling properties are roughly inversely proportional to the silicon present. Its effect in increasing the rate of graphitizing the combined carbon in white iron for malleable castings is remarkable. In a certain casting, for example, raising the silicon from 0.7 per cent to 1.0 per cent shortens the annealing cycle

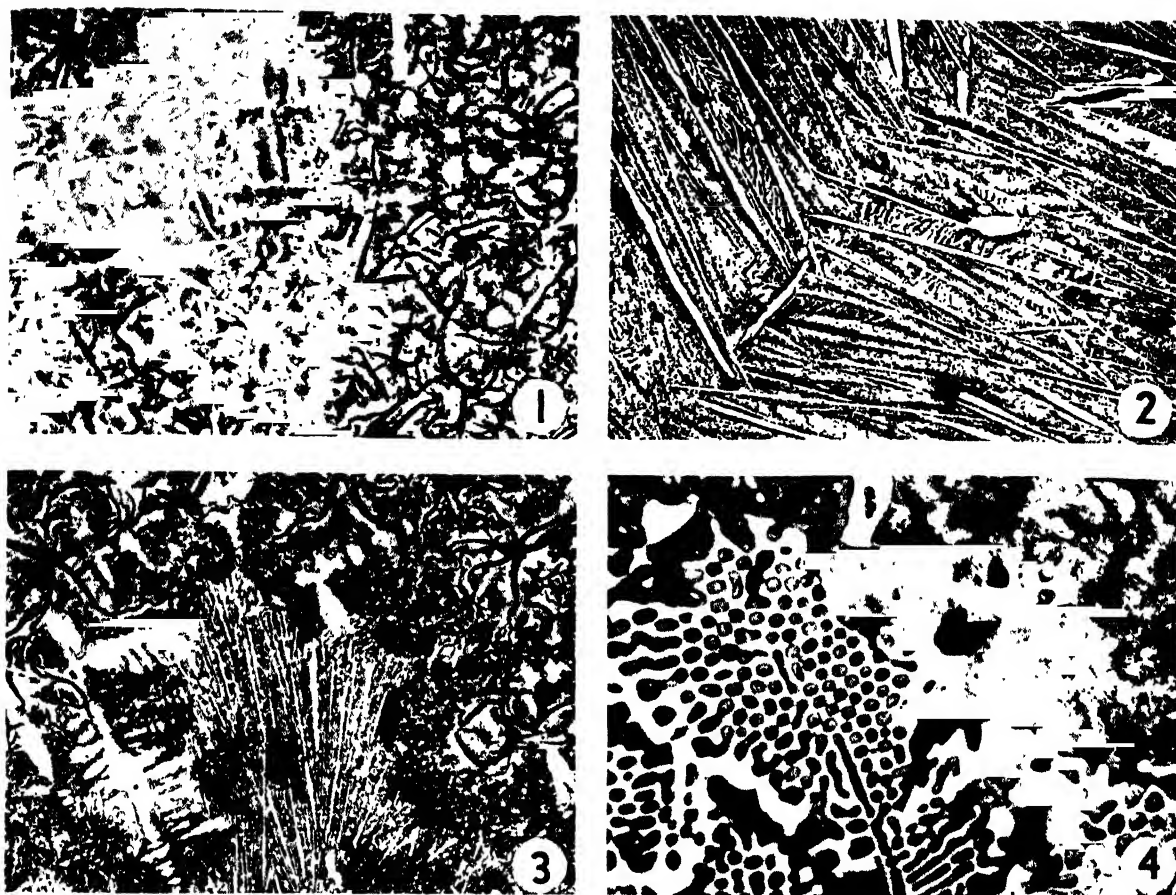


FIG. 18-7. Photomicrographs of typical microstructural constituents of cast iron as influenced by heat treatment and cooling rate. Sample 1: annealed specimen, showing graphite flakes in a fine pearlite matrix. Sample 2: "Chilled" sample showing ledeburite containing long needles and irregularly shaped areas of iron carbide (cementite; Fe_3C). Sample 3: Annealed specimen; region near chilled surface, showing areas of "chilled" ledeburite and graphite in a fine pearlite matrix. Sample 4: Annealed specimen; showing ledeburite and graphite flakes in a fine pearlite matrix. White constituent is cementite. All samples etched with picral. Magnifications: Samples 1, 2 and 3, $100\times$; Sample 4, $1000\times$.

from 180 hours to 72 hours. Silicon above 2 per cent hardens the matrix of iron solid solution to such an extent that machinability is adversely affected. Silicon added to white iron until it just turns gray increases the toughness by changing massive Fe_3C to pearlite and graphite, but more weakens it by forcing the pearlite ratio below 0.80 per cent carbon and increasing the graphite. Silicon tends to decrease shrinkage, to prevent blowholes, but increases the tendency to growth; silicon is oxidized in the cupola or air furnace and excessive amounts thus may favor a dirty casting, due to the entrapment of silicate-type inclusions. Silicon, from 1.5 up to 4.5 per cent, increases the resistance of the iron to atmospheric and acid corrosion, and more than 10 per cent greatly protects the metal from all forms of oxidation and from chemical attack.

Effects of Manganese—As to whether high manganese content has an overall good or a bad effect on cast iron, there is much difference of opinion, some considering it almost as a cure for all troubles and others condemning it as a source of much trouble, especially in chilled castings. While it tends to hold carbon in solution, iron in which chill is produced by increasing the manganese content alone is "soft" and tends to spall. In moderate amounts, it is said to prevent cracking of

the surface and also spalling to some extent, especially in chilled rolls, and it may harden the chill, if other conditions are right. It does tend to decrease blowholes, increase fluidity, and to neutralize the effect of the sulphur present. The amount used in castings varies from 0.1 to 2.0 per cent, 0.5 to 0.7 per cent being most common. In malleable castings, it is added in proportion to the sulphur, and according to the formula, $\text{Mn} = 2\text{S} + 0.15$. Held to these proportions, it will be found ultimately as MnS and in the cementite, and will not prevent graphitization of the cementite while larger amounts of manganese tend to stabilize cementite and increase growth. In high-grade gray-iron castings, the per cent manganese is held between 5 and 7 times the sulphur.

Influence of Sulphur—Sulphur in pig iron is considered now to be less injurious than was thought formerly. As sulphur, except for special purposes (e.g., enhanced machinability), is undesirable in steel, and as the blast furnace affords the only positive and economical means of reducing it, pig iron containing less than 0.05 per cent is preferable for making steel by all the fusion processes. In castings, it is varied from about 0.04 per cent to 0.20 per cent, though in irons for foundries it will seldom exceed 0.1 per cent. Iron melted in

cupolas always takes on sulphur from the coke, the percentage sometimes being doubled. Sulphur with iron forms sulphide, which is soluble in the liquid metal and has a melting point that is lower than the other constituents of the iron. This sulphide in iron used for castings has a three-fold influence. First, it tends to hold the carbon in combined condition, hence, can be used to increase the depth of chill in chilled castings, but in malleable castings and other castings that are to be heat treated, it thus retards graphitization, if it is not fully neutralized with manganese, and may be very undesirable. Chill produced with sulphur may be very brittle; low-silicon iron containing between 0.2 and 0.4 per cent sulphur often cracks when cooled rapidly. Second, its low melting point causes it to segregate as the iron solidifies, thereby causing the condition in castings known as bleeding. Third, it increases the shrinkage of the iron to a marked degree, thus increasing the difficulty of making accurate castings and increasing the tendency to form cracks, which are a result of the high shrinkage.

Influence of Phosphorus—Since compounds of phosphorus present in the materials charged into the blast furnace are completely reduced, all the phosphorus in the raw materials is ultimately found in the metal. Therefore, its content must be regulated by proper selection of raw materials. High phosphorus content causes a slight brittleness in pig iron and markedly reduces the total carbon content. Ferrophosphorus containing about 24 per cent phosphorus is almost carbonless and melts between 2245 and 2310° F (1230 and 1265° C). The melting point range for the 17 to 19 per cent grade is 2426 to 2498° F (1330 to 1370° C). Lesser amounts permit a proportionate increase of carbon, so that the total carbon in an iron containing 0.20 per cent phosphorus may be as high as 4.25 per cent. In this respect its action is not selective, since the ratio of combined to graphitic carbon is not affected. Phosphorus is known to form a compound, Fe_3P , with iron, containing 15.6 per cent phosphorus, but it apparently is able to combine or alloy with it in any proportion up to 25 per cent.

In pig iron and cast iron, Stead found that phosphorus forms a ternary eutectic solution containing 91.19 per cent iron, 6.89 per cent phosphorus, and 1.92 per cent carbon, to which Sauveur has given the name steadite. The amount formed in pig iron depends upon the phosphorus present, and since steadite has a low fusion point, the influence of phosphorus is to lower the freezing range of the iron. Another effect is to decrease the pearlite, so that more than one per cent phosphorus decreases the strength of the casting rapidly, the maximum strength being obtained with 0.25 to 0.40 per cent. For uniformly thin castings, such as stove plate and sanitary ware that is to be enameled, from 0.55 to 0.75 per cent is used with marked benefit to obtain the necessary fluidity and good mechanical properties. But in castings with thin and thick parts, such as engine blocks, these percentages of phosphorus make the thick portions porous, so that not more than 0.30 per cent can be permitted. High phosphorus also increases the shrinkage, particularly of the heavier parts, and increases the harmful stress-producing shrinkage (called "draw" shrinkage) and those making highest type castings generally agree that phosphorus above 0.30 per cent tends to produce unsound, porous, and brittle castings. Since the required fluidity can be had by raising the temperature, better results are obtained by lowering the phosphorus to 0.12 to 0.20 per cent and increasing the temperature. Castings thus made have a closer grain and machine better than those made with higher phosphorus content.

Therefore, this range is recommended as the best for general work, including machinery castings of various kinds. In general, the resistance of unalloyed cast iron to corrosion decreases as the phosphorus content is increased above 0.05 per cent.

Effects of Chromium—Chromium occurs only occasionally, and in traces, in pig iron made from ordinary iron ore, but it now is added commonly to iron for high-grade castings in amounts from 0.1 to 3.5 per cent. It is a carbide-forming element, hence it holds the carbon in the combined state, opposes graphitization, decreases the tendency for growth, and increases the hardness of the matrix. In moderate amounts of 0.1 to 0.5 per cent, it increases the strength, but greater additions are made with some increase of brittleness. In high-grade castings, it is used mainly to increase the resistance to wear. It is used also with other elements, particularly nickel and copper, to obtain resistance to corrosion and growth. Additions of chromium are made to the iron in the ladle, as solid or preferably molten ferrochromium. Its effects were early studied and reported by Hadfield in 1892 (see: Iron and Steel Institute, Vol. II, 1892).

Influence of Nickel—Nickel, like chromium, is rare in pig iron made from ordinary iron ores; but its use in high-grade castings is common, the amount added to gray-iron castings varying from 0.10 to 2.50 per cent, and to special alloy castings from 5 to 15 per cent. Nickel dissolves in the iron, and, like silicon, promotes graphitization; but, unlike silicon, it does not graphitize eutectoid cementite (that is, the portion of cementite that goes to make up pearlite) and it causes a reduction in the size of the graphite plates, giving a "closer grain." Therefore, in the smaller percentages, it is added to toughen the iron, prevent formation of massive carbides, and increase machinability. For example, one per cent added to gray iron, with carbon reduced from 3.50 to 3.00 per cent and silicon from 1.50 to 1.00 per cent, doubles the tensile strength and increases the Brinell hardness from 175 to 225, and gives an iron that is readily machinable. From 5 to 6 per cent nickel hardens the iron and may result in a martensitic matrix, the Brinell hardness being 250 to 280. A maximum of about 360 Brinell is obtained with 10 to 12 per cent nickel; while larger amounts, about 12 to 16 per cent, produce a distinct type wholly austenitic (except for cementite or graphite) and much softer, the Brinell hardness being as low as 130 with 18 per cent nickel. In these larger amounts, it is used in conjunction with chromium, with copper and chromium, or with silicon and chromium, to produce various special corrosion- and heat-resistant irons. In cupola practice, nickel is added in the form of shot to the metal in the ladle; in air- or electric-furnace practice, it may be added as pig nickel to the charge in the furnace, or in the form of shot in the ladle. Commercial forms of nickel used for this purpose contain some carbon, which should be considered if it is added in large amounts.

Influence of Copper—Copper may occur in pig iron, if it is present in the charge. Usually, it is absent. It is added to castings in various proportions from 0.10 to 2.00 per cent in gray-iron castings and up to 7.00 per cent in special alloy-iron castings. In the smaller amounts, its effects are similar to nickel, decreasing slightly the combined carbon, increasing the strength, preventing formation of massive carbides, improving machinability, and in addition, increasing the fluidity and decreasing the shrinkage slightly. Its solubility in iron is limited, but is increased by nickel, in conjunction with which it is used up to 6.5 to 7.0 per cent to produce corrosion- and heat-resistant castings. As to its influence on graphitization, one per cent copper is equivalent to 0.1 to

0.2 per cent silicon. In malleabilizing, it may be, therefore, both a hindrance and a help, tending to make the iron mottled in casting but shortening the annealing time. The strength of malleable iron containing about one per cent copper can be increased 2000 to 8000 lbs. per sq. in. by precipitation hardening, a treatment usually carried out by heating for one hour at 1380° F and reheating for 3 hours at 930° F.

Effects of Molybdenum—Molybdenum is rarely found in pig iron, but is added in the ladle as ferromolybdenum in cupola practice, or in the furnace as ferromolybdenum or calcium molybdate in air- and electric-furnace practice. It strengthens and toughens the metal, and is added in amounts of 0.30 to 1.25 per cent to increase strength, hardness, and resistance to shock. It is credited also with preventing cracking. It does not decrease the shrinkage on solidification, but does decrease slightly the contraction following solidification. Its effect upon graphitization is slight, and it appears not to affect the amount of combined carbon, but does increase the depth of chill when added in amounts of 0.25 to 0.60 per cent. It increases resistance to wear, hinders graphitization somewhat, has little effect on growth, improves the properties at high service temperatures, and promotes uniformity in mechanical properties as between large and small sections.

Effects of Titanium and Aluminum—Both of these elements have been added to cast iron, but only titanium is used regularly. Both are reported to promote graphitization, titanium decreasing the size of the graphite flakes. A little titanium, 0.05 to 0.10 per cent, is common in pig iron, and its effect appears to be generally beneficial. The addition of titanium to cast iron is reported to impart greater tenacity and resistance to wear.

Influence of Vanadium—Vanadium has been added to cast iron in amounts of 0.10 to 0.15 per cent. It is an expensive addition, and its chief effect seems to be that of opposing graphitization.

Effects of Special Additives—Various chemical elements other than those discussed above are added to iron for casting to effect changes in microstructure, improve mechanical properties, and so on. Such elements may be added singly or in various combinations. A notable instance is that of adding magnesium or cerium alloys to iron to produce what is designated as "nodular iron;" this practice will be described later in this chapter under the heading of "Kinds and Uses of Iron Castings."

IRON-FOUNDRY MELTING METHODS

The chapter on pig iron describes the manufacture of iron used in the production of iron castings. The pig iron for castings may be melted in one of several types of furnaces, the cupola, the air furnace or the electric furnace, though the cupola is the principal source of molten metal for iron castings. In the case of certain alloy and high-test iron castings, the metal is frequently duplexed; i.e., melted in a cupola and further processed in an electric furnace. The open-hearth furnace also has been used for melting iron for castings, but its use for this purpose is limited.

The Cupola—The cupola resembles a miniature blast furnace. It differs primarily in that pig iron and steel scrap replace iron ore in the charge. The cupola is lined with fireclay or firestone refractories. Since, in most installations, no water cooling is utilized in the melting zone, the lining has to be repaired with plastic-fireclay patching between periods of operations. In the few installations where water-cooling of the melting zone is employed, such frequent repair of the lining is not re-

quired. The charging door is located on the side of the cupola near the top. The cupola is supported by legs, permitting the use of a drop bottom which facilitates the removal of the remaining burden after the last charge has been tapped. Intermittent operation is the general rule, but continuous operation is possible.

The charge is composed of coke, steel scrap, iron scrap and pig iron in alternate layers of metal and coke. Sufficient limestone is added to flux the ash from the coke and form the slag. The ratio of coke to metallics varies, depending on the melting point of the metallic charge. Ordinarily, the coke will be about 8 to 10 per cent of the weight of the metallic charge. It is kept as low as possible for the sake of economy and to exclude sulphur and some phosphorus absorption by the metal.

During melting, the coke burns as air is introduced at a 10 to 20 ounce pressure through the tuyeres. This melts the metallic charge and some of the manganese combines with the sulphur, forming manganese sulphide which goes into the slag. Some manganese and silicon are oxidized by the blast and the loss is proportional to the amount initially present. Carbon may be increased or reduced, depending on the initial amount present in the metallic charge. It may be increased by absorption from the coke or oxidized by the blast. Phosphorus is little affected, but sulphur is absorbed from the coke. Prior to casting, the slag is removed from the slag-off hole which is located just below the tuyeres. The molten metal is then tapped through a hole located at the bottom level of the furnace. The depth between these two tapping holes and the inside diameter of the furnace governs the capacity of the cupola.

The Air Furnace is a type of reverberatory furnace somewhat similar to the puddling furnace described in Chapter 11. It has a fireplace at one end, the stack at the other end, and between them a hearth covered by a roof sloping toward the stack. A cross, or "bridge," wall near the stack (the flue bridge) and another next to the fireplace (the fire bridge), together with the lining (usually of silica sand) form a rectangular basin which holds the charge. A removable-bung type of roof is used to permit charging large pieces through the top with a crane. Coal, fuel oil or gas are used as fuel, the liquid or gaseous fuels being preferred. When coal is used, the fireplace is constructed and manipulated to serve as a crude producer. About 15 per cent of the carbon, 30 per cent of the silicon, 45 per cent of the manganese, and 1 to 2 per cent of the iron of a pig-iron charge are oxidized in the air furnace, the exact amounts varying with the composition of the charge and the oxidizing conditions of the flame.

The Electric Furnace—A description of the arc-type electric furnace, which is used to a limited extent for melting iron for iron castings, is given in Chapter 16.

KINDS AND USES OF IRON CASTINGS

One of the principal reasons for using iron instead of steel castings is the fact that cast iron is more resistant than cast steel to warping and cracking in the presence of heat. There are other reasons for the use of iron castings, such as hardness and wear resistance, damping effect, corrosion resistance, intricacy of shape and massiveness of the casting. The various kinds of iron castings are: chilled-iron castings, malleable castings, alloyed-iron castings, gray-iron castings, and nodular-iron castings. Among the innumerable uses may be mentioned pipe, rolls, permanent molds, ingot molds and stools, sanitary ware, engine blocks and all kinds of machinery parts. A few of the various kinds of iron castings, with their compositions, properties and uses, are

Table 18-III Some Varieties of Iron Castings, Their Compositions, Properties and Uses

	Composition—Range (%)											Mechanical Properties				
	Si	S	P	Mn	T.C. *	G.C. *	C.C. *	Cu	Ni	Cr	Mo	Other Elements	Brinell Hard. No.	Tensile Strength p.s.i.	Transverse Load-Lbs.	Deflection In.
Gray Iron—Ordinary	2.00 2.40	.12 max.	.20 max.	.60 .70	3.00 3.35								187 235	30,000–45,000	1800	.14–.16
Cast Iron Pipe	As re- quired	.10 max.	.90 max.	.40 .60			about 3.00							20,000–25,000		
Locomotive Cylinders	.90 1.10	.12 max.	.90 max.	.60 .80	2.90 3.10	2.15 2.35	.65 .85		1.00 1.25				220 240	4,500–5,500		.15–.20
Automotive Cylinder Blocks	2.00 2.20	.09 .12	.14 .18	.70 .85	3.25 3.40	2.60 2.75	.55 .65		.75 .85	.25 .35			187 196	33,000–36,000	3,600–3,900	.12–.14
Cylinder Blocks	2.35 2.40	.15 max.	.18 max.	.55 .75	3.25 3.40		.50 .65	.75 1.00		.20 .30			192 220	38,000–42,000	4,000–5,000	.17–.20
Cylinder Blocks	2.10 2.50	.10 max.	.20 max.	.50 .90	3.00 3.40		.60 .80			.20 .40			187 240	35,000–45,000	3,500–4,500	.11–.17
Cylinder Blocks	1.80 2.00	.12 .15	.16 .20	.70 .90	3.20 3.40	2.60 2.80	.60 .80			.15 .20	.15 .20		212 231	38,000–41,000	4,600–5,000	.14–.16
Crank Cases	1.90 2.10	.09 .12	.14 .18	.70 .85	3.20 3.40	2.70 2.80	.50 .60		1.00 1.10				240 270		3,800–4,000	.12–.15
Crank Shafts	2.20 2.40	.15 max.	.18 max.	.65 .90	3.20 3.40		.70 .90		.40 max.	.80 .90	.40 .50		263 300	48,000–52,000	4,800–5,000	.17–.20
Crank Shafts	2.20 2.50	.08 max.	.08 max.	.90 1.00	2.60 2.80		.60 .75		.75 1.00	.10 .20	.70 1.25		220 240	60,000–80,000	6,000–7,000	.18–.22
Pistons—Diesel	1.65 1.80	.07 max.	.15 max.	.50 .70	2.50 2.70	1.80 2.15	.50 .70		1.25 1.70	.55 .65		<0.12 0.25	229 241	63,000–68,500		
Piston Rings	1.90 2.25	.10 max.	.35 max.	.50 .70	3.40 3.60	2.70 3.10	.45 .75		.90 1.10	.20 .30			297 311			
Piston Rings	2.50 3.10	.40 max.	.30 max.	.50 .80	3.50 3.75	2.70 3.20	.50 .80									
Dies—Forming	1.40 1.50	.10 max.	.18 max.	.60 .80	3.20 3.50				.90 1.00	.60 .70	.80 .90		200 220			
Dies—Forming & Bending	2.00 2.40	.10 max.	.20 max.	.55 .70	2.60 2.90	1.90 2.10	.65 .80		1.40 1.70	.35 .50	.40 .60			55,000–65,000	3,200–3,600	.22–.29
High Alloy Irons Ni-Resist—Typical	1.10 2.00	.06 max.	.04 max.	.60 1.50	2.40 2.90	1.60 1.80	.60 .75	5.00 7.30	14.50 16.50	1.50 4.00			120 170	20,000–35,000	2,000–4,000	.20–.30
Silal—Typical	5.70 4.50	.06 max.	.30 max.	.70 .60	2.40 1.80	2.30 1.50	.08 .30							33,000 23,000	2,400 2,000	.11–.20 .40–.60

*T.C. = Total carbon content; G.C. = Graphitic carbon content; C.C. = Combined carbon content

given in Table 18—III. Further details may be found in "Cast Metals Handbook" published by the American Foundrymen's Association.

Chilled-Iron Castings are extremely hard on the surface. They are made by melting iron of certain compositions, and casting the molten metal in such a way that the parts to be hardened will be solidified on contact with a metal or graphite block capable of abstracting heat rapidly and thus causing quick cooling. Chilled-iron castings are made of ordinary low-silicon iron and of irons alloyed, usually, with nickel and chromium. The chilled surface is very hard, and when fractured, shows white for a distance beneath the chilled surface varying with the rate of cooling and the composition of the iron; hence, such irons are commonly spoken of as chilled iron. The hardness and the white fracture are due to the fact that all the carbon, in the clear chill at least, is in combined form, the rapid cooling preventing the separation of graphite. In heavy sections, the clear chill will extend only a short distance, $\frac{1}{8}$ inch to 2 inches from the surface, where it merges into mottled, then into a gray appearance, all these structures being the result of varying the rate of cooling. Such castings are used for rolls and various other articles which require a hard, wear-resisting surface. All three types of furnaces are used for melting, depending upon the kind of iron and other factors.

Malleable Castings, while not strictly malleable, are soft and can be bent without breaking. They are of two kinds, known as **white heart**, or **European**, and **black heart**, or **American**, these terms indicating differences in the process and the products and countries of origin. As an initial step in making malleable castings, a charge consisting of malleable grades of pig iron (10 to 15 per cent), steel scrap (35 to 40 per cent), and cast-iron scrap, (45 to 55 per cent), consisting of feeders, runners, sprues, and defective castings, may be melted to give metal containing 2.25 to 3.00 per cent carbon, 0.3 to 0.50 per cent manganese, 0.05 to 0.08 per cent phosphorus, 0.06 to 0.11 per cent sulphur, and 0.60 to 1.15 per cent silicon, the exact composition, particularly in regard to silicon, being varied according to the thickness of the section being made. If the carbon is under 2 per cent, graphitization in annealing is slow and if the carbon is near 4 per cent, graphite may be formed in casting. To prevent the latter occurrence, the silicon is lowered as the carbon is increased in a given casting, if the iron must be white all the way through after casting, with all the carbon in the combined form. When the iron has melted, it is tapped and cast in well-prepared green-sand molds, made about $\frac{1}{4}$ inch per foot oversize to allow for total shrinkage, or liquid-to-solid contraction. When cool, the castings are removed from the molds and, unless finish is of no consideration, cleaned by tumbling, sand blasting, pickling, or hand scouring. The castings next are packed carefully in annealing boxes or pots with an oxidizing agent, such as roll scale or crushed furnace slag for white-heart castings, or with nonoxidizing materials such as blast-furnace slag, fine sand, etc., for black-heart castings. The packed castings then are placed in an annealing furnace and heated gradually to about 900° C (1650° F) for white-heart castings or to near 871° C (1600° F) for black-heart castings. The former are held at this temperature 3 to 5 days, during which time the iron carbide is eliminated almost completely by a process of migration and surface oxidation of the carbon to CO or CO₂, leaving a metal similar to soft steel but of much coarser grain and less ductile. Since the migration of the carbon is very slow, only thin castings can be decarburized successfully or "malleableized" by this process. Black-heart castings

are treated by an "annealing cycle," requiring about 30 hours for heating to temperature, about 45 hours holding at temperature, 30 to 35 hours for cooling to and holding at 650° C (1205° F) and 5 hours for cooling to handling temperature. In this cycle, the combined carbon is completely graphitized, causing the casting to grow slightly and leaving the iron as ferrite. But instead of forming plates, the graphite is dispersed in a very finely divided form known as temper carbon, which, under the microscope, is seen as black spots distributed in haphazard fashion throughout the metal except near the surface of the casting where the metal may be partly or wholly decarburized. Thoroughly malleableized castings have a yield point of about 27,000 lbs. per square inch, and a tensile strength of about 45,000 lbs. per square inch. Some standard specifications require a yield point of more than 32,000 lbs. per square inch (32,500 and 35,000 minimum for two grades), and a tensile strength of more than 50,000 lbs. per square inch (50,000 to 53,000 minimum for two grades). The modulus of elasticity is about 25,000,000 lbs. As to ductility, standard specifications require a minimum elongation of 10 to 18 per cent in 2 inches. Castings that have been malleableized are immune to growth.

Alloyed Castings—Alloyed irons are used most extensively for applications where resistance to wear, to heat (including growth), and to corrosion, along with the high strength of the alloyed iron, rigidity, "damping" of vibrations and amenability to heat treatment are of prime importance. They are produced and used extensively by the steel industry but are used most widely by the automotive industry for purposes where the above properties are a requirement. The alloying elements used are silicon, nickel, chromium, molybdenum, copper and titanium, in amounts varying from a few tenths to 20 per cent or more. They may be classed as: (1) low-alloyed gray-iron castings, (2) high-alloyed gray-iron castings, and (3) austenitic alloy castings, the latter containing sufficient alloying elements to hold the iron in the austenitic condition. These are used for resistance to corrosion, both atmospheric and chemical; for resistance to heat, including oxidation and growth; and for their high thermal coefficient of expansion. Many of these irons are patented compositions and are sold under various trade names, such as Ni-resist, Causal metal, Silal, Ni-crosilal, etc.

Gray-Iron Castings—Gray-iron castings are made of pig iron, of mixtures of pig iron and steel, or of mixtures of pig iron, steel and other metals in smaller amounts, and have been referred to as semi-steel, high-test iron, and alloy iron. They are frequently sold under trade names, such as Meehanite, Gunite, Ermalite, Ferrosteel, Gun-iron, etc. Chemically, gray-iron castings include a large number of metals covering a wide range in composition, with carbon varying from 2 to 4 per cent, and silicon from 0.5 to 3 per cent, with small amounts of Ni, Cr, Mo, and Cu frequently added. Grouped according to uses, they include (1) pipe-foundry castings, (2) sanitary ware, (3) automotive castings, (4) locomotive castings, (5) light machinery, (6) heavy machinery, (7) miscellaneous shapes.

Mention has been made of the use of cast iron for ingot molds and stools. If available, molten iron direct from the blast furnace may be used instead of cupola iron because of its lower cost and high percentage of graphitic carbon. In some cases where iron direct from the blast furnace is not available, cupola iron is used for this purpose; however, studies invariably indicate that blast-furnace iron produces a mold with longer life than those produced with more refined cupola iron. The composition of the iron, pouring temperature and their

relation to ingot-mold and stool life require considerable study; however, the consensus of opinion of mold makers in this country indicates that the composition should be about as follows:

Silicon	1.25 to 1.75%
Phosphorus	0.120 to 0.140%
Sulphur	0.035 to 0.050%
Manganese	1.00 to 1.40%

A casting temperature of 2300° F to 2350° F is desirable.

Pipe-foundry castings include cast-iron pipe and fittings. A great part of cast-iron pipe now is cast centrifugally from ordinary iron containing less than 0.1 per cent sulphur, under 0.9 per cent phosphorus, with carbon and silicon controlled to give the required mechanical properties. When not centrifugally cast, the molten metal is poured into cored dry-sand molds supported in a vertical position, those 18 inches or more in diameter being cast with the hub end down.

Nodular-Iron Castings—Nodular iron, also called ductile iron and spheroidal graphite iron, is a relatively new grade introduced around 1948; it has been used for castings having sections from ¼ inch up to 40 inches thick. It is produced by treating molten iron that normally would produce a soft, weak, gray iron casting with cerium or magnesium alloys. The addition of these special alloys results in castings which have the carbon present in spheroidal form. Castings so made have relatively high strength and better ductility than ordinary gray iron. Pearlitic nodular irons have a tensile strength in excess of 80,000 lb. per sq. in., with an elongation of at least 3 per cent, while ferritic grades, having tensile strengths of over 60,000 lb. per sq. in., will show an elongation of from 10 per cent to 25 per cent. For economy, iron having a sulphur content below about 0.15 per cent is required for the process. Several types of matrix structures can be developed by alloying or heat treatment; the pearlitic and ferritic matrices were mentioned above. As this type of iron has been discussed extensively in recent literature, no further details will be given here.

IRON-FOUNDRY MOLDING AND CASTING PRACTICE

Molding practice for iron castings is somewhat similar to that already described for steel castings in Section 1 of this chapter, the chief differences being in the preparation and types of sand and the placement and size of gates and risers, the latter being modified by reason of the lower shrinkage in cast iron.

The scope of this book does not permit more than a brief description of the casting of iron castings. The metal is cast in one of five types of molds: namely, (1) **green-sand molds**, made of moist sand which is rammed about a pattern (usually of wood) in a "flask" of wood or iron, and the metal is poured with the mold in the condition as rammed; (2) **dry-sand molds**, made up like green-sand molds, but dried before they receive the metal; (3) **loam molds**, made up of loam (a kind of low-grade sand) which, for heavy castings, is backed with brick and faced with other more refractory material; (4) **permanent molds** or **semi-permanent molds**, which have become more and more popular for certain applications; and (5) **shell molds** which are a recent development.

The permanent mold is a cast-iron or graphite mold into which the molten iron is poured. The semi-permanent mold is built up of cast iron and sand, the latter having to be replaced after use. In both the latter cases, the molds are prepared with a graphite coating and warmed to 150 to 200° F before the hot metal is

poured into the mold. Shell-molding techniques are described in numerous recently published articles and papers and will not be discussed in detail here; briefly, shell molds are made by applying a coating of sand mixed with a synthetic resin or other suitable binder to a prepared pattern and then "curing" the coating by heating to form a solid shell that can then be stripped from the pattern. Shell molds are used in the production of steel and nonferrous castings, as well as iron castings.

TESTING OF CAST IRON

The tests most commonly employed for gray cast iron are the tension test, transverse load and deflection (measured bend) test, and hardness (Brinell and file) tests. In making tension tests, standard test specimens are machined from a standard cast test bar, and "pulled" in a tensile-testing machine until the piece breaks, the load calculated to pounds per square inch being taken as the tensile strength. In short-time tension tests, most grades of cast iron show no point corresponding to the yield point of steel and very little, if any, elongation or reduction of area. The form of specimen used for making tensile tests is somewhat different from that used for testing steel. There are three standard sizes; viz., 0.505 inch, 0.750 inch, and 1.25 inches in diameter (at the center), the diameter being varied with the thickness (and design) of the castings they represent. As standard specifications do not require the measurement of elongation but only the tensile strength, specifications covering the lengths of specimens merely state that the affected test length shall not be less than the diameter. The tensile strength varies: (a) with the diameter of the test bar, being higher for bars of smaller diameter and lower for bars of greater diameter; (b) with the temperature above 250° F, being almost constant up to 600° F and decreasing rapidly above 800° F; and (c) with the time after casting, most castings increasing in strength with age. This change is attributed to relief of casting strains, which may be relieved also by tumbling and by heat treatment. The relief of strain by aging is called **seasoning**.

Transverse testing consists of placing a standard bar upon supports 12, 18, or 24 inches apart, then either applying a specified load and noting the deflection, which is measured in inches, or of loading until deflection occurs and then gradually increasing the load till the specimen breaks. With the latter procedure, the **modulus of rupture** is found from the equation
$$M R = \frac{2.546 L S}{D^3}$$

where $M R$ = modulus of rupture, L = distance between supports in inches, S = the breaking load, and D = the diameter of the test bar.

In the Brinell hardness test, a special machine is used to apply a load of 3,000 kg. to a steel ball 10 mm. in diameter, resting on a filed-smooth surface of the iron to be tested. The Brinell number is taken as the quotient of the load divided by the area of the impression made by the ball. Thus, small numbers up to 100 indicate softness, while high numbers (400 to 600) indicate great hardness.

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Chapter 19

PLASTIC DEFORMATION OF STEEL

SECTION 1

INTRODUCTORY

In discussing the subject of plastic deformation in a book of this nature, it is necessary to point out that it is possible to consider only the major aspects of this very complex and very important behavior of metals. There is a voluminous literature on the subject to which recourse should be had if any extensive study of the subject is contemplated. The contents of this chapter are intended to serve as an elementary introduction to some of the concepts that are useful in understanding how solid metals can be worked to change their shape into useful articles.

The process of shaping steel by casting (pouring the molten metal into suitable molds where it is allowed to solidify in the desired form) was described in previous Chapter 18. The very large majority of the steel produced, however, is processed by forging, rolling, or other forms of mechanical working, to give it the desired shape. Steel so worked is called **wrought steel**. The numerous means of working involved in shaping steel are categorized as **mechanical treatment**. The present chapter will briefly discuss some of the properties and characteristics of steel that enable it to undergo a change of shape without rupturing, and will summarize the principles of the more important methods used to shape steel. More detailed discussions of most of the methods will appear in subsequent chapters.

Objectives of Mechanical Treatment—The primary objective of mechanical treatment, then, is the production of wrought steel in shapes and sizes that adapt it to specific end uses. It is a fortunate circumstance that the operations involved in the mechanical shaping of steel, if properly performed, result in the attainment of a second objective—that of improving the quality of the steel by altering its mechanical and physical properties.

All wrought steel originates in the form of ingots. Ingots, as described in the next chapter, are large castings of relatively simple shape which possess certain inherent weaknesses and defects. These shortcomings can be eliminated, to a great extent, through proper mechanical treatment as the metal is reduced, through successive stages of working, into finished products. The amount of work performed, the rate at which it is carried out, the temperature of the steel while being worked, and the rate at which it cools to ordinary temperature after working, all affect the ultimate properties of the finished wrought-steel products.

Crystalline Nature of Steel—Solid steel is crystalline in nature; that is, it is made up of innumerable allotriomorphic crystals called grains. The grains possess all of the characteristics of true crystals except the regularity of external shape associated with perfect (idiomorphic) crystals. Some of the properties and the structure of crystals were discussed in Chapter 2 under "Crystallography."

Elasticity and Plasticity—It was stated in Chapter 2, and is elaborated in Chapter 49, that steel, when subjected to loads under the elastic limit, will be deformed by an amount proportional to the load. However, when the load is removed, the steel will resume its original shape and dimensions, and no permanent distortion occurs. This restorative ability is called **elasticity**.

If, however, the metal is subjected to loads or forces exceeding the elastic limit, permanent distortion will take place. Between the elastic limit and the load that causes actual rupturing of the steel, the metal "flows" under load in an attempt to adjust itself to the imposed forces. This ability to react to imposed loads above the elastic limit by changing shape permanently without rupturing is referred to as the **plasticity** of steel, and the mechanism by which shape is altered without rupture is called **plastic deformation**.

Mechanism of Plastic Deformation—The ability of solid metals to behave in a plastic manner is related to their crystal structure. *Elastic* behavior may be conceived as representing simple distortion of the crystal lattice under load; when the load is removed, the lattice is restored to its original dimensions. However, when plastic deformation occurs, the crystals themselves are distorted. A simplified explanation of how such permanent distortion can occur without actually rupturing the crystals is as follows:

The locations of atoms in the crystal lattice define certain planes along which parts of each crystal can be displaced in relation to adjacent parts without rupturing the crystals. These displacements result in permanent deformation of the crystals. The planes along which displacements (called "slip") can occur lie in definite directions related to the crystallographic axes of each crystal, and are limited in number. The effect of plastic deformation on metals at ordinary temperatures will be considered first.

Strain Hardening—It is an interesting fact that slip can occur only to a definite extent along any one plane. Any further deformation of a crystal must take place through slip on other planes within that crystal. Eventually, if stress above the elastic limit is applied, either continuously or in a succession of applications, all possible slip will have occurred on the available planes, and any additional application of sufficient stress will rupture the crystal instead of deforming it further. Each increment of slip in its crystal system reduces the ability of a metal to deform further plastically, and is reflected in a change in the properties of the metal. In general, the yield strength, tensile strength and hardness of the metal are increased, while the ductility is decreased—all to a degree proportional to the amount of plastic deformation that has occurred. This phenomenon is called **strain hardening**, and metals which have been

strain hardened contain internal stresses resulting from deformation.

Recovery—In some metals which have been strain-hardened, the metal atoms possess sufficient mobility at ordinary temperatures to move about (through a mechanism called **diffusion**) and assume positions that eventually tend to relieve the internal residual stresses. This phenomenon is called by the scientific name of **recovery**. Among the more common metals at ordinary temperatures the rate of recovery is so slow that it has little practical significance. By heating mechanically-worked metal, the rate of recovery can be increased many-fold, and most of the residual internal stresses are eliminated in a comparatively short time.

Recrystallization—If mechanically worked strain-hardened metal is heated to higher and higher temperatures, a temperature level is reached where a second process—quite different from recovery—occurs. This is called **recrystallization**, and results in the complete rearrangement of metal atoms to form an entirely new set of crystals, and elimination of the strain-hardened condition.

The greater the degree of prior plastic deformation, the larger will be the amount of stored energy of distortion in the metal, and the lower will be the temperatures at which recrystallization begins.

Hot Working and Cold Working Defined—From the foregoing, it is evident that if a metal was worked me-

chanically at a high enough temperature, any strain hardening that occurred during working would be eliminated almost instantly before the metal cooled to ordinary temperatures. On this basis, **hot working** is defined as permanent deformation of a metal at temperatures such that no strain hardening is evident in the wrought metal after it cools to room temperature. **Cold working**, conversely, leaves the metal in the strain-hardened condition. In the case of steels, the effects of cold working can be removed entirely by heat treatment (see Chapters 34, 40 and 42).

In either hot working or cold working, if stresses of a sufficient magnitude to change its shape are applied to a piece of steel, its over-all change in dimensions represents the aggregate of the tiny displacements which occurred in individual crystals making up the piece.

Actually, as stated in the section on "Crystallography" in Chapter 2, the simplified mechanisms described above are complicated by many factors which cannot be discussed in detail here.

Many theories have been advanced for estimating the strength of the facilities and the power required for the plastic forming of steel, but no one of them has as yet been demonstrated to be completely valid. The chief reasons for this situation seem to be the lack of physical data for evaluating the deformation resistance of steel and the friction coefficients which obtain under the conditions of actual operations.

SECTION 2

HOT WORKING AND ITS EFFECTS

The upper temperature limit for the hot working of steel depends upon the carbon content (for plain carbon steels) because hot working must be carried out at a safe temperature below the melting point of a given steel. The upper limit for starting hot work may be considered to be between 2300° and 1950°F, the higher temperatures being used for steels of the lower carbon contents. At such temperatures, the steel is composed entirely of austenite (see Chapter 42).

The chief benefit derived from proper hot working of steel is the refinement of its grain structure. In the case of ingots, hot working also welds unoxidized cavities shut, tends to eliminate porosity if it exists in the center of an ingot, and tends to lessen segregation.

During hot working, the metal crystals (grains) are

deformed or fragmented, and a strained condition is set up in the steel. The relatively great mobility of atoms at hot-working temperatures permits the strained condition to be relieved by the formation of new small grains of austenite from the deformed, fragmented crystals either during or immediately after hot working (Figure 19-1). The new grains grow in size at a rate which is greater the higher their temperature and the longer the time they have to grow at that temperature. The new small grains are equi-axed and are randomly oriented. As the hot-worked steel cools to room temperature, it passes through the critical temperature range (Chapter 42) and a new crystal structure is produced through allotropic transformation.

Without discussing the somewhat complex effects of

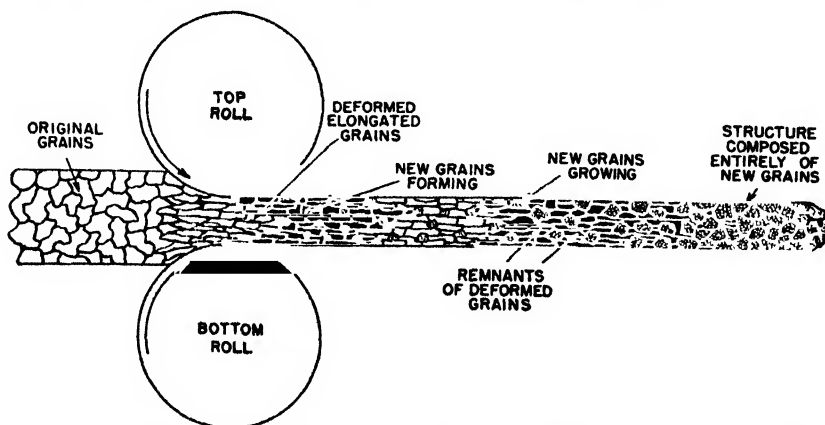


Fig. 19-1. Diagrammatic representation of the combined effects of the mechanisms of hot rolling and recrystallization on the grain structure of steel. (Grain size is shown much exaggerated for clarity.)

working steel in the critical temperature range, and the resulting microstructures, the lower critical temperature may be stated to be the practical lower limit for hot working plain carbon steel, regardless of its carbon content. In the case of alloy steels, the alloying elements affect the transformation temperatures, and their effects, along with other factors, will establish the proper minimum finishing temperature.

Finishing Temperature—Hot work is usually performed in a succession of steps, such as a series of passes in rolling or a series of blows or squeezes by a hammer or press. The more frequent the applications of pressure and the lower the temperature at which they occur (but still above the lower critical temperature), the smaller the grain size will be at the completion of hot working. The more frequent the pressure applications, the less time the new grains have to grow between successive workings. The lower the finishing temperature at which the working is completed, the slower the new grains will grow. Also, the greater the amount of hot work, the finer will be the ultimate grain size. The best mechanical properties can be expected from steel having a fine (small) grain size.

If the temperature at which hot working is begun could always be selected so that the piece would be finally shaped when it reached the lower critical temperature, the steel would have the minimum grain size and the best possible mechanical properties obtainable by hot working and still be in an unstrained condition. In actual practice, shapes of large size or great intricacy require a high degree of plasticity obtainable only by starting hot working at a high temperature in order that the metal will flow or work with relative ease without requiring excessive amounts of power for working or too long a time for effecting the desired shaping. Under such conditions, working is usually completed at temperatures well above the lower critical temperature. Discrimination is required to establish heating and working schedules for hot working that will give the steel its best properties within practical economical and metallurgical limits.

Fiber and Banding—Even after hot working, when the microscopic grains have become entirely recrystallized, there still persists in the hot-worked piece a certain elongated structure, which affects the mechanical properties in some degree. This elongated structure has its origin in two features of the ingot structure. One feature is the presence of microscopic foreign matter or particles, called non-metallic inclusions; it is impossible to produce steel absolutely free from these inclusions. Such inclusions (or regions of inclusions) are elongated as the steel is worked, and result in a condition sometimes called **fiber**. In practically all rolled steel, or any steel hot-worked principally in one direction, the mechanical properties (especially the ductility) in the direction of working are different from those in a perpendicular direction. Thus, if a tension test is made in the longitudinal direction (that is, parallel to the direction of rolling or working) the elongation of the test piece (before rupture) will be greater than when the test is made transversely (perpendicular to the direction of rolling or working). This "directional" phenomenon is thought to be associated with fiber. Another appearance thought to influence the directional behavior is **banding**. This is the second phenomenon inherited from the ingot. If a rolled product (e.g., a bar or a plate) is cut longitudinally, and this section is examined with a microscope, the banded structure, if present, will be seen to consist of a myriad of bands parallel to one another and parallel to the direction of rolling. They are due to slight differences in chemical composition from edge to center of each band.

These slight differences in composition had their origin in the solidification process in the ingot, inasmuch as each grain in the ingot shows such differences. The grains in the original ingot were by no means elongated and parallel, but became so because of the rolling. The spontaneous recrystallization which occurs in hot rolling is unable to affect these minute chemical segregations, which therefore remain in the banded condition. The banded condition is believed to contribute to the directional phenomenon in mechanical properties, described above. The directional differences are found not only in the tension test, but also commonly in the notch-impact test.

PRINCIPAL METHODS FOR HOT WORKING

The principal methods of forming steel by hot working are hammering, pressing, rolling and extrusion. Hot working by the first two methods is called forging.

Forging may be performed under hammers, in mechanical presses and upsetters or by a method known as roll forging. Pressing generally includes the manufacture of forged articles in hydraulic presses. Extrusion usually is performed in hydraulic presses which force the hot plastic steel through a die. Rolling is performed in rolling mills of a variety of types described in detail in various other chapters. The present discussion will be limited to general descriptions of the various types of hot-working equipment and some of the principles of their operation.

Hammering—Hammering was the first method employed by man in shaping metals. The first forging was done by hand hammers wielded by workmen.

The first known power hammer, called a **tilt hammer**, was built in England. It was driven by water power and consisted of a beam of wood, hinged at one end and provided with an iron hammer head or die at the opposite end. At an intermediate point between the hinged end and the free end carrying the hammer head, cams on the revolving shaft driven by the water wheel alternately raised the free end and allowed it to fall upon an anvil or die fixed upon a suitable foundation. This was a crude tool compared to the steam hammers now used.

The first steam hammer was built in France during 1842. It consisted of a two-piece frame, constructed so as to support a vertical steam cylinder, fitted with a piston and piston rod, directly over a die or anvil. To the piston rod of the steam cylinder was attached a tup or hammer head. By admitting steam to the cylinder below the piston, the hammer was raised for any desirable length of stroke and then allowed to drop upon the work piece supported on the anvil or bottom die.

In order to increase the striking force of the steam hammer above that derived from gravity alone, there was developed the **double-acting steam hammer**, in which steam can be admitted above the piston also and employed both on the downward stroke as well as for lifting the tup. This type of hammer is illustrated in Chapter 32, relating to the forging of axles. The first double-acting steam hammer was built at Midvale, Pennsylvania, in 1888.

A variety of other types of forging equipment employing the impact principle for forming hot steel have been developed but cannot be described here. Descriptions are available in reference works listed at the end of this chapter.

Pressing—The hydraulic forging press is an English invention dating from the year 1861. It was introduced into the United States about 1887. It consists (see Figure 19-2) essentially of a hydraulic cylinder supported by



FIG. 19-2. Hydraulic press in operation, forging a massive ingot supported on a porter bar which, in conjunction with the link-chain support, permits manipulation of the ingot as desired.

two pairs of steel columns which are anchored to a single base casting of great weight and strength. The piston or ram of the cylinder points vertically downward and carries the upper forging die, which is directly above a stationary die resting on the base casting to which the columns are attached. By admitting water under high pressure to the cylinder at its top, the ram carrying the upper die is forced down upon the material to be forged, which rests upon the lower forging die. Small auxiliary cylinders lift the ram after each application of pressure.

The pressure, which must be very high if the forging press is to do effective work, is increased gradually and maintained until the metal yields. In practice, it has been found that the lowest pressure that can be effective in shaping steel at a full forging heat is about 1.2 tons per square inch, but the pressures employed in actual work often will exceed 13 tons per square inch.

Extrusion—The hot-extrusion process consists of enclosing a piece of metal, heated to forging temperature, in a chamber called a "container" having a die at one end with an opening of the shape of the desired finished section, and applying pressure to the plastic metal through the opposite end of the container. The metal is forced through the opening, the shape of which it assumes in cross-section, as the metal flows plastically under the great pressures used.

The equipment and methods for carrying out one type of hot extrusion are described in detail in Chapter 41.

Mechanical Forging—Many hot forgings are produced on mechanical presses. In machines of this type, pressure is applied to a vertical ram (carrying the upper forging die) through a connecting rod from a crankshaft. The heated work piece rests on the bottom die. The stroke of such a press is limited to the "throw" of the crankshaft.

Upsetting—A special type of mechanical press is the upsetting machine, in which the piece to be shaped is clamped between two dies with vertical faces and

shaped by the action of a tool on a ram operated by a crankshaft. The ram of the upsetting machine operates with a horizontal, instead of vertical, stroke.

Hot Rolling—Of all the known methods of shaping steel, that of rolling, as introduced by Henry Cort in 1783, has come to be employed the most extensively. Though Cort is credited rightly with being the "father of modern rolling", because of his successful development of mills employing grooved rolls, the use of this principle in shaping metal antedates his mill by many years. There are records, for example, to show that in the year 1553, rolls were employed in France to produce sheets of uniform thickness for the stamping of gold and silver coin. In Sweden, rolls were employed to produce certain sections prior to the year 1751, and even at that time the assertion was made that as many as twenty times more bars could be reduced in a given time than could be shaped under the tilt hammer of those days. From the days of Cort to the present time, the rolling mill has passed through a rapid process of development, not only in the size, power and productive capacity of mills, but also in their design and in the increasing variety of shapes of sections that can be produced.

Mechanical Principle and Effects of Rolling—The process of shaping steel by rolling consists essentially of passing the material between two rolls revolving at the same peripheral speed and in opposite directions, i.e., clockwise and counterclockwise, and spaced so that the distance between them is somewhat less than the height of the section entering them (Figure 19-3). Under these conditions, the rolls grip the piece of metal and deliver it, reduced in cross-sectional area and increased in length. The extent of sideways or lateral spreading (called spread) is found to depend mainly upon the amount of reduction and the shape of the cross section entering the rolls; thus, in rolling plates of considerable width, the actual total spread is independent of the width, and actually may be less than that resulting from the first pass in the reduction of small, square

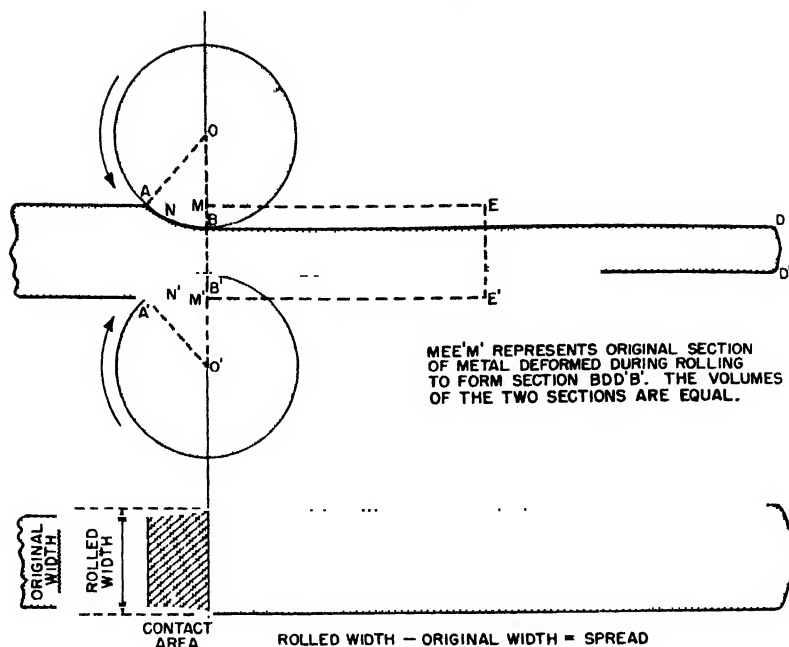


FIG. 19—3. Diagram illustrating the action of plain rolls upon a piece of hot, plastic steel of originally square cross-section.

billets, especially if the percentage reduction in cross-sectional area of the latter is great.

The turning of the rolls in contact with the work introduces a frictional force which acts along the arcs AB and A'B' of Figure 19—3, and is proportional to the pressure between the rolls and the piece. This pulls the work into the opening between the rolls, against the wedging action of the tapered section entering the rolls. The piece is delivered at a higher speed than the roll-surface speed; it enters at a velocity lower than the roll-surface speed. The ratio of the speed with which the work leaves the rolls to the surface speed of the rolls themselves is called the **forward slip**. Evidently, there must be some point between A and B and A' and B' where the speed of the bar is equal to the roll-surface speed. This point, indicated on the drawing by N and N', is called the **neutral point**, which coincides with the point of maximum pressure. The arc AB is called the **contact arc**, and its including angle OAB the **contact angle** or **rolling angle**. When this angle is the maximum at which the piece will enter (without pushing), it is called the **angle of bite**. The area of steel under the contact arc is called the **contact area**, which is projected to show the spreading that may occur.

Effects of Work Temperature—The effects of rolling with respect to changes in physical dimensions of the piece are influenced very markedly by the temperature of the piece being rolled with respect to both degree and uniformity of heating. Additional plasticity imparted to steel by relatively slight increases in temperature lessens the power required for rolling and increases the ease with which it can be made to flow plastically in the desired directions. Chemical composition of a particular steel and the nature of the rolling operation to be performed may limit, respectively, the maximum and minimum rolling temperatures that are applicable.

Effects of Roll Diameter—Small diameter rolls require less force than rolls of larger diameter to effect a given reduction. Advantage is taken of this fact in four-high

and other mills employing small-diameter work rolls backed up by heavier rolls that prevent the smaller rolls from bending. Small rolls lessen the separating force for two reasons: first, the area of contact is less so that, with a given pressure, the total force required is less; and second, the required average pressure is less because the smaller area of contact reduces the total frictional forces.

Miscellaneous Hot-Working Methods—**Rotary swaging** is performed to taper the end of bars, wires and tubes. The machine in which the work is done has two or four shaped dies, in opposing pairs, suitably mounted in a rotating ring. As the ring rotates, alternate pairs of dies are forced against the metal being swaged, the resultant pressure shaping the piece. Such a machine is illustrated in Chapter 41 (Figure 41—40).

In **hot spinning**, which is limited to shapes symmetrical about the spinning axis, the heated piece to be shaped is mounted in a lathe or similar machine that can rotate it rapidly. A tool is then brought to bear against the spinning piece and, by manipulating the position and pressure of the tool, the work piece can be shaped. For example, bowl-shaped sections, such as flanged and dished heads, can be formed from flat, circular plates by spinning.

Many steel parts are formed hot from plates and sheets by **hot deep-drawing** operations that would be impracticable if the material were at room temperature. Roof ribs for railroad box cars, one-piece gates for hopper cars, deep bowls, etc., are typical. The heated steel is formed in hydraulic or mechanical presses equipped with forming dies that produce the desired shapes. Another example is the manufacture of closed-end cylinders by a combination of hot cupping and drawing operations, described in Chapter 41.

Roll forging, **die rolling**, and other hot-forming processes of limited use have not been included in the foregoing descriptions. Reference to these processes can be found in various handbooks.

COMPARISON OF METHODS FOR HOT WORKING

It is a very difficult matter to make a fair comparison between rolling and forging, or even between hammer and press forging. Each method has a field of its own with rather well-defined boundaries. Many irregular shapes are so intricate in design that rolling or extruding them is out of the question, and such shapes must be formed under the hammer or in the press. Certain crankshafts or a claw-hammer head serve as examples of these classes of shapes, which can be produced by no other form of mechanical working than by the forging method.

Tonnagewise, the hammer and the press are both slower and more expensive to operate than rolls. Special care must be given to all phases of the forging operation, including heating for forging. It is, perhaps, this meticulous care required to produce high-quality forgings that has given rise in some quarters to the belief

that forged articles always are superior to rolled articles. Assuming that a given section can be produced by either rolling or forging, and that an equivalent amount of attention is given to all details of both processes with regard to heating, proper speed and amount of reduction, etc., the quality of rolled material can be equal to that of forged product.

With the complex stress-strain relationships involved in plastic deformation in hot working, and the basic differences in the nature of the major hot-working processes, direct comparison is difficult. However, studies have shown that the effects of rolling and pressing are comparable, provided equipment for carrying out both processes is of comparable capacity and the same sized pieces are being worked.

Rolls have the distinct advantage of speed of production where the shapes involved are of a nature suited to rolling. There is one field of operation in which rolling, hammering and pressing all can be applied; this is the shaping of blooms and billets from ingots.

SECTION 3

COLD WORKING AND ITS EFFECTS

In contrast to hot working, the effect of the cold working of steel by rolling, drawing or other means, is evidenced by changes in the physical structure of the material by grain distortion and is accompanied by changes in the mechanical properties due to strain hardening. Cold working, generally applied to bars, wire, tubes, sheet and strip, is a process of reducing the cross-sectional area by cold rolling, cold drawing, or cold extrusion. Cold working is employed to obtain the following effects: improved mechanical properties, better machinability, special size accuracy, bright surface, and the production of thinner gages than hot work can accomplish economically. The present brief discussion will be supplemented in later chapters by discussion of methods and effects of cold working various steel-mill products.

Steel cold worked in the temperature range between about 400° and 750°F may become brittle and difficult to work. This temperature range is called the "blue-brittle range," a name given it by early metalworkers because steel with a clean surface heated in that range assumes a blue color due to oxidation.

A special technique called "warm working" is described in Chapter 41.

While steel can be cold worked at temperatures just below the lower critical temperature, better control of grain size, surface condition and dimensions can be effected by working at atmospheric temperatures.

The over-all effect of cold work on steel is to increase its strength and hardness and to decrease its ductility (see Chapters 34 and 40). If steel is cold worked in a series of steps, each succeeding deformation increases the hardness until a point is reached where either no further deformation can be effected with the forces that can be exerted by the available cold-working equipment, or the steel will rupture if excessive force is applied in an effort to obtain further deformation. Proper heat treatment will restore the plasticity of the steel so that further cold working can be performed if necessary (see Chapters 34 and 40).

If cold working is carried out in one direction only, as in cold rolling sheets or cold drawing bars or wire, the cold-worked steel is stronger in the direction of working than at right angles to it.

PRINCIPAL COLD-WORKING METHODS

Cold Rolling—Cold working by cold rolling consists of passing unheated, previously hot-rolled bars, sheets or strip (cleaned of scale) through a set of rolls, often many times, until the final size is obtained. Methods and effects of cold rolling wide strip are discussed in detail in Chapter 34.

Cold Drawing—In this process a bar, wire or tube, after being cleaned, is pulled through a die having an opening smaller than the entering piece to reduce the latter to the required size (see Chapter 40).

Cold Extrusion—The cold extrusion of steel is carried out in a manner similar to the hot-extrusion process, with two main exceptions: (1) The steel is at room temperature, and (2) the surface of the piece is treated by some chemical process such as bonderizing to assist in reducing friction between the steel and the container wall and die, in conjunction with special lubricants.

Chapter 20

INGOTS AND THEIR PREPARATION FOR ROLLING

SECTION 1

INGOT CHARACTERISTICS

Ingots—Following completion of the refining operation in the open hearth, Bessemer converter, or electric furnace, the molten steel is poured from the furnace into a ladle. From the ladle, the steel is **teemed** into a series of **molds** of the desired dimensions, where the liquid metal is allowed to solidify in whole or part. These solidified steel castings are called **ingots**. Before rolling begins, however, the ingots must have been allowed to solidify throughout, and the whole body of metal should be of uniform temperature. During natural cooling, these conditions are not fulfilled, because the outside of the ingot, being the part from which the heat is removed most rapidly, is the first to solidify and is cooler than the interior. The molds are stripped from many ingots while their central portion is still so hot as to be in the liquid state. This fact was recognized by early steel workers, so it was originally the custom to strip the molds from ingots as soon as possible, and then place the ingots in a tightly covered hole or pit in the ground, where the heat from the interior of the ingot was conveyed slowly to its relatively colder outside portion by conduction. This sufficed not only to heat the colder ex-

terior part of the ingot, but also to supply heat to the pit which, with careful manipulation, was kept hot enough to maintain ingots at a suitable rolling temperature. This process was called **soaking**, hence the name **soaking pit**. In order to bring the soaking under better control and make it adaptable to varying conditions of operation, means for supplying additional heat were introduced, so that the modern soaking pit is, in reality, a kind of heating furnace; detailed descriptions of soaking pits are given in Chapter 21.

Ingot Characteristics—An ideal ingot would be one free from non-metallic matter, and from all cavities or openings, and of uniform physical structure and chemical composition throughout. Unfortunately, the natural laws that govern the solidification of a liquid metal operate against these requirements, and develop the well-known natural phenomena called **pipe**, **blowholes**, **segregation**, **columnar structure**, and **internal fissures**. Added to these are detrimental occurrences, both incidental and accidental, such as **ingot cracks**, **non-metallic inclusions**, and **scabs**. A brief discussion of these phenomena will follow, but a proper understanding of

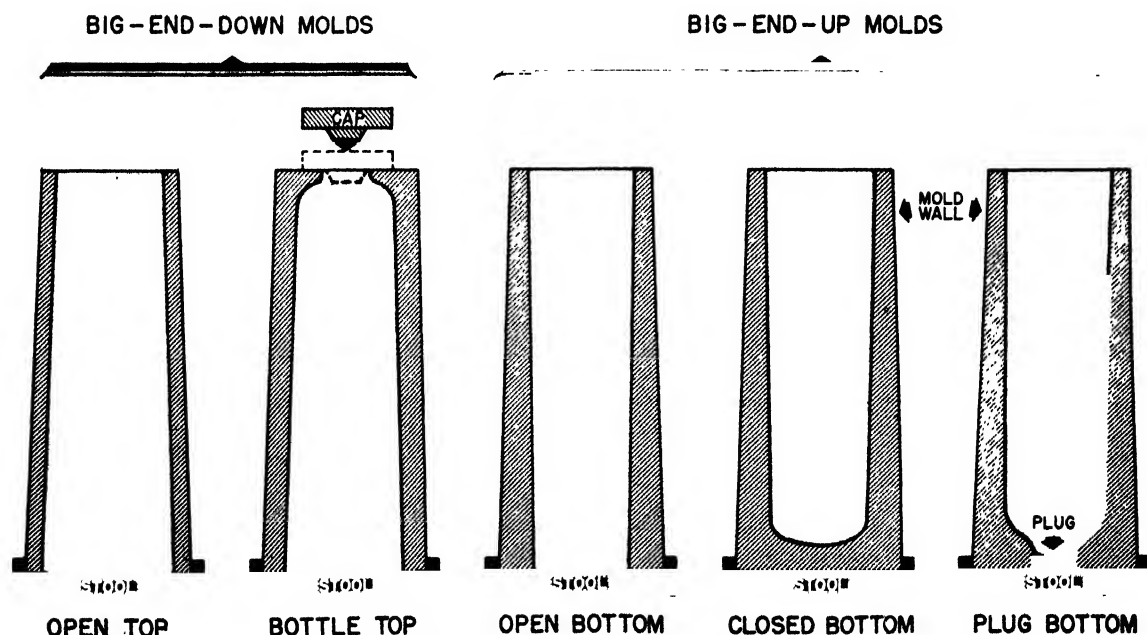


FIG. 20—1. Cross-sections (not to scale) of the five principal types of ingot molds. Molds usually are cast from molten pig iron directly from the blast furnace.

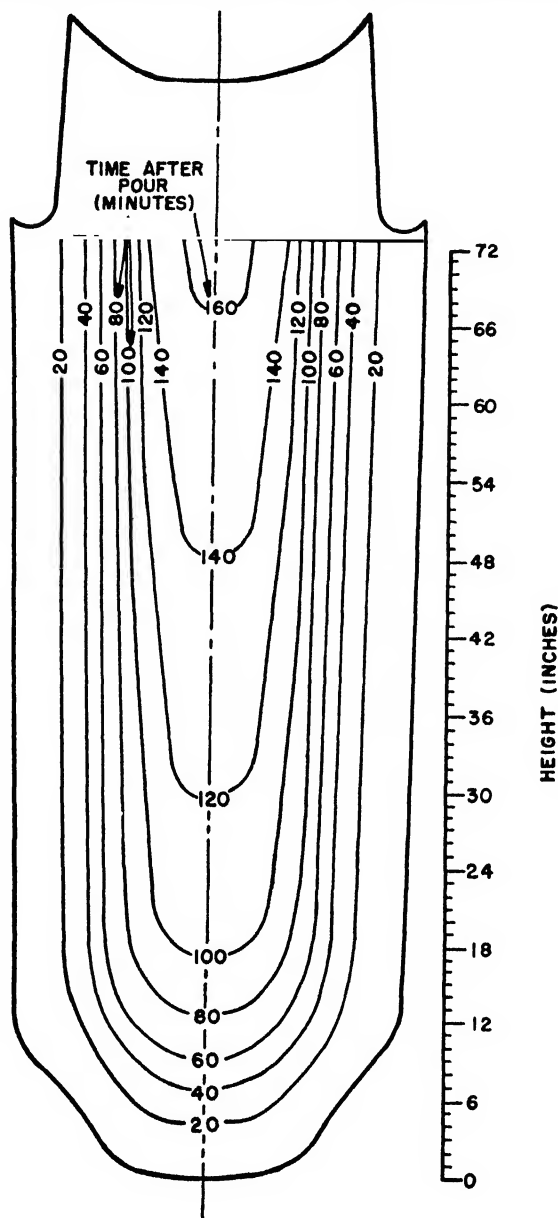


FIG. 20-2. Solidification pattern of 32" \times 32" hot-topped big-end-up ingot of killed steel.

pipe, blowholes, and segregation, and the methods for controlling them, requires a brief study of the mechanism of solidification of ingots.

Nature of the Cooling of An Ingot—The ingot molds in common use are tall, box-like containers made of cast iron, weighing usually from one to one and one-half times as much as the ingot they produce. The inside cavity is slightly larger at one end; thus, the ingots taper from one end to the other to make it easier to separate the ingot and mold. As shown in Figure 20-1, this gives rise to two main types of molds—big-end-down and big-end-up. Big-end-down molds are classified as open-top and bottle-top. Big-end-up molds are classified further into open-bottom, solid-bottom and plug-bottom molds. In the last-named type, the interior is constricted at the bottom to a small, circular opening,

closed with a refractory or metal plug before the ingot is poured. In all open-bottom molds, the bottom opening is closed by placing the mold on a cast-iron base called a stool (Figure 20-1).

As molten steel is poured into an ingot mold, the metal next to the mold wall and stool is chilled by contact with the cold surfaces and solidifies on the bottom and sides to form what is called the skin of the ingot. As more and more heat is extracted by the mold, this skin grows in thickness, but, for several reasons, it grows at a rapidly decreasing rate. As the temperature of the inside of the mold approaches that of the outside of the ingot, the rate of heat transfer decreases. Also, in both big-end-down and big-end-up molds, an air gap forms between the ingot and the mold wall because of the contraction of the former and the expansion of the

latter. As solidification progresses the distance between the mold wall and the remaining molten metal increases and the temperature gradient decreases, thereby decreasing the rate of heat transfer.

Also during the solidification process, gases are evolved from the metal to a greater or lesser degree, depending upon the type of steel, as will be discussed later.

Time for Solidification of Ingots—The rate of solidification is affected by the thickness of the mold wall and the design of the mold. Figure 20- 2 shows the idealized solidification pattern of a 32-inch by 32-inch killed-steel ingot, with the lines labeled 20, 40, etc., indicating the limits to which solidification had progressed after a corresponding number of minutes. The location of these lines was established on the basis of data obtained by pouring a series of identical ingots and then dumping one after another at set intervals to pour out the re-

maining liquid steel. The solidified shells were then removed from the molds and split vertically for study and measurement. These were hot-topped ingots of killed steel of the following composition:

Element	(%)
C	0.83
Mn	0.77
P	0.014
S	0.024
Si	0.18
Ni	2.08
Cr	0.15

The relationships for determining the comparative rates of solidification of ingots of various sizes and shapes are quite complex and are outside the scope of this book.

SECTION 2

TYPES OF INGOT STRUCTURES

When molten steel cools to the temperature range in which it begins to solidify, the solubility of the gases dissolved in the metal decreases, and the proportions of gases that can no longer be held in solution are expelled from the metal. Still more important to the present subject, the chemical equilibrium between carbon and oxygen is changed as the temperature falls, and these two elements react to form carbon monoxide that is evolved from the metal during cooling as the system attempts to establish a new equilibrium. The molten steel does not solidify at a precise temperature but over a range of temperature, so that gases evolved from still-liquid portions may be trapped in semi-solid plastic zones to form blowholes.

The various types of ingots—**killed**, **semi-killed**, **capped** and **rimmed**—are dependent for their characteristics upon the extent to which gas evolution is suppressed. Control of gas evolution is effected by controlling the amount of oxygen dissolved in the liquid steel so that the unavoidable reaction between carbon and oxygen will occur only to the desired degree. The amount of oxygen in the steel is controlled to some degree by controlling the characteristics of the slag in the furnace, but chiefly by the addition of deoxidizers to the molten steel in the furnace during the finishing period, in the ladle or in the molds or in all three as required for the particular type of steel being produced.

Figure 20—3 illustrates diagrammatically eight typical

conditions of commercial ingots, cast in identical bottle-top molds, in relation to the degree of suppression of gas evolution. The dotted line indicates the height to which the steel was poured originally in each mold. The ingots range from dead-killed (No. 1) to violently rimming (No. 8). The differences between these structures are the result of the varying amounts of gas evolved by these ingots as they solidified. No. 1, a killed ingot, evolved no gas, the shrinkage which occurred on solidification causing the slightly concave top, and also the large cavities below the top, separated by bridges and together constituting the **pipe**. Dead-killed steels usually are poured in hot-topped molds, which are also usually big-end-up, thus modifying the shape and location of the pipe cavity, but the ingot shown here is better for comparison. No. 2, a typical semi-killed ingot, evolved a slight amount of gas which, however, was sufficient to compensate fully for the shrinkage during solidification, the pipe being replaced by scattered blowholes in the upper half of the ingot. Ferrostatic pressure suppressed the evolution of gas from the lower half of the ingot, and thus prevented formation of blowholes in that zone. The gas pressure was sufficient to bulge upward the solidifying top surface of the ingot.

No. 3 evolved considerably more gas than No. 2. The excess, above that required to compensate for shrinkage, formed **skin** or **honeycomb blowholes** damagingly close to the surface in the upper half of the ingot. The gas

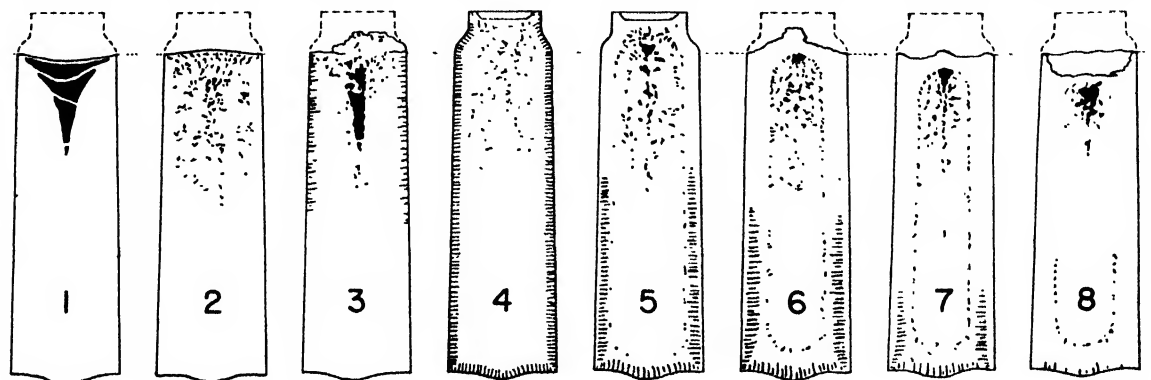


FIG. 20—3. Series of ingot structures.

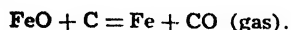
pressure ruptured the initially frozen top surface of the ingot, forcing liquid steel up through the rupture, where it then froze; this phenomenon is called bleeding.

No. 4 evolved so much gas that the top surface could not even start to solidify. Instead, so many honeycomb blowholes formed damagingly close to the surface, and from top to bottom of the ingot, that the steel rose rapidly in the mold after pouring with a boiling action, called **rimming action**, so that it had to be checked by the application of a metal cap at the top of the mold. No. 5, a typical **capped** ingot evolved still more gas; so much that the strong upward currents along the sides in the upper half of the ingot swept away the gas bubbles that otherwise would have formed blowholes. Even in the lower half of the ingot, the blowholes could not form until gas evolution had moderated somewhat, so that the solid "skin" over the honeycomb was thick enough that the blowholes, in an ingot of this type, seldom would be exposed due to scaling during heating in the soaking pit. In No. 5, owing to fewer honeycomb blowholes being formed than in No. 4, the steel rose less rapidly to the cap at the top of the mold.

No. 6 is a **rimmed** ingot, as are also Nos. 7 and 8. In No. 6, the evolution of gas, while more than for No. 5, was insufficient to keep honeycomb blowhole formation from slightly exceeding shrinkage, so that the top surface of the ingot rose slightly as it froze in from the sides of the mold. In No. 7, a typical rimmed ingot, there was a strong evolution of gas so that honeycomb formation, confined to the lower quarter of the ingot, just about balanced shrinkage, the top surface not rising or falling appreciably as it solidified in to the center. In No. 8, a violently rimming ingot, typical of low-metalloid steel, no honeycomb blowholes could form and the top surface of the ingot fell markedly as solidification proceeded.

These eight ingot structures, representing the four main types of ingots, namely, killed, semi-killed, capped, and rimmed (Nos. 1, 2, 5 and 7, respectively) together with the intermediate or extreme forms (Nos. 3, 4, 6, and 8) are merely selected samples from a continuous series of structures, varying from fully killed to fully rimmed. The gas responsible for this series of structures is a mixture, with carbon monoxide as the chief component. The other gases in the mixture probably are dissolved in the steel, while the carbon monoxide is the result of a chemical reaction taking place about as follows:

Carbon monoxide gas is formed during the cooling of the liquid steel to the solidification temperature range, and during solidification, by reaction between the iron oxide and the carbon dissolved in the steel, according to the equation:



Evolution of the gases is due to several complex, inter-related factors. Simply, however, it may be stated that: (1) At the finishing temperature, the iron oxide and carbon contents of the liquid steel are in equilibrium; as the steel cools, the equilibrium is disturbed, and these two constituents begin to react in an effort to restore a chemical balance. Since cooling is continuous, a new state of equilibrium is not attained, and gas continues to be evolved. (2) The solubility of gases in molten steel is much greater than in solid steel; consequently, most of the gas still in solution in the steel at the time of solidification is ejected from solution in the steel at the time the latter solidifies. As the steel is in a more or less plastic condition at the solidification point, the last gases evolved or liberated may not be able to escape from the body of the metal, and will collect in bubbles as all gases do in making their way out of any liquid.

Since there is sufficient carbon even in low-metalloid steel for a violent evolution of carbon monoxide, it follows that the type of ingot structure will depend chiefly on the iron oxide (FeO) available for the above reaction while the steel is solidifying in the molds. This in turn depends on the carbon content of the steel at tap (the FeO content being greater as the carbon is less) and upon the amount of deoxidizers used.

From the above explanation, it will be understood that types of ingots requiring the evolution of large amounts of gas, such as rimmed or capped ingots, cannot be produced if the carbon content is high (above 0.30 per cent), because the oxygen content would be too low for a strong evolution of gas, but enough for a mild or disturbing influence. Semi-killed and killed ingots can, therefore, be made from such steel by proper additions of substances (deoxidizers) to combine with any unwanted excess of iron oxide. On the other hand, any type of ingot structure can be produced from low-carbon steel, merely by adjusting the amount of deoxidizer used in furnace, ladle, or molds. However, in very low-carbon steels, the excessive amounts of deoxidizer required for the production of killed or semi-killed ingots add to the expense and give rise to objectionable amounts of products of deoxidation, so that there are often practical advantages in making the lower carbon grades rimmed or capped, and the higher carbon grades semi-killed or killed.

Pipe—The shrinkage cavity, or pipe, located in the upper central portion of the ingot, is largest and most deeply located in the two extremes of ingot structure represented by ingots Nos. 1 and 8. Less extreme structures, such as No. 2 (semi-killed) or No. 7 (strongly rimming) exhibit this tendency to a lesser extent, while the product of an ingot of intermediate structure such as No. 5 (capped) will be practically free from pipe after rolling. Big-end-down killed ingots (poured without a hot-top) often have the lower portion of the pipe cavity below the bridges clean enough to be welded satisfactorily during rolling. This is true particularly for the higher carbon steels or where the percentage of reduction during rolling is high, as in the lighter, flat-rolled products. In these cases, a satisfactory yield of sound rolled product often can be obtained without special steps being taken to eliminate pipe. If assurance of complete freedom from pipe is required, it is accomplished best in killed-steel ingots by making them big-end-up, with a hot-top, as shown in Figure 20—4, No. 1.

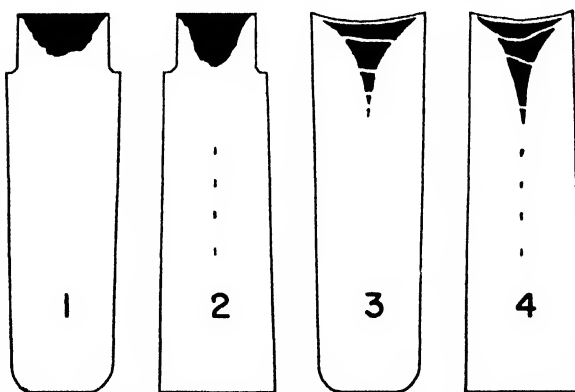


FIG. 20—4. Types of killed ingots.

1. Big-end-up, hot topped.
2. Big-end-down, hot-topped.
3. Big-end-up, not hot-topped.
4. Big-end down, not hot-topped.

The refractory material with which the hot top is constructed or lined absorbs heat less rapidly than the cast iron of the mold so that the top of the ingot remains liquid until after the remainder of the ingot has solidified, thus furnishing an overlying pool of liquid steel which feeds down into the lower portions of the ingot to overcome the shrinkage due to solidification. By using big-end-up molds, this feeding is made still more effective.

Blowholes—In all except killed ingots, the evolution of gas produces cavities of roughly cylindrical shape (skin or honeycomb blowholes) or spherical shape (located deeper in the ingot). Except for the ones located within several inches of the top of the ingot, such blowholes tend to have interiors clean enough to weld easily and completely during rolling, unless they extend to the surface of the ingot, or lie at such shallow depth beneath the surface that they become exposed by oxidation of the surface of the ingot during heating in the soaking pits. In both such cases their interiors become oxidized and they will not weld, resulting in numerous seams in the rolled product. Properly made ingots, therefore, will have gas evolution during solidification so controlled that there will be a skin of adequate thickness over those blowholes closest to the surface. The fact that blowholes serve a useful purpose in diminishing or preventing the formation of pipe already has been mentioned.

Segregation—When a complex material like liquid steel undergoes solidification, the first parts of the ingot to become solid are purer than the original liquid steel. This is called **negative segregation** and means that such solid steel is lower in carbon, manganese, phosphorus, sulphur, and other elements than the liquid steel from which it solidified. As a result of this action, the remaining liquid steel becomes increasingly enriched in these elements, so that the tendency of the solidifying steel to be purer than the original liquid steel is after a time arrested and then reversed. In this way the last steel to solidify contains notably greater amounts of these elements. This is **positive segregation**, and occurs to the greatest extent in the very last steel to solidify. This is in the top central part of the ingot, and, in killed steels, is adjacent to and just below the pipe cavity.

Some elements segregate to a greater extent than others. Sulphur is the worst offender in this regard, followed by phosphorus and carbon, while manganese and silicon show little tendency to segregate.

The tendency of the elements to segregate while solidifying increases with the time required for solidification, so that large ingots tend to segregate more than smaller ones. Still more effective in promoting segregation is any movement of the liquid steel relative to the solidifying surface. Killed steel in general, therefore, is least segregated. The slight stirring of the liquid steel caused by the evolution of gas in semi-killed steel produces a corresponding slight increase in segregation, and the rimming action of capped steel prior to capping produces a thin rim zone exhibiting negative segregation, the thickness of which depends on the time elapsing between pouring the ingot and sealing the top with a metal cap. After sealing, the evolution of gas is mechanically suppressed, and the remainder of the steel solidifies after the manner of a semi-killed ingot. However, since the thin rim layer is strongly segregated negatively, the positive segregation of the center of the ingot is somewhat greater than for a semi-killed ingot. Rimmed ingots are the most segregated of all, as the prolonged rimming action produces a thick rimmed zone of negative segregation. Since the rim zone can be almost half the volume of an ingot, it follows that the

core zone, which solidifies after the fashion of a semi-killed ingot when the top of the ingot finally freezes over, exhibits strong positive segregation. The boundary between rim and core zones is relatively sharp, and these zones are so different as regards composition and appearance after etching as to resemble different steels.

There are certain other special aspects of segregation in killed steel which are of interest, but can only be mentioned here; these include **axial porosity** (associated with the "V segregate" along the central axis of an ingot) and **ingot pattern** which may be due to the ingot being disturbed while solidifying, or to the type of segregation called "inverted V segregate."

Columnar Structure—Steel after solidification is a crystalline material, and the first molten metal to strike against the comparatively cold mold wall freezes rapidly into a structure characterized by small and randomly oriented crystals, which occasionally may persist (in small ingots) to the center of an ingot, and which tend to become larger as they approach the center of the ingot. Usually, however, this initial zone of randomly oriented crystals soon gives way to large crystals called **dendrites**, characterized by a branching structure developed as the crystals "grow." The principal direction of growth is perpendicular to the mold wall, so that all dendrites so formed have essentially the same orientation. If growth of the individual dendrites occurs principally along their longitudinal axes perpendicular to the surfaces of the ingot, these large elongated crystals, under suitable circumstances, may extend all the way to the center of the ingot. An ingot having a preponderance of these large elongated crystals is referred to as possessing a **columnar structure** and, if the structure is exaggerated in extent, it is referred to as **ingotism**. Ingots exhibiting ingotism tend to crack excessively during rolling unless light drafts are employed for the first few passes in the rolls. In most ingots, however, columnar structure gives way, toward the center of the ingot, to rather large, equiaxed, randomly oriented crystals, which also are dendritic in character. The relative proportion of columnar and equiaxed dendritic crystallization appears to be dependent upon many variables, among which are composition of the steel, mold temperature, pouring temperature and gas content of the steel.

Internal Fissures—Tensile stresses in the interior of an ingot, arising during cooling, heating, or rolling of the piece, may produce **internal fissures** or **internal bursts**, sometimes of a very large size. If these do not extend to the surface they will weld completely during the rolling operation, provided the amount of hot work (percentage of reduction) is sufficient.

✓ **Ingot Cracks**—If the fractures produced by tensile stresses extend to the surface or originate at it, they produce the visible defects called **ingot cracks**. Such exposed cracks become oxidized, producing large seams in the rolled product. This type of defect receives more detailed attention in Chapter 26 on steel conditioning.

Non-Metallic Inclusions—All steel ingots contain more or less non-metallic matter, consisting almost exclusively of oxides, with lesser amounts of sulphides, in various combinations and mixtures with each other. They are derived chiefly from the oxidizing reactions of the refining process and the deoxidizing materials added to the steel in the furnace, ladle, or molds. Some result from erosion of ladle refractories during pouring. They were supposed formerly to be admixed furnace slag, but this material now is recognized as only a minor source of these inclusions.

Scabs—In top-poured ingots the liquid steel, falling

from a height upon the stool or the bottom of the mold, splashes violently against the lower part of the mold walls. Many of these splashes adhere and solidify, forming a continuous layer on the bottom foot or two of the mold walls. Above this point the adhering splashes rapidly decrease in number and become discontinuous. This splashing diminishes as a pool of liquid metal forms in the bottom of the mold. The adhering splashes cool rapidly, and their surfaces oxidize. If the cooling and oxidation have progressed too far by the time the liquid steel in the mold rises past them, they will not be incorporated into the ingot, but remain as adhering and imperfectly bonded scabs on the surfaces of the ingot which, if thin, may be oxidized away in the soaking-pit operations. If thick, they produce a similar defect on the rolled product, as discussed in Chapter 26. As the continuous layer of splashes cools, its upper edges tend to bend inward and, as the rising liquid steel overflows them, to become enfolded. Horizontal ingot cracks, called **butt cracks** often occur below and parallel to such folds, and the folds themselves can produce surface laminations or seams in the rolled product.

The defects associated with pouring splashes can be reduced by filling the mold more rapidly, so that the rising level of liquid steel covers the splashes before they can cool and oxidize. This is done by using larger or multiple nozzles, which practice, however, gives rise to various mechanical difficulties if carried to extremes.

There is also a danger, especially with higher pouring temperatures, that high pouring rates will lead to greater ingot cracking.

Bottom pouring also will minimize this defect, since the molten steel enters the mold from a runner through an opening in the mold bottom and there is little splashing as compared with top-pouring practice.

Mold Coatings—Another method of improving ingot surface is to coat the inside of the molds with a substance which tends to repel splashes. Many substances have been proposed or tried for this purpose. Among these few are tar, powdered pitch, gilsonite, graphite, and aluminum paint. The last often is used on molds for killed steel of the higher-alloy grades. Tar is probably the most effective splash repellent, but, if used under unsuitable conditions, gives rise to annoying fumes. Powdered pitch has some of the effectiveness of tar, and the fumes are less objectionable. Gilsonite approaches tar in effectiveness and produces a minimum of fumes when properly applied. With carbonaceous coatings, such as tar, powdered pitch, or gilsonite, the temperature of the molds at the time they are coated is important. If they are too hot, the coating is decomposed and the residual charred film has no beneficial effect. If they are too cold, the coating is extremely heavy, and the excessive amount of gas accompanying its decomposition by the liquid steel gives rise to subsurface blowholes in the ingot.

SECTION 3

CONTROL OF INGOT STRUCTURE

Steelmaking and Deoxidation Practices—The previous discussion has shown that the final structure of an ingot is determined almost entirely by the degree to which the steel from which it was cast has been deoxidized. The several types of steel require different steelmaking and deoxidation practices, which are described briefly in the following summary of the principal steps involved.

Rimmed Steels—For rimmed steels, proper rimming action in the molds has been described as necessary to produce the surface conditions and ingot structure desired. Slag control is aimed at adjusting the lime-silica ratio and iron-oxide content of the slag to give the desired level of oxidation of the bath of metal when the heat is ready to tap. The exact procedures followed depend on whether the steel has a carbon content in the higher ranges (0.12 to 0.15 per cent), in the lower ranges (0.06 to 0.10 per cent), or under 0.06 per cent.

Rimmed steel usually is tapped without deoxidizing additions to the furnace and with only small additions in the ladle, in order to have sufficient oxygen present to give the desired gas evolution when reacting in the mold with carbon. This type of steel, when properly made, has a minimum of pipe and a good surface though it is subject to segregation. When the metal in the ingot mold begins to solidify, there is a brisk evolution of gas resulting in an outer ingot skin of relatively clean metal. For many applications, particularly where the surface of the product is important, this steel is used to a considerable extent.

The thickness of this outer skin and the absence of blow holes and oxides from it depends on the skill of the steelmakers. When the temperature and the oxygen level of the steel, as it is poured from the ladle, are within the most desirable limits, the desired evolution of carbon monoxide from the steel as poured into the molds is obtained by the reaction of oxygen in the steel with carbon, and to finally control and obtain the most

effective rimming action, the very careful use of shot aluminum, in small quantities, is necessary as the ingots are poured. The amount to be added is determined in pouring the first few ingots. The largest proportion of this steel has a carbon content of under 0.15 per cent. Ferromanganese may be added to the furnace before tapping or in the ladle, but it is usual to make the addition in the ladle. Aluminum, ferrotitanium or other deoxidizers in small amounts may be added in the ladle, if needed. If the steel is deoxidized to too great an extent up to the point where the pouring of ingots begins, the rimming action will be incomplete because the evolution of gas is too small in volume and too slow in starting its evolution.

Capped steel practice is a variation of rimmed practice. The steel is poured into big-end-down bottle top molds in which the constricted top or mouth of the mold facilitates the capping operation. The rimming action is allowed to begin normally but is then terminated at the end of a minute or more by sealing the mold with a steel or cast-iron cap. The addition of a small amount of shot aluminum during pouring insures that the steel rises to press against the cap. The oxygen level of the steel as poured into the mold is preferred to be not more, and possibly slightly less, than the desired level for rimmed steel. The product is an ingot with a thin rim which is relatively free of blow holes and with segregation less than is usual for a rimmed ingot of the same volume. On steels (other than killed or semi-killed) of higher carbon content than 0.15 per cent, this method is used to advantage. Steel of this type is applied on sheet, strip, skelp, tin plate, wire, and bars.

Semikilled steel is deoxidized less than killed steel and there is oxygen still present in the molten steel to react with carbon and to form gas after the steel is poured into the molds. The steel finds wide application in structural shapes, plates and merchant bar. Structural

steel is generally in the range of 0.15 to 0.30 per cent carbon, because this provides appropriate strength, and such steel is generally semikilled. The usual practice is to bring the carbon content of the bath to the desired level for tapping. The manganese may be added as ferromanganese to furnace or ladle, or divided. If large amounts are necessary, at least part should be added to the furnace. Carbon, ferrosilicon, ferrotitanium and aluminum may be added to the ladle. It is usual to accomplish most of the deoxidation in the ladle, so that only a few ounces per ton of aluminum will be required in the mold.

Killed Steel—The term "killed" indicates that the steel has been deoxidized sufficiently for the metal to lie perfectly quiet when poured into an ingot mold. There is no evolution of gas and the top surface of the ingot solidifies with relative rapidity. Killed steel generally is used when a homogeneous structure is required. Alloy steels, forging and carburizing steels are of this type, when the essential quality is soundness (freedom from blow holes and segregation). In general, all steels with more than 0.30 per cent carbon content are killed. In making killed steel, the usual practice is to "catch the heat coming down;" that is, to lower the carbon content of the bath to the desired level and then either block the heat (deoxidize it) by adding high-silicon pig iron (15 to 25 per cent silicon), silicomanganese, or spiegel, or to tap the heat and depend upon ladle deoxidation.

Blocking lowers the oxygen content of the liquid metal to prevent further oxidation of carbon; it also serves to protect alloying elements which are susceptible to oxidation and consequently are added after the heat has been blocked.

At the final part of the finishing period, the carbon will have been worked down until it is at a level which

in best practice is such as to bring the steel within the required range in tapping and pouring. The phosphorus and sulphur should be below the specified maximum, the manganese usually will be below the minimum required, and the bath temperature should be proper for the composition and grade of the steel being produced. The steel is then ready for whatever ferroalloys need to be added (ferrosilicon, ferromanganese, other alloys).

The decision as to whether a ferroalloy addition is made to the furnace or to the ladle is determined largely by the susceptibility of the ferroalloy to oxidation. Manganese may be added to the furnace or to the ladle, or divided between them, but the additions to the ladle must not be so large as to chill the metal too much. The furnace additions are chosen and the timing of addition set so that the maximum elimination of the solid oxides formed will take place by floating up through the metal to the slag before the metal is tapped from the furnace. After tapping, other deoxidizing additions may be added to the steel as it runs into the ladle. These complete the deoxidation to the desired degree up to the pouring into molds. These ladle additions are usually ferrosilicon, aluminum or other special alloys (calcium-silicon is an example) containing elements which have a strong affinity for oxygen. Additions containing such elements as manganese or silicon furnish part of the elements required to meet the specified chemical composition. Additions of deoxidizers to the molds may be made, depending on the type of steel.

In producing certain extra-deep-drawing steels, a low-carbon steel (under 0.10 per cent carbon) is killed, usually with a substantial amount of aluminum, added in the ladle, molds or both.

A large proportion of killed steel is poured into hot topped, big-end-up molds.

SECTION 4

HEATING INGOTS FOR ROLLING

The main function of soaking pits (see Chapter 21) is to bring ingots to a uniform temperature suitable for rolling and to do this without damage to the ingots themselves. Between the time the ingots are poured and the moment when they are ready for rolling, the various manipulations to which they may be subjected are all for the purpose of attaining one or more of the following three objectives:

1. **To Insure Solidification of the Ingot**—Ingots of some killed steels must be allowed to become completely solid before being moved from the pouring platform. For some other killed steels, the ingots can be moved away from the pouring platform within a short interval after pouring, say 20 minutes, but not thereafter until completely solid. For all other types of steel, there are usually no holding requirements from the standpoint of internal quality. Such ingots may, therefore, be moved, stripped, and charged into the soaking pits as soon as solidification has progressed far enough to enable them to endure the stresses involved in such handling.

To save time and fuel, therefore, ingots of many grades of steel are charged soon after pouring, when their interiors are still largely molten. If the heating of such ingots should be started immediately after charging, solidification would be retarded to such an extent that the center portions of the ingots would still be liquid, or at least in the mushy stage of solidification, when the remainders of the ingots were ready to roll. In such cases, therefore, the pits may be dampered for

a period after the ingots are charged; that is, the fuel is shut off and the connection to the waste-gas flue is closed, the latter to prevent natural draft from causing cold air to pass through the pit. During such a dampering period, the exterior portions of the ingots do not receive much heat from the pit, while the centers of the ingots continue to solidify by transferring heat to the exterior surfaces. The duration of such dampering periods should, of course, increase with increasing ingot size, especially as determined by the least dimension (thickness), and decrease with increasing "track time," i.e., the interval between finish pour and charge.

An alternative method of insuring complete solidification of the ingots after charging and before rolling is to set up minimum track times, which increase with increasing ingot thicknesses.

2. **To Avoid Rupturing the Surfaces or Interiors of Ingots**—Ingots of some grades of steel are notoriously more sensitive to thermal shock than those of other grades and can, therefore, be damaged or ruined by cracks or internal bursts if cooled or heated at too rapid a rate. This sensitivity increases with ingot size, degree of killing, carbon content, and metalloid or alloy content. Thermal shock by cooling may occur between pouring the ingots and charging them into the pits, especially in the interval between stripping and charging.

In certain grades of steel, such as high-sulphur screw stock, it is advisable to make the time between pour and charge, and especially the time between strip and

charge, as short as possible. This particular type of sensitivity perhaps may be due to something else than simple thermal shock, as, for example, the formation of an intercrystalline precipitate within a critical temperature range. If the charging of such steels is to be delayed too long, it may be better to allow the ingots to cool further in the molds before stripping, charging, and reheating.

Thermal shock by heating may occur if ingots are charged into pits which are much hotter than the ingots or if, after charging, the ingots are heated too rapidly. The more sensitive the grade of steel, and the colder the ingots at the time of charging, the more elaborate must be the precautions against such thermal shock. For example, with large cold alloy ingots, it may be necessary to cool the pits until they are not hotter than 800° F, then, after charging the ingots, leave the pit uncovered for some definite period, then cover and let stand for another period, after which firing is begun at a minimum rate, the heat input being controlled carefully and stepped up very gradually.

3. To Bring All Parts of the Ingots to the Desired Temperature—The temperature at which ingots properly heated for rolling are withdrawn from the soaking pit is called the **drawing temperature**. The heating of ingots to this temperature may be considered as taking place in two successive steps, although, for small ingots charged quite hot, the second step occurs so nearly simultaneously with the first step that its existence is not so obvious.

a. Heating the Exterior Portions of the Ingots to Drawing Temperature—To achieve this first result, the pits are fired as rapidly as is safe, having due regard to avoiding thermal shock to the ingots, and to avoid any **burning** of the ingots (see below). Depending upon the type of pits, heat input is controlled by visual observation of the surfaces of the ingots, or by thermocouples recording pit temperatures. These latter usually are arranged to proportion, or at least to regulate, heat input in accordance with pit temperatures, the controls being set to prevent the pits from exceeding a predetermined temperature. This temperature limit has been determined by previous experience to result in the desired drawing temperature.

b. Heating the Interior Portions of the Ingots to Drawing Temperature—A portion of this second result is, of course, being achieved while the exteriors of the ingots are being heated to drawing temperature, and the remainder of this result is secured after the

exteriors of the ingots have reached drawing temperatures. This is accomplished by reducing the heat input (in automatically controlled pits) until it is just sufficient to maintain the exteriors of the ingots at drawing temperature. The heat input at this stage is, therefore, only that required to offset heat losses from the pit and the heat absorbed by the interiors of the ingots. In manually fired pits, fuel input can be reduced by hand control during this soaking period, or an approximation to this controlled reduction of heat input can be secured by alternately firing and dampering the pits for short periods of time.

Since the temperatures of the interiors of the ingots cannot be measured, past experience is the only guide as to the time required to "soak out" the centers of the ingots, taking into account their temperature at the time of charging, their size, and their thermal conductivity.

"Burning" of Ingots—If ingots are heated to excessive temperatures, the surface scale may melt and run. Such ingots are said to be **washed**. If the areas of melted scale on ingots are small, due to localized flame impingement, the ingots are said to be **port marked**. In addition to these effects, excessive pit temperatures can produce the damaging effect called **burning**. This is a complex phenomenon, which can occur at temperatures below the solidus of the steel being heated. The low-carbon grades of steel are almost immune to this trouble, which becomes an increasing hazard as the carbon content of the steel increases, particularly above 0.30 per cent. It is a penetration of iron oxide, usually in combination with iron sulphide, between the grain boundaries at and near the surface of the ingot. It results in scabby, crumbled surfaces on the rolled products (see Chapter 26).

Burning can be avoided by careful control of the temperatures attained by the surfaces of the ingots.

Overheating of Killed and Semi-Killed Non-Hot-Topped Ingots—In these two kinds of ingots, the yield of sound rolled product depends upon the extent to which pipe cavities are welded by the rolling operation. If the top portions of the ingots are heated to too high a temperature, the pipe cavities will not weld successfully. Apparently, the high temperature liquefies non-metallic matter which is present on or near the tops of the ingots, and this then finds its way down into the pipe cavities. Cold ingots particularly are subject to this trouble, and special care in heating is required to minimize pipe rejections.

Chapter 21

CONSTRUCTION AND OPERATION OF HEATING FACILITIES

SECTION 1

PRINCIPLES OF FURNACE DESIGN

Objectives and General Metallurgical Requirements—

A heating furnace is utilized to raise the temperature of steel for hot working (shaping) and for heat treating. Heating furnaces may be divided into three general classes:

1. Soaking-pit furnaces
2. Reheating furnaces
3. Heat-treating furnaces

The function of soaking pits and reheating furnaces is to raise the temperature of steel in the course of processing until it is sufficiently hot to be plastic enough for economic reduction by rolling or forging to the desired section. The function of heat-treating furnaces is to heat the steel to some specific temperature for the purpose of obtaining the desired mechanical properties, which may be developed either by: (a) regulating the speed of cooling of the steel in the furnace; (b) by removing the hot steel from the furnace and permitting it to cool in still or agitated air; or (c) immersing the work directly from the furnace into some liquid to cool it suddenly and thus develop maximum hardness after which, usually, the hardened steel is tempered by further heat treatment.

Heating furnaces must be constructed of suitable materials to withstand the effects of the temperature levels at which they must operate. They must be provided with charging facilities which are adequate for the material size and handling rate, and with the proper means for heating the steel at the specified production rate. From the metallurgical standpoint, all three types of furnaces must be constructed to heat the steel uniformly and, by suitable temperature and combustion control instrumentation, hold it at the desired temperature for a specified length of time. Heat-treating furnaces, in addition, may have to be designed to cool at controlled rates to a predetermined level, and be provided further, in many cases, with extra equipment by which special requirements, such as the control of furnace atmosphere for developing or maintaining the desired surface condition of the steel, may be met.

Basic Elements of Furnaces—There are many different designs for each of the three general furnace classes noted above, but each design (exclusive of salt-bath or lead-bath furnaces, which are discussed in detail in Section 4 of this chapter) consists of certain common parts, as:

1. The heating chamber; an enclosure to contain the material and retain heat.
2. A hearth or support in the furnace for carrying the charge.

3. Facilities for the development of heat to raise and maintain furnace temperature.
4. Means for the distribution of heat and the removal of spent gases from the furnace.
5. Means for the introduction of work to be heated and removal of heated stock.

The enclosure to contain the material and heat generally is called the furnace proper. It usually is constructed of refractory material, although furnaces for operation at relatively low temperatures may be fabricated exclusively of steel. The furnace hearth or support for the charged material also may be constructed either of refractory or metallic material. In high-temperature furnaces, metallic supports generally are water cooled. Furnace hearths are constructed to permit the charge either to remain in a fixed position in the furnace or to be moved during the heating process. An example of the first type of hearth is the conventional batch-type or "in-and-out" furnace. Examples of the second general type are found in roller-hearth furnaces in which the material moves as the series of rollers that constitute the hearth rotate, and in continuous furnaces in which a continuous line of material is pushed over skids. The combustion of fuel usually is employed to develop the required furnace temperature, but the conversion of electrical energy into heat is used also in some important heat-treating furnace installations. The circulation of heat in the majority of furnaces is accomplished by natural convection and stack draft; in others, by forced circulation.

Facilities for the introduction and removal of heated stock vary with the type and size of furnace, the size and shape of the stock to be handled, and the general layout of the furnace and auxiliary facilities. Roller tables, conveyors, charging machines, overhead cranes, furnace pushers and pinch rolls are the principal kinds of equipment used for this service.

Furnace Size and Capacity—The size of a heating furnace usually is described by its hearth area. The hearth areas of the various types of the three general furnace classes differ greatly. Soaking pits may have a hearth area of from 10 sq. ft. to something over 300 sq. ft. Reheating furnaces may have hearths ranging in area from only a few square feet, as in a small forge furnace, to over 2000 sq. ft. in large, continuous slab-heating furnaces. Heat-treating furnace hearths vary in size from the laboratory type of furnace of only a few square feet to areas of 1600 sq. ft. or more found in some of the larger continuous normalizing or annealing furnaces. The productive capacity of a heating furnace often is related to its hearth area, and figures of from

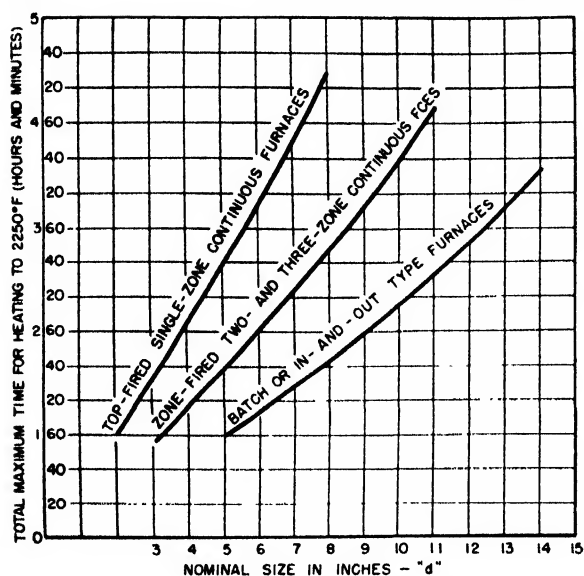


FIG. 21-1. Normal average time for heating various sizes of square mild-steel (low-carbon) and medium-carbon steel billets from room temperature to rolling temperature of 2250°F.

2 pounds to 250 pounds per sq. ft. of hearth area per hour are quoted for the various types. The low rate may apply for some conditions in heat treating steel and the high rate to pit furnaces when heating already hot ingots to a higher temperature. The capacity of a furnace is determined primarily by the area of the surface of the piece to be heated which is exposed to the furnace temperature, and the shape, thickness and composition of the material, its temperature and that of the furnace, and the emissivity of the heat source and of the material to be heated (e.g., surface-ground or machined stainless steel). The desired rate of heating is regulated by manually or automatically controlling the rate of heat input to the furnace.

High-carbon steels and heat-resisting alloys require longer heating cycles to attain uniformity of heating at the same temperature levels, as compared to low-carbon steels. Selection of the time required for properly heating various grades and sizes of steel is based largely on experience with given furnaces heating specific types of products. Figure 21-1 shows the normal time for heating various sizes of cold mild steel and medium-carbon steel blooms and square billets to rolling temperature. The heating time for high-carbon steel is about one-third longer and for heat-resisting grades about twice that shown.

The flow of heat through a thick body of steel is relatively slow compared to surface absorption in high temperature furnaces and, therefore, caution must be exercised in regulating the supply of heat to prevent the surface from "sweating" (partial melting) while bringing the temperature inside the material up to the required level. Figure 21-2 shows the average rate of heat absorption for an entire cycle as generally practiced in heating mild, or medium-carbon grades of steel from atmospheric to rolling temperature. During the early stages of the heating cycle, the heat-transfer rates are considerably above the average shown, while during the latter part of the cycle the rate is very low. In heat treating steel, considerably lower average heat-transfer rates are practiced than those shown in Figure 21-2.

Furnace Type and Shape—There are many types of each of the three general classes of heating furnaces. The selection of type is determined by its suitability for heating economically particular grades, shapes and sizes of the material at the rate and to the temperature level desired. For instance, batch-type furnaces are especially suitable for heating blooms of mixed sizes and lengths in thicknesses over 8 inches; continuous furnaces are used for heating slabs or billets for large orders of uniform length and thickness, and car-bottom furnaces are used for annealing miscellaneous shapes and sizes. The general shape of a furnace depends upon a number of factors, such as capacity desired, space available, auxiliary equipment and metallurgical requirements in heating. Refinements in furnace lines depend within rather wide limits on the kind of fuel used and on the grade and size of steel to be heated. The desired combustion space, temperature level requirements, uniformity of heating and fuel flow are major considerations in furnace design.

Thermal Efficiency—The thermal efficiency of heating furnaces varies considerably because of differences in the temperature level of the heated stock and of the charged material, in the provision of heat recovery equipment such as regenerators and recuperators, in furnace insulation, in operating schedules, and in heating requirements. To cover the full range of all common types of heating furnaces, the thermal efficiency may fall anywhere between 5 per cent and 60 per cent. Large production-line furnaces, such as continuous furnaces equipped with recuperators and insulation, generally give 30 per cent to 40 per cent thermal efficiency over an average month's period of operation. Small shop furnaces, poorly loaded, with no insulation or heat-recovery facilities, have low thermal efficiency. The heat requirement per ton of heated product from heating furnaces for the production line varies from 300,000 to 3,000,000 Btu. The lower figure is obtained when heating hot steel, the higher one with poorly-loaded furnaces heating cold steel.

The sensible heat lost in stack gases is the principal source of heat loss in a fuel-fired furnace. Other losses include the heat loss by conduction through furnace walls, hearth and roof; radiation through furnace openings and from the outer surface of the furnace proper;

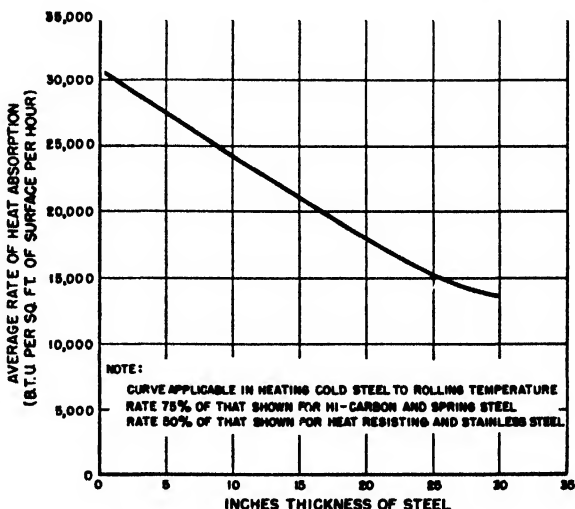


FIG. 21-2. Curve showing average rate at which heat is absorbed during an entire heating cycle by mild-steel and medium-carbon steel in heating from room temperature to rolling temperature in reheating furnaces.

the heat absorbed by water-cooled furnace parts; and the latent heat and unburned combustibles in stack gases.

In Chapter 3, dealing with fuels, the means employed for reducing furnace losses and the salvage of heat in waste gases already have been reviewed. Regenerators are provided usually for batch furnaces operated at high-temperature levels. They provide, in addition to heat salvage, a reservoir of potential heat for equalizing the temperature of steel during the soaking period. Ordinarily, in regenerative batch-type furnaces, the actual time for firing fuel amounts to only 50 per cent of the total time from charge to charge; the balance of time is taken up by charging, drawing and soaking. Firing in regenerative furnaces is usually at a constant rate with intervals of soaking, which occur more frequently as the charge approaches rolling temperature.

The heat stored in thick furnace walls of ordinary firebrick at high hot-face temperatures is considerable. When the so-called flywheel effect of hot walls and regenerators is not desirable, such as in furnaces which occasionally must be cooled as quickly as possible to lower than usual operating temperatures to receive special alloy or high-carbon heats, a material saving in production, fuel economy and maintenance can be effected by constructing the walls and roof of insulating rather than of regular firebrick.

Recuperators generally are supplied for high-temperature heating with continuous reheating furnaces and soaking pits, and in some instances for batch-type furnaces. They are more desirable than regenerators when the control of atmosphere and a constant flow of fuel into the furnace is important. They usually are designed to provide a lower preheat temperature than is obtainable with regenerators.

Materials of Construction—The temperature level carried in various parts of the furnace determines the kind and grade of construction materials that must be used. The hot end of continuous and regenerative batch furnaces, which employ burners or a fuel developing intensive combustion, may reach temperatures up to 2800° F. For this level, it is customary to construct the walls and roof of first-quality or super-duty firebrick and to water-cool any metallic parts which are exposed to high temperature. In high-temperature furnaces, there are applications for the use of special refractories.

Hearths are constructed usually of refractories resistant to abrasion, slag attack or adherence to the steel being heated. Door jambs are made of refractories with non-spalling characteristics. Pier walls, such as those used in top- and bottom-fired continuous furnaces, use refractories with good hot-load-bearing properties. In pit furnaces, slag-resistant refractories are used in the lower wall areas to combat chemical attack by cinder.

Flue temperatures, especially following recuperators or regenerators, seldom exceed 1600° F, and for this application second-quality firebrick generally is used. Cast iron or heat-resisting steel is used for constructing dampers up to this temperature level; in flues subject to higher temperatures, the dampers are water-cooled. Annealing and normalizing furnaces seldom are operated in excess of 1700° F, and tempering or drawing furnaces normally operate at even lower temperatures. In these furnace types, insulating firebrick is used generally, especially when the operations are intermittent. In many heat-treating furnaces which require a careful control of the furnace atmosphere, particularly in those where special inert gas is used for this purpose, the heating chamber proper consists of a leakproof muffle of heat-resisting steel in which the work is sealed and surrounded by the controlled atmosphere, while heat is supplied to the outside of the muffle and transferred to the work through the muffle walls.

Besides the refractories and metallic parts directly exposed to internal furnace temperature, many furnaces contain a number of other essential parts which must be given careful consideration due to the temperatures to which they will be subjected. Cast-iron doors and door frames usually are lined with refractory material for protection; roof hangers are cast of heat-resisting metal; furnace casings and steelwork usually are constructed of ordinary grades of carbon steel; regenerators generally utilize refractories; recuperators are constructed either of refractory or metal, depending upon the temperature level at which they are to operate. Various grades of insulating material are used, dependent upon load, location and temperature level. Insulation for reheating-furnace walls, roof and hearth, not only provides a saving in fuel but also aids in maintaining a uniform temperature within the furnace, reduces stresses in furnace brick and steelwork, and improves working conditions around the furnaces.

SECTION 2

SOAKING-PIT FURNACES

Introductory—The modern soaking pit has been developed to provide uniform heating of ingots to the desired temperature with a minimum of over-heating of the surface. In most modern designs, this is accomplished with automatic controls. The normal range for heating ingots is between 2150° F and 2450° F. The proper temperature level varies with grades of steel and sizes of ingots and characteristics of the rolling mill. Low-speed mills with many passes require the higher level of heating for certain grades of steel. Soaking pits serve the dual function of heating and acting as a reservoir to correct irregularities in the flow of ingots between the steelmelting shop and the primary rolling mills. Briefly, soaking pits are deep chambers, or furnaces, of square, rectangular or circular shape, into which ingots are placed in an upright position through an opening at the top (Figure 21—3). A removable cover closes the pit opening. A series of pits, installed usually in rows, are placed under cover of a building adjacent

to the entering side of the blooming or slabbing mill to be served. The top of the pit is usually several feet above ground level. The pits are spanned by one or more electrically operated traveling cranes equipped with a traveling hoist for charging the ingots into the pits and for lifting them out as they are needed by the mill. The lower end of this hoist is provided with adjustable tongs, by which an ingot may be grasped at the top and moved vertically a distance greater than the depth of the pit. This crane is used to transfer the ingots to the pits from the cars on which they were brought from the stripper, moving on tracks usually located along the side of the pits. For heated ingots, a pot car or ingot buggy is provided, which usually is propelled electrically along a track leading to the primary-mill tables, upon which it automatically deposits the ingots.

Types of Soaking-Pit Furnaces—There are several modern designs of soaking pits. Each design has special heating characteristics. The oldest of the modern types

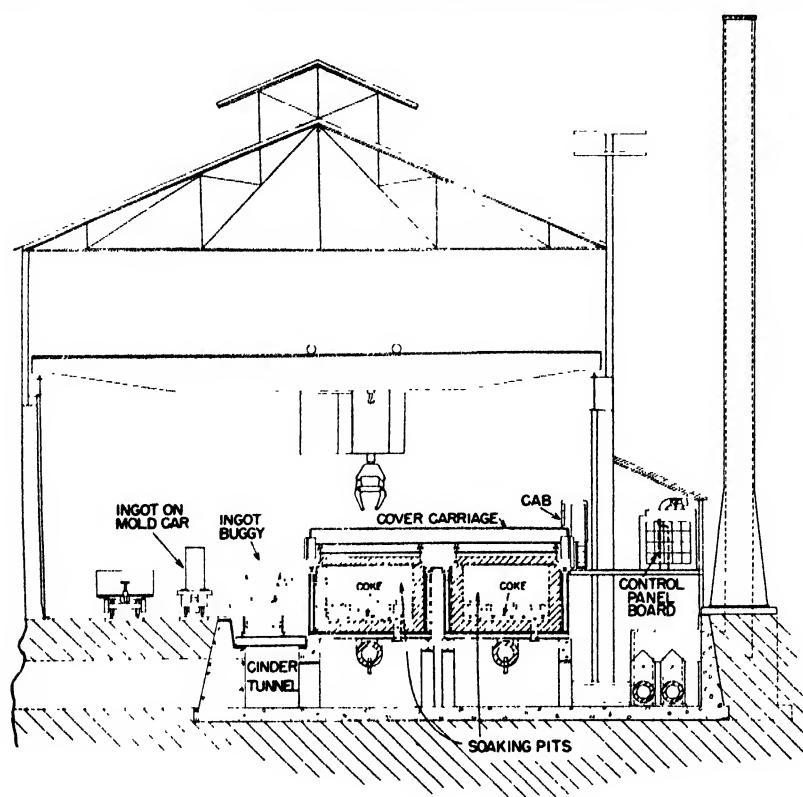


FIG. 21-3. Schematic cross-section through a soaking-pit furnace building. (Courtesy, Amsler-Morton Company.)

is the regenerative pit. In this type, the ingots are heated by alternately burning the gas through a port in the pit wall on one side, permitting the products of combustion to pass across the pit and out through the regenerator flues and stack to the atmosphere. The air, after each reversal, is passed through the hot regenerators to provide preheat for combustion of the fuel. If fuels of high calorific value, e.g., fuel oil, natural gas, or coke-oven gas are burned, they are introduced either through pipes in the top of the checker chamber and directed toward the bridgwall, or through a well in the checker brickwork adjacent to the port bridgwall. In some installations, particularly when cold blast-furnace gas is used, burners have been installed in the rear walls of the checker chambers. Since the ports are located in the endwalls of the pit, the ingots are exposed to conditions of unequal heating on their opposite sides. To equalize ingot temperature, the practice of firing and dampering is generally followed. This practice provides uniformly-heated steel at a relatively-fast heating rate, but generally causes high scale losses, due to the flow of air through openings in the flues or checker chambers into the pit proper and out through holes in the pit cover or through faulty cover seals each time the pit is dampered. Corrective measures have been applied in modern regenerative pits by careful sealing of the checker-chamber brickwork and the installation of self-sealing covers. Difficulties from scaling are not confined only to regenerative-fired pits, but also may be met in recuperative-fired pits if there are leaks in the recuperators.

One of the first major steps towards improvement in pit-furnace design was to provide sufficient space for the combustion of fuel. Pits of a continuous-fired design, known as one-way fired pits (Figure 21-4), equipped with recuperators, were designed to provide combustion space above the ingots, where the space available

for combustion was not affected by ingot coverage. "Ingot coverage" is a term used to denote the tonnage or number of ingots charged into a pit. Instead of the horizontal flow of gases through the pit as in the regenerative type, the flow in this type is vertical in accordance with hydrostatic principles.

In another type, designated as the bottom center-fired or vertically-fired pit, a section of which is shown in

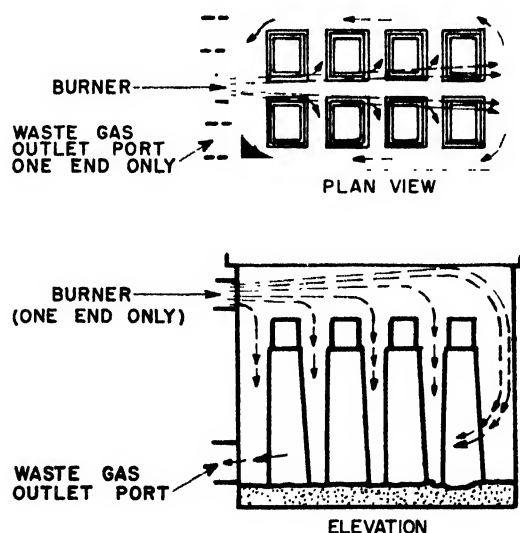


FIG. 21-4. Diagrammatic elevation and plan view illustrating principle of continuous firing and flow of hot gases in a "one-way fired" soaking pit. Hot gases from outlet ports pass to recuperators (not indicated).

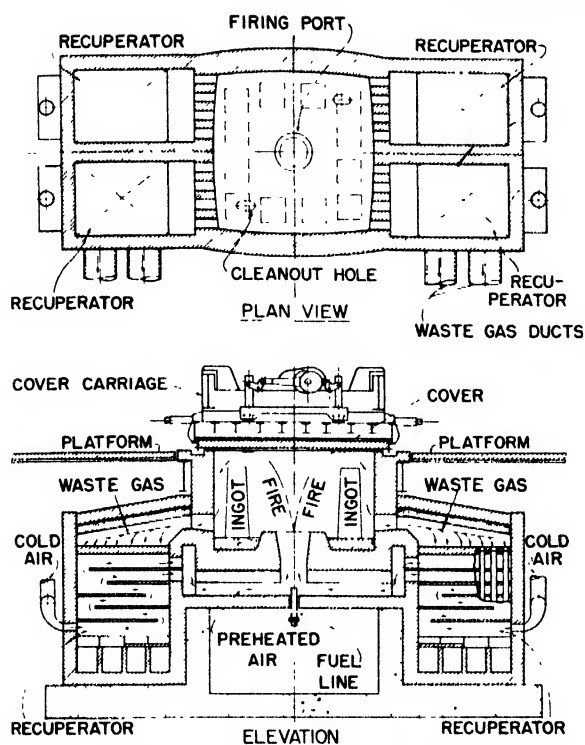


FIG. 21-5. Schematic elevation and plan view of a "vertically-fired" soaking pit, with the burner opening in the center of the pit bottom. (Courtesy, Amsler-Morton Company.)

Figure 21-5, the fuel is fired vertically through a port, centrally located in the bottom of the pit, around which the ingots are placed. As the products of combustion rise in the combustion zone, some of the spent gases which are moving downward around the ingots and next to the furnace walls are drawn into this inner column of hot rising gases, and the resulting good circulation

equalizes furnace temperature. Flues are located at the bottom corners of opposite sides of the pit to remove products of combustion and to aid in heat distribution. This design was introduced with recuperators and a full quota of controls for carrying out program heating. Adequate provision for combustion of fuel and careful sizing of pits, of nearly square shape, to suit loading conditions are incorporated in the design.

Another type, which is called the **circular pit** (Figure 21-6) employs tangential firing from a series of recessed burners located in the lower periphery of inclined side walls, to permit unusually long travel of the gases and to induce recirculation of the spent gases before they leave the pits through a centrally-located exit port at the bottom of the pit. This design utilizes a method for tempering the flame and securing uniform pit temperatures through gas circulation. These pits normally are fired with a high-calorific fuel and cold air in premix or nozzle-mix type burners to obtain complete combustion of the fuel before the gases come in contact with the ingots.

Still another pit is a refinement of the continuous-fired pit furnace, known as the **bottom two-way fired pit** (Figure 21-7). In this design the pits are fired by burners located in opposite endwalls about two feet above the bottom of the pit. The waste-gas ports are located in the same endwalls at each of the four pit corners and the gases on each end go directly to a recuperator. Combustion of the fuel takes place in a centrally located aisle, at the sides of which the ingots are placed. The method of firing and the position of the burners and waste-gas ports provide turbulence to the flow of gases in the pit, and results in improved heating of the ingot bottoms.

The **top two-way fired pit** is a deep rectangular pit in which the fuel is fired from opposite ends into a combustion space above the ingots (Figure 21-8). The burners are set to fire horizontally at an angle to the centerline of the pit to obtain a swirling motion of the gases. Outlet ports are located in the endwalls just above the cinder line. Long-flame burners are used generally to distribute properly heat from combustion above and between the ingots. The flow of gases is ver-

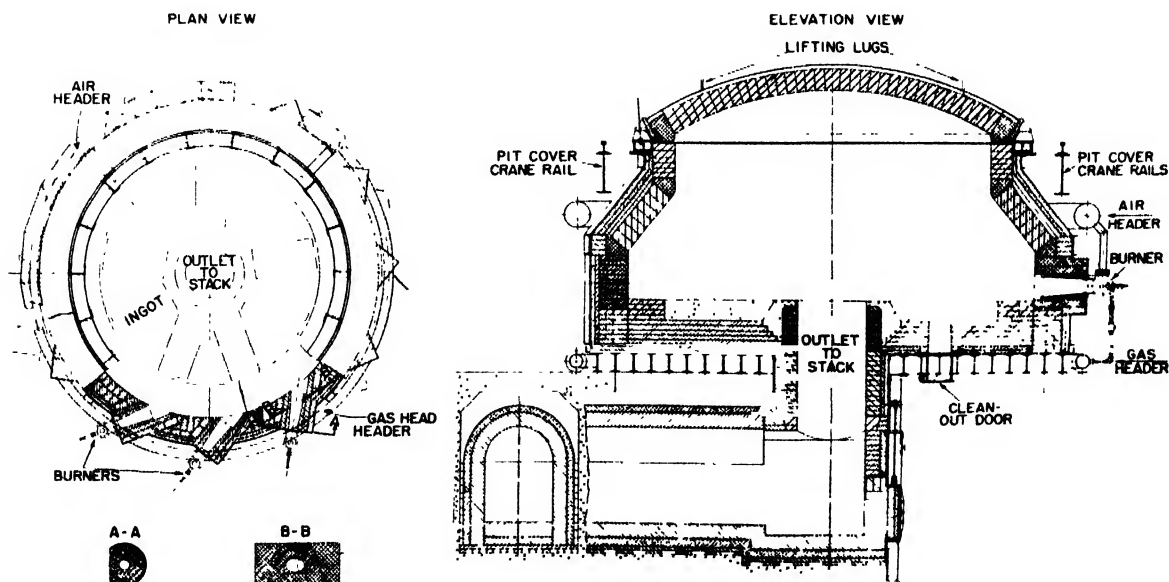


FIG. 21-6. Simplified section and plan view (not to same scale) of a tangentially-fired soaking pit. (Courtesy, Salem Engineering Company.)

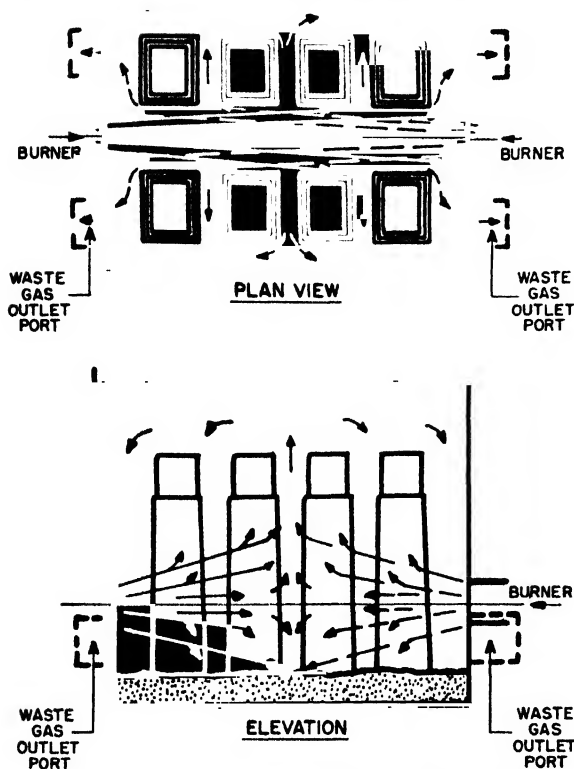


FIG. 21-7. Schematic diagram showing principle of firing and flow of gases in a side-fired or bottom two-way fired soaking pit. Gases from waste-gas ports go to a recuperator (not shown).

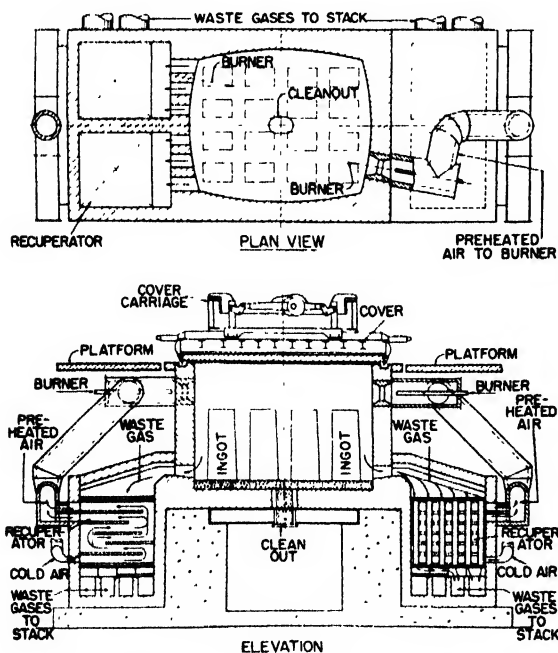


FIG. 21-8. Diagrammatic elevation and plan view of a top two-way-fired soaking pit. Left-hand section of plan view is through waste gas ports; right-hand section through one of the burners. Left-hand half of elevation is section through air passages of recuperator; right-hand, through waste-gas passages. (Courtesy, Amsler-Morton Company.)

tical, similar to that in the one-way fired pits. In this design, the method of firing allows the pit shape to be selected that will give the desired coverage; it may be built long and narrow or square. Curved (elliptical) sides and endwalls, now common to the design of most square and rectangular types of pits, are utilized also in these pits.

Auxiliary Facilities—The principal pit-furnace auxiliaries are pit covers, ingot pit cranes, facilities for ingot delivery to and from the pits and for cinder removal, along with the necessary instrumentation and controls for fuel and air supply and draft regulation.

Pit covers are constructed in various ways, the essential parts being a metal framework to support the refractories and means for quickly removing the cover replacing it upon the pit. In the older pits, the metal framework usually was made of cast iron, and the bricks were slightly arched for support. The covers were supported upon rollers or wheels which moved over tracks, originally by the use of hydraulic cylinders and later by electric motors. A depression in the track permitted the cover to drop, when in position directly over the pit, and seal the pit opening. This design, however, seldom provided a tight seal at all points and leakage of gases caused high cover maintenance. More modern design consists essentially of a heavy steel frame which supports a suspended arch of high-grade firebrick. They are equipped with either self-actuating motors for lifting the cover vertically to free it from the sand seal and then move the cover sideways to the desired pit opening, or with a cover crane to effect similar movements. The cover crane spans a row of pits and serves several. The steel frame of the modern pit cover is protected on its under side by heat-resisting-alloy castings. These castings are provided with a lip which penetrates into the sand seal. The sand seal, located around the periphery of the top of the pit, is a gutter-shaped space filled with sand. The suspended arch of a modern cover usually is backed up with insulation to reduce radiation losses.

Ingot Pit Cranes—In modern plants, the ingots are handled from the ingot delivery cars to the pits and from the pits to the ingot buggy by pit cranes. These cranes are of the conventional electric overhead design except for the trolley and position of the crane cab. Instead of the conventional hoist drum or drums, the trolley consists of the usual track wheels and a frame supporting a vertical housing in which travels a ram equipped at the bottom with mechanically operated tongs. The crane cab is attached to the lower end of the vertical housing and moves with the trolley. This arrangement of the crane cab is to provide maximum vision for the operator in order that the ingots may be placed in proper position in both pits and ingot buggy without damage to either. The modern pit crane also is used to handle coke breeze for bottom making and in many installations for actually making bottom.

Ingot delivery facilities consist usually of a track and ingot buggies. The ingot buggies which deliver the ingots to the pit are the same ones which held the molds for casting at the steel melting shop. The ingots usually are stripped of molds at the stripper, but in some cases where big-end-up ingots are used, the molds are only loosened at the stripper and are later removed by the ingot pit cranes, thus reducing heat losses enroute to pits. The layout of ingot-delivery tracks to avoid crossings and switches which interfere with other plant movements is very important. Minimizing delay in the delivery of ingots from the steelmaking furnace pouring platform through the stripper and the pit furnace building into the pit furnace not only conserves heat

but provides better metallurgical control of quality and less conditioning cost of the mill product.

Mill delivery facilities consist usually of an electrically-operated ingot buggy or pot car which runs directly between the pits and the mill approach table. There are two modern types of ingot buggies. On one type, the contact of the buggy with a cam arrangement at the mill approach table tilts the cradle of the buggy and throws the ingot from its vertical position in the buggy to a horizontal position on the mill approach table. The other type, in order to avoid battering the mill tables with heavy ingots and the necessity of approaching the tables at a high rate of speed to throw the ingot onto the table, is a self-tipping type. This design has a motor for driving the track wheels, for tilting the cradle, and for turning the table rollers which deliver the ingot in a horizontal position to the approach table.

Cinder Removal Facilities—Pit bottoms are made up with coke breeze to a depth of approximately 12 inches to 16 inches. When this breeze becomes burned or contaminated with scale, refractory or other material, it is removed through cinder holes, of which there are usually two, located in the bottom of the pit. In modern practice these holes, provided with gates at the outer end, discharge the cinder into a box located in the cinder alley under the pit bottom. The cinder box often is supported by a lift tractor or car which carries the box through the cinder tunnel to a hoist where it can be lifted to yard level and dumped into broad-gage cars for disposal. Some plants are equipped with an underground narrow-gage track system for moving the car to the cinder hoist. Bottoms for the older pit designs generally are made up each day; in modern pits they are made up only every 5 to 7 days under normal conditions.

Fuel, air and draft facilities of a soaking pit are extremely important as these control furnace temperature and atmosphere. In modern pits the quantity of fuel, the desired fuel-air ratio and the draft or pressure in the pits are controlled automatically. The rate of fuel input is controlled by temperature measurement to maintain some predetermined level. The air is proportioned to the amount of fuel fired at a ratio that will give a slight excess or deficiency of air as desired. The furnace draft or pressure is controlled by automatically raising or lowering the stack damper to maintain the desired furnace draft. Pit-furnace burners are of various designs to provide the type of flame most suitable for heating in each particular pit design. In the regenerative pit the port acts as a burner. Some pit designs utilize a long-flame burner in which the air and gas are not mixed intimately in order to develop slow combustion and a longer path of heat release. In others, the mixing is very rapid and a short non-luminous flame is developed. Some burners are designed to inspire either the gas or air. Low-pressure inspiring burners usually employ gas at a low pressure to inspire air for combustion. With high-pressure inspiring burners, either the gas or air is carried at sufficient pressure to inspire the required volume of the other that is at low or atmospheric pressure. Modern pits are provided with accurate means for measuring and controlling the air volume required for the combustion of fuel. Generally, motor-operated fans are used to supply the air. Fans are used for delivering either hot or cold air to the burners. If recuperators or regenerators are not installed, the fans deliver cold air. Fans may be installed in some designs on either the cold or hot side of the recuperator. If on the cold side, the fan construction is simple and does not require special features such as

alloy blades and an insulated casing, as is the case with hot-air fans. Cold-air fans, however, subject the recuperator to a high pressure differential across its tubes and joints which sometimes causes excessive air losses and lowered recuperator efficiency.

Objectives in Modern Soaking Pit Design—The principal requirements of modern pit furnaces are:

1. Uniform heating of all ingots in the pit with no localized overheating of any ingot.
2. Heating rate equal to the ability of the steel to safely absorb heat.
3. Sufficient holding capacity to accommodate the required number of ingots without overcrowding.
4. Low operating cost.
5. Control of furnace atmosphere.
6. Ability to duplicate heating practices.
7. Minimum expenditure of ground space.
8. Minimum capital expenditure.

The first objective noted above is the most difficult to attain. Many different types of pits have been introduced to solve this problem. Uniform heating of all ingots in a pit with no localized overheating of any ingot permits holding the highest possible temperature in the pit compatible with the ability of the steel to absorb heat without sweating the surface or injuring the steel from severe temperature strain. Optimum heating rates are obtained by holding the surface just below the sweating point. This permits the most rapid flow of heat by conduction into the body of the steel. Since the flow of heat from the surface decreases as the temperature of the center of the ingot rises, the rate of firing must be reduced as heating proceeds to prevent overheating the surface. Variations in firing rate introduce problems in pit furnace heat distribution, as the hottest part of the flame, or the length of travel of heat release of the fuel, changes with fuel volume and air-fuel ratio. Experience with various pit types generally dictates the proper adjustments necessary for variation in firing rate. In some cases alterations in fuel-air ratio or furnace draft are the remedy chosen. In many instances, when heating with regenerative pits, and occasionally with other types, adjustment for lack of uniform temperature distribution is secured by firing at a uniform rate for a period of time followed by a period during which the pit is dampered and no fuel is admitted. During the latter period, the steel is "soaked" to equalize the temperature between ingots in the pit and between the surface and the interior of individual ingots. This practice, known as "firing and dampering," provides a product with a uniform temperature at high production rates even though uneven heating developed during the firing period.

In order to aid uniform heating, circulation of the gases in the pit furnace has been given considerable thought in modern design. Entrainment of cooler, spent gases with the hotter products of combustion, especially near the burner, dampens the tendency of the flame to develop a hot spot, lengthens the time for full release of heat from the fuel and aids in uniform heating. Other effective methods utilized for obtaining uniform temperature distributions are: (1) designing the burner to release the heat of combustion of the fuel over as great an area in the pit as possible to avoid concentration of heat, (2) developing the flame sufficiently distant from the ingot surface to control radiation within safe limits, (3) reducing flame emissivity by using non-luminous-flame type burners or the equivalent, and (4) use of higher pit pressure. A cause of uneven heating in pit furnaces is the fact that the top of an ingot is exposed to heat from all sides while the

bottom rests on a relatively cold unexposed surface. It is necessary to compensate for this either by proper distribution in the flow of gases in the pit, or, as is done in some designs, by maintaining higher effective flame radiation at the pit bottom. In some cases, as noted above, uniform heating is secured only by "soaking." Since the corners of ingots are heated from two adjacent sides, these parts reach the desired temperature first and, therefore, prevent the carrying of an otherwise permissible higher furnace temperature during a considerable part of the heating time.

The modern trend in pit design has been to construct larger holes than the two to six ingot capacity size formerly used. However, the over-all economy of using large pits for small ingots is not favorable for some mills and careful study of cycles is required in the selection of the proper size of pit.

Fuel consumption and the maintenance of modern pits have been reduced considerably by improved design, the installation of automatic controls and better materials of construction. Fuel-air ratio control for combustion and furnace atmosphere have reduced scale losses and the frequency of bottom making. Temperature measurements associated with automatic fuel-input control have made it possible to duplicate heating practices.

Modern Heating Practices—The time required to heat a charge of ingots in soaking pits generally is associated with transit time. The heat content of the ingot when charged into the pit is related to transit time, and it obviously requires less time to heat hotter steel. Transit time is the elapsed time beginning with the "start to pour" of ingots at the steel-producing source and ending with the charging of the first ingot of the heat into the soaking pit. The heating time required for ordinary carbon steels is approximately $1\frac{1}{2}$ times the transit time when the track time is not excessive. Heating cold ingots usually requires from 8 to 12 hours, but, if the ingot is unusually large or of some particular type of steel which requires special treatment, it may require up to 18 hours.

Modern instrumentation has made it possible to estimate the relative state of temperature difference between the surface and center of the ingot from the temperature-time and fuel-input readings. Formerly, the heater, by careful observation of the ingots through sight holes in the cover and his knowledge of temperature requirements for each particular type of steel, determined when the charge was ready for rolling. While this practice is still common with old regenerative-type pits, the heating in continuous-fired pits is governed largely by automatic controls which carry out the heating program desired.

In heating ordinary grades of hot carbon steel in modern pits with automatic control, the operator sets the control dial at the temperature level desired for drawing. A full, or maximum, head of fuel automatically is fired into the pits in the early stages of heating. The fuel input is cut back gradually as the heat soaks into the ingot until it is finally reduced to a minimum, just sufficient to cover pit radiation losses. When the fuel input

has remained at this low level for some specific time, such as a half hour or more, as experience dictates, the ingots are drawn. With high-carbon steel and many alloy grades which are cold or have had a long transit time, it is customary to cool the pit down to the temperature of the charged steel before placing these grades in the pits. They are then brought up rather slowly to some fixed temperature below the rolling temperature. After the ingots have been soaked at this lower temperature, most types of steel may be heated rapidly to rolling temperature. A common practice in heating certain alloy steels is known as **step heating**. Step heating consists in heating ingots to rolling temperature through several levels, each level being held for some specific time to permit equalization of temperature through the ingot. This procedure eliminates stresses due to severe temperature gradients in the ingots and provides slow, uniform heating. With step heating, it is customary to reduce the fuel-rate input considerably below the maximum setting permitted for heating ordinary types of steels. The best practice for heating each type and size of steel may be duplicated with modern control facilities. The fuel-air ratio control affects the flame characteristics and the furnace atmosphere. Flame characteristics have an important bearing on temperature distribution in the pit, and furnace atmosphere affects scale formation and character of the scale. By controlling the furnace atmosphere to advantage, the frequency of bottom making may be reduced, and minor surface defects on the ingot (from cooling, pouring or deoxidation practice) may be corrected or prevented from becoming more serious.

Operating Statistics—The principal operating data, aside from quality of heating and maintenance, relate to production output and fuel consumption. For comparative purposes, production is based generally on "live" pit area since there are so many different sizes and types of pits. The live pit-hole area is the area available in the pits on which ingots can be placed for heating. With proper loading this coverage of live pit area amounts to approximately 35 to 40 per cent (never more than 50 per cent) in modern pits. An average month's practice with modern pits is heating 30 to 100 tons of ingots per hour per 1000 square feet of live pit-hole area. This wide variation is due principally to the temperature at which the ingot is charged and to the type of steel. Other important factors causing the wide spread in pit furnace productivity are differences in the size and length of ingots, irregularity in mill operations and melting-shop pouring schedules, and the portion of live pit area occupied by the ingots.

The amount of fuel consumed per ton of steel heated varies from approximately 300,000 to 2,000,000 Btu. This variation also is due largely to variations in the temperature of the charged steel. Other factors contributing to fuel economy are careful design, proper use of insulation, reduction of stack-gas losses with recuperators and regenerators, proper installation of controls, utilization of the maximum percentage of hearth, controlled metallurgical practices, and regularity of melting shop and rolling-mill schedules.

SECTION 3

REHEATING FURNACES

Furnace Types—Reheating furnaces are divided into two general classes:

1. Batch type.
2. Continuous type.

Batch furnaces are those in which the charged material remains in a fixed position on the hearth until heated to rolling temperature. Continuous furnaces are those in which the charged material moves through the

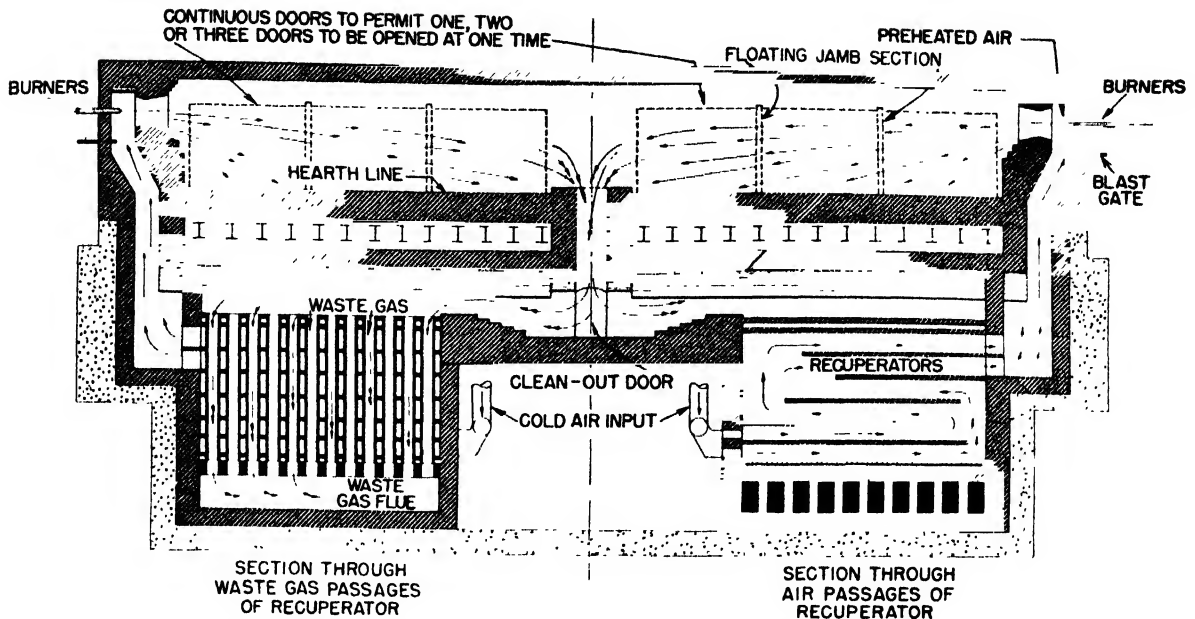


FIG. 21-9. Schematic longitudinal section through a recuperative batch-type reheating furnace. (Courtesy, Amsler-Morton Company.)

furnace and is heated to rolling temperature as it progresses through the furnace. Batch furnaces are the older type and are used for heating all grades and sizes of steel. However, small billets seldom have been heated in this type since the introduction of continuous furnaces. Batch furnaces are fired with either gaseous or liquid fuel, with preheated or cold air for combustion. The air may be preheated by regenerators and the furnace firing reversed from one end to the other, as in open-hearth furnaces. Batch furnaces, in which the air is preheated by recuperators, are not reversed and firing is continuous from one or both ends, depending upon the location of the gas outlet port (Figure 21-9). The steel to be heated in a batch furnace commonly is charged and drawn through front doors by a charging machine. Batch furnaces vary in size from those with hearths of only a few square feet, with a single access door, to those twenty feet in depth by fifty feet long, with five or six doors.

Continuous furnaces were designed initially for heating billets and small bloom sections. The hearths were relatively short in length and were sloped downward longitudinally towards the discharge end to permit an easy movement of billets through the furnace. In early designs, the furnaces were fired by burners located at the discharge end and the billets were heated by the hot gases flowing through the furnace above the top surface of the steel toward the charging end. Pushers were used to push forward the charge of cold billets. The flow of gas and steel in the furnace were counter-current. The modern continuous furnace has been altered in many respects from those of early design, although a large number of the older ones, particularly billet-heating furnaces, are still used. Longer furnaces generally are constructed now. Some have hearths 80 to 90 feet long, with top and bottom firing, and contain preheating, heating and soaking zones. The hearths usually are constructed level. A transition from the early designs to the modern three-zone slab heating furnace is illustrated in Figure 21-10.

The steel to be heated in a continuous furnace can be

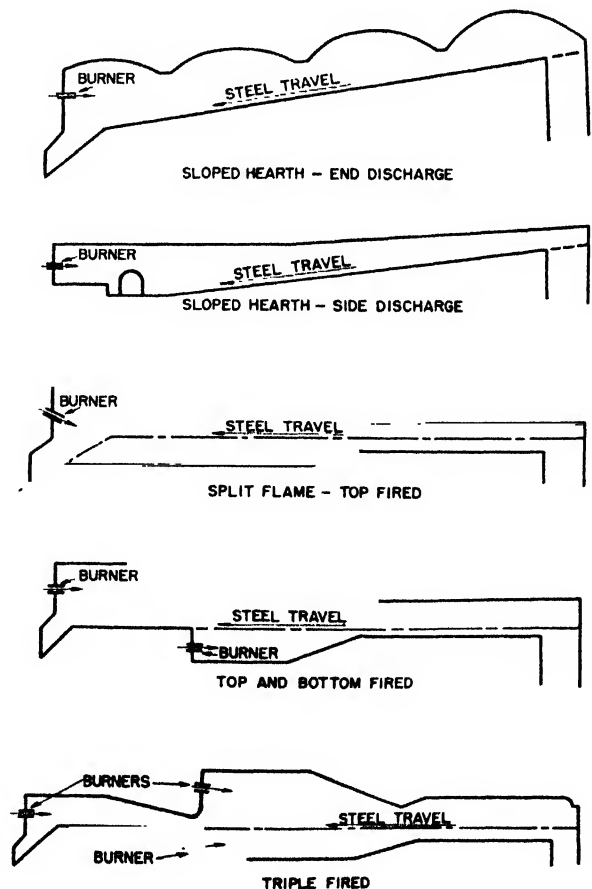
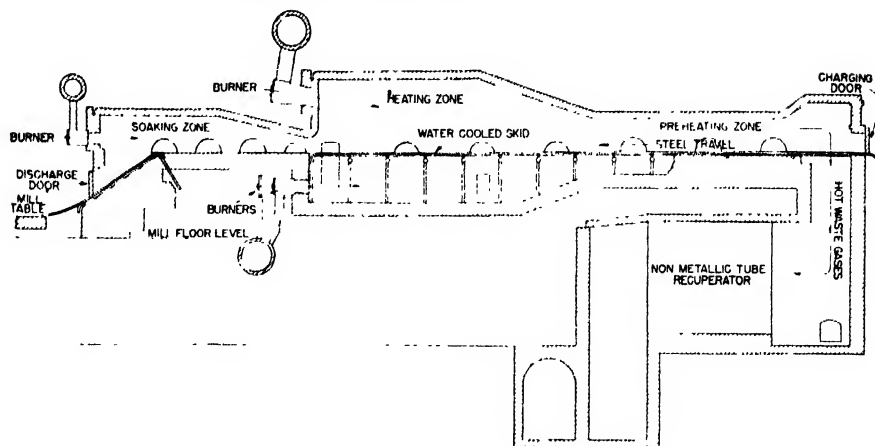


FIG. 21-10. Diagrammatic steps in the evolution of the modern three-zone slab-heating furnace from earlier designs.

FIG. 21-11. Schematic longitudinal section through a counter-current fired continuous reheating furnace. (Courtesy, Rust Furnace Company.)



charged either from the end or through a side door. In either case, the steel is moved through the furnace by pushing the last piece charged with a pusher at the charging end. As each cold piece is pushed into the furnace against the continuous line of material, a heated piece is removed. The heated piece is discharged either through an end door by gravity upon a roller table which feeds the mill, or it is pushed through a side door to the mill table by suitable manual or mechanical means. Figures 21-11 and 21-12 show sections through modern furnaces using counter-current and co-current flow of gases and steel, respectively.

A distinctly different type of continuous reheating furnace is the rotary-hearth type, a cross-section of which is shown schematically in Figure 21-13. It is used frequently for heating rounds in tube mills and for heating short lengths of blooms or billets for forging. The rotary-hearth type permits the external walls and roof to remain stationary while the hearth section of the furnace revolves.

General Considerations in Furnace-Type Selection—Some of the particular advantages of batch-type furnaces are:

1. They provide means for heating steels of various types and sizes, which can be heated more properly in separate batches than when mixed with other types in the same furnace, especially when specific heating practices are required.

2. They are suitable as a reservoir for holding hot steel directly from the primary mill for later rolling in the finishing mills.

3. They can be operated to heat steel to temperatures above 2400° F more satisfactorily than a continuous furnace. Steel can be given a "wash heat," when desirable, without trouble from the pieces sticking together. (Steel is sometimes "washed" or heated until an earlier oxide scale is melted and a new scale formed, to reduce surface defects.)

4. They provide excellent means for soaking steel.

Some of the important disadvantages of batch furnaces are:

1. High capital investment per unit of production.

2. Low hearth area efficiency. That is, the hearth in the conventional type is not utilized fully because of interference of door jams and clearance required inside the furnace for handling the stock with charging machines.

3. High man-hours per ton of heated steel.

4. Lack of flexibility for heating steel slowly in the lower temperature range. Furnaces must be cooled to charge high-carbon steel and some alloy steels.

5. Length of pieces to be heated is limited by tendency to bend when they are removed.

The advantages of the conventional continuous-type furnaces are:

1. High production per dollar of investment.

2. Low man-hours per ton of steel heated.

3. Greater ease in charging and drawing steel.

4. High hearth area efficiency.

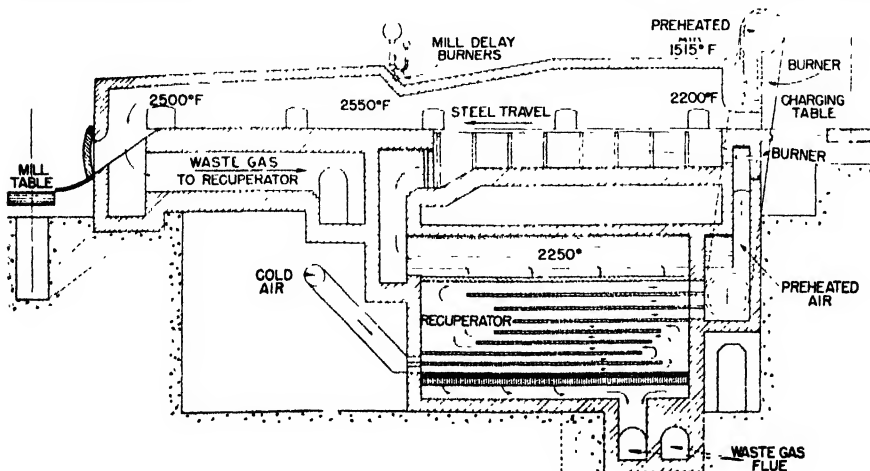


FIG. 21-12. Schematic longitudinal section through a co-current fired continuous reheating furnace. (Courtesy, Amsler Morton Company.)

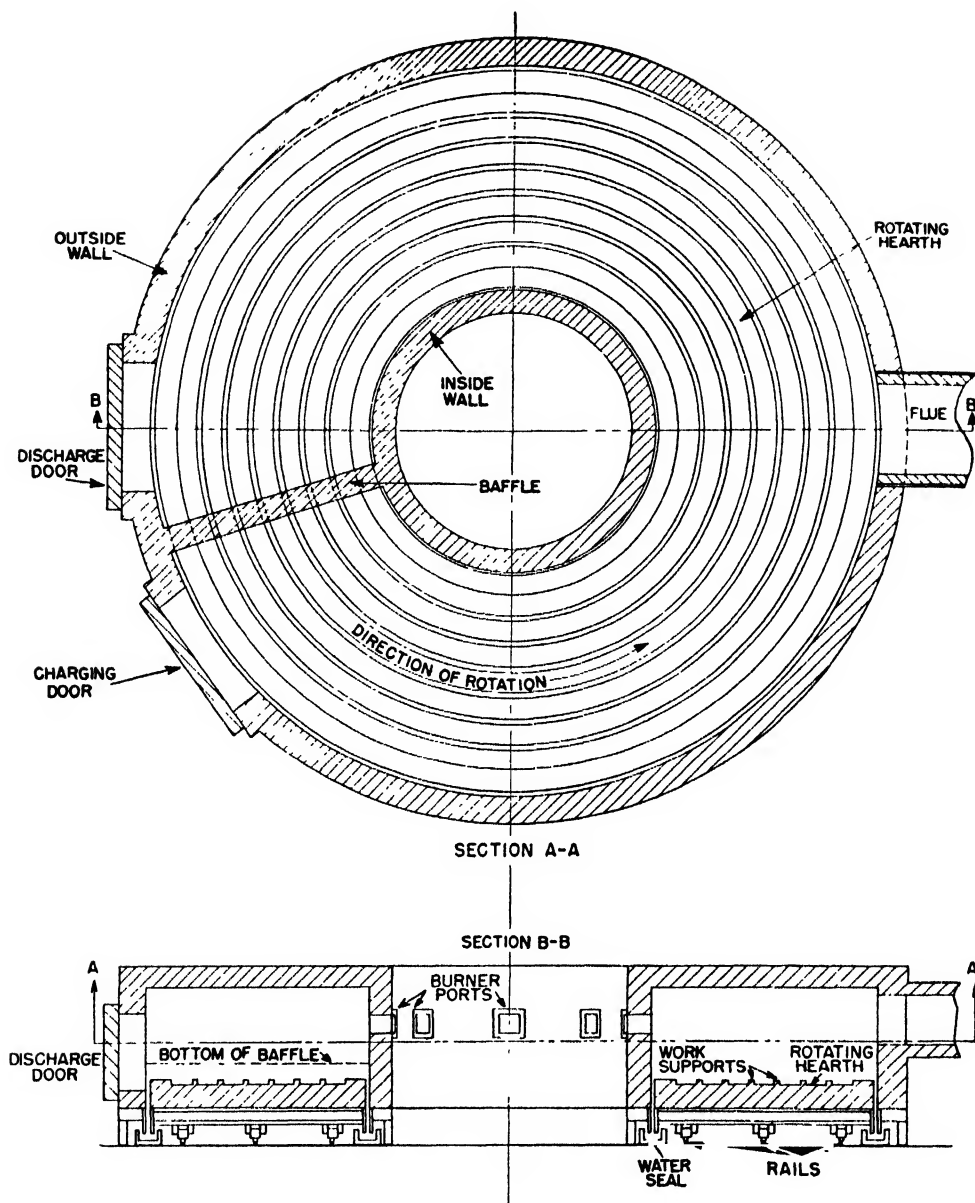


FIG. 21—13. Schematic arrangement of a relatively-small continuous rotary-hearth heating furnace. The plan view (above) shows a section above the hearth near the bottom of the baffle. Larger furnaces of this type have burners firing through both inside and outside walls above the hearth. (Adapted from a Salem Engineering Company drawing.)

5. Better means for controlling the rate of heating at all temperature levels. Gradual rise in temperature permits charging all grades of cold steel without cooling furnace.

6. Less trouble from temperature inequalities between each succeeding piece drawn.

7. High production per square foot of ground space occupied.

8. Can be built for any reasonable length of piece to be heated, resulting in higher mill yield, because of fewer crop ends from lengths beyond batch-furnace possibilities.

Some of the important disadvantages of conventional continuous furnaces are:

1. Lack of flexibility for heating efficiently small orders of different lots of steel or thicknesses. Heating time must be increased to suit the heating cycle of the piece, requiring lower heat transfer rate which may be detrimental to the heating of the other grades and sizes in the furnace.

2. Trouble from water-cooled skid maintenance, and from building up of scale on hearth, which results in piling of steel in furnace, due to climbing of pieces over each other, because of lack of square contact between adjacent pieces, when pressure is applied by the pusher.

3. Face of contacting surface of stock must be square to prevent piling.

4. Expensive to empty furnace at end of schedule.

5. Difficulty in pushing mixed sizes through furnace.

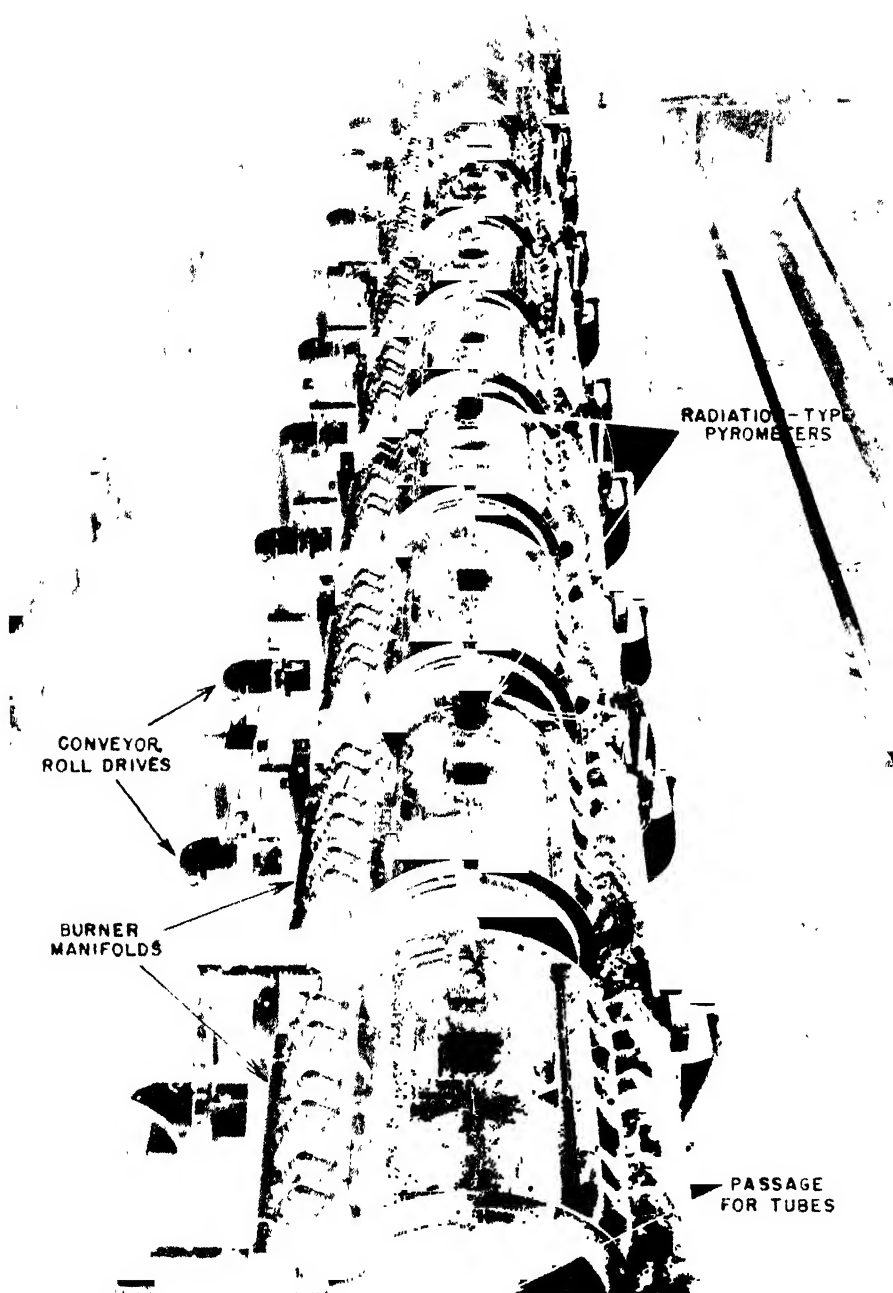


FIG. 21-14. Barrel-type furnaces in tandem for continuous reheating of pierced seamless tubes for sizing. Conveyor rolls between adjacent furnaces support the work in its travel. Temperatures are automatically controlled by radiation-type pyrometers.

Some of the important advantages of rotary-hearth furnaces are:

1. They eliminate either the manual labor required for rolling rounds forward on horizontal or moderately sloped hearths, or the disadvantages of excessively sloped hearths in continuous furnaces.

2. They have better means for controlling the rate of heating at all temperature levels than batch-type furnaces.

Some of the important disadvantages of the rotary-hearth furnace are:

1. High capital cost per unit of production.
2. Low hearth area efficiency.

3. Large ground space required per unit of production.

Some features in both batch and conventional continuous furnaces are worthy of note:

1. Regenerators or recuperators act as a reservoir of heat supply, which is especially valuable for efficient soaking of steel.

2. Continuous furnaces provided only with top firing require longer hearths for equal production than those with top and bottom firing, but do not require a special soaking zone to eliminate cold spots on the work caused by contact with water-cooled skids. Heating of sizes greater than 4 inches thick is not economical in furnaces that are top-fired only.

3. Continuous furnaces with single-zone firing have higher scale losses and greater tendency to cause decarburization of high-carbon steel than the top-and-bottom fired furnaces, since the steel is in the furnace longer. Decarburization is caused primarily by hydrogen and water vapor combinations in furnace gases, and increases almost directly with the time the steel is at elevated temperatures. Free air and carbon dioxide in the furnace atmosphere cause decarburization to a lesser degree. The scaling of steel is practiced sometimes deliberately to remove the decarburized surface layer.

4. A level hearth in a continuous furnace eliminates the stack effect of hearths sloping upwards towards the charging end. This stack effect draws cold air into the furnaces at the hot end causing higher fuel consumption and scale losses.

5. Batch-type furnaces used for heating in a production line require supplementary furnaces for preheating certain grades of alloy and high-carbon steels for transfer into the hotter furnaces. The preheating zone of a continuous furnace makes this unnecessary.

6. Back-fired or continuous furnaces with co-current flow of steel and gases subject the steel in the charging end to rapid surface heating which is not suitable for heating steels that have low thermal diffusivity, such as high-carbon and heat-resisting alloys. (Thermal diffusivity is equal to the thermal conductivity divided by the product of mass specific heat times density.) This type of furnace provides well-soaked heating for mild and medium-carbon grades of steel.

7. Side-discharge continuous furnaces have less air infiltration at the hot end than end-door discharge furnaces. End-door discharge of the usual gravity type induces cold air into the furnace by the stack effect at the discharge section of the furnace. End-door discharge, however, is mechanically simpler for removing the heated stock, particularly slabs and heavy blooms.

Furnaces have been developed to raise the temperature of steel for hot shaping by exposing the piece to be heated to intense radiant energy from gas burners of special design, set very close to the path of the steel. This method of heating has been very effective in obtaining fast heating for small round sections. The barrel-type furnaces in tandem shown in Figure 21—14 are used for reheating pierced tubes for sizing in seamless-tube mills. This method has recently been applied to soaking pits.

The trend in reheating furnace design is towards the continuous type, with increased attention to refinements to satisfy conditions pertinent to each mill or forge shop, or the types of steel being heated.

Operating Statistics—The production rate of furnaces varies widely. Batch furnaces, as a general class, produce much less tonnage per furnace operating hour than the continuous type. While batch furnaces often are used for heating hot charges, as in structural and rail mills, continuous furnaces usually are used only for heating cold steel. Batch furnaces heating cold charges for rolling mills are designed to heat from 5 or 6 tons up to 25 tons per furnace hour. When heating a hot charge, the production is much higher, and the rate depends largely upon the temperature of the charged material and the size of the furnace.

The production of a continuous furnace depends principally upon the hearth size of the furnace and whether the material is heated from more than one side. Many

continuous furnaces heat over 50 tons of steel per furnace hour. A modern three-zone slab-heating continuous furnace, 20 feet wide, with an effective hearth length of 80 feet, top and bottom fired, is capable of heating 110 tons of slabs per hour to a rolling temperature of 2250° F. The productive rate of a furnace for a fixed thickness of steel is directly proportional to the surface exposed to the flame, other conditions remaining the same. For comparison of furnace performance, the relative number of pounds heated per effective square foot of hearth area per hour is used as a common factor. By this comparison, furnaces heat from 30 to 150 pounds per square foot of hearth area per furnace hour. The variation in heating capacity is due principally to the difference in the total surface of the steel exposed to heat in the furnace, which the hearth area does not measure. Other factors are the temperature of the heated product and the grade of steel. In many batch furnaces, the material receives heat from three sides and in continuous furnaces from two sides. Heating from more than one side decreases heating time and, therefore, increases production rates. The production and fuel rates of furnaces are affected by continuity of operation. A furnace will not provide high production rates and fuel economy if it is operated for only short intermittent periods. It also must be fully loaded to provide optimum production and fuel economy.

The fuel consumption of reheating furnaces varies from 300,000 to 3,000,000 Btu per ton. When hot steel is heated, very little additional heat may be required to attain rolling temperature. If the steel charged is at a temperature of 1800° F, only about 140,000 Btu per ton must be added in the furnace to heat to 2250°, while with cold steel about 700,000 Btu must be added in the furnace. If the hot steel requires 500,000 Btu per ton input to the furnace and only 140,000 Btu are absorbed,

the furnace fuel efficiency is $\frac{140,000}{500,000} \times 100$, or 28 per cent.

If cold steel is heated with a fuel consumption of 2,000,000 Btu per ton, as obtained in many modern furnaces,

the furnace fuel efficiency is about $\frac{700,000}{2,000,000} \times 100 = 35$

per cent. The best fuel efficiency is obtained in continuous furnaces that are relatively long, have hearths fully covered with steel, and are operated at a high rate of continuity. In both batch and continuous furnaces, the length of gas travel and the velocity of the gases through the furnace have an important effect on fuel economy. An excessive velocity of furnace gases tends to increase the exit temperature of the gases leaving the furnace and, therefore, low fuel efficiency results. Fuel efficiency is improved by providing features in design that reduce the inherent furnace losses. The greatest losses usually are represented by the heat carried away in the stack gases. Reduction of this loss can be obtained by the installation of recuperators, regenerators or waste-heat boilers, and by designing the furnace cross-sectional area for proper gas velocities. Automatic fuel-air ratio and furnace-pressure control reduce stack losses by reducing the volume of gases from excess air for combustion, or from air infiltration. Radiation losses are reduced by insulation of the furnace walls, roof and hearth. Losses from water-cooled skids in continuous furnaces can be reduced by eliminating any unnecessary exposure of them to hot furnace gases, and to some extent by use of refractory pipe covering.

SECTION 4

HEAT-TREATING FURNACES

General Design Requirements—Heat-treating furnaces are grouped into either batch or continuous furnaces. There are many different types in each group. The simplest furnaces are the direct-fired batch type with manual controls. The more elaborate installations used for large production lines are continuous furnaces with automatic program control. In a number of installations, special facilities for controlling the atmosphere in the working chamber are provided to obtain the desired surface condition. The most common heat treatments performed in furnaces are annealing, normalizing, spheroidizing, hardening, tempering, carburizing and stress relieving. Heat-treating furnaces seldom are designed for temperatures in excess of 2000° F, and generally are operated in the 800° F to 1600° F range. They are usually well insulated and built tight to prevent air infiltration or a loss of special atmosphere gas. Attention in design is directed toward procuring uniform temperature distribution in the working chamber of the furnace. The position and method of heat application, and the circulation of gases in the furnace, are of major consequence in this. In annealing furnaces, means for controlling both the rate of heating and cooling of the stock usually are provided. Since the required heating and cooling rates of different types of steel vary, it is necessary to provide flexible means for controlling these functions. Insulating firebrick is used generally in heat-treating furnace construction, due to its low heat-storage capacity, to permit heating and cooling the furnace quickly. For intermittent furnace operation this is particularly vital. Attention is directed in design to spacing the charge in order to attain the most efficient flow of heat around the stock. In coil-annealing furnaces, and in furnaces for heat treating other material, special facilities often are provided to improve the circulation of special atmosphere gases during heating and cooling. In all heat-treating layouts, special consideration is given to providing sufficient furnace capacity to maintain the desired time-temperature relation of the treatment. Furnaces forced beyond their normal capacity usually yield an erratic and non-uniform product.

For handling batch loads of material to be heated, quenched, and tempered, quench tanks and cranes should be located in such a way that little time is lost in getting the material from the furnace into the quenching medium. Furnaces also must be arranged so that, after quenching, a second furnace is available for taking the charge promptly for further treatment. Usually three furnaces are provided for operations of this type: two for heating and one for tempering.

In large-scale heat-treating operations, the layout of facilities is very important. The furnaces must be arranged to suit an orderly flow of material through the shop. Adequate space for temporary storage and handling is necessary for both the raw and the finished material. A sufficient number of cranes or other stock-handling facilities of the proper capacity must be provided to eliminate bottlenecks or interference with prescribed furnace cycles. A central station for the preparation of atmosphere gas generally is provided in the larger furnace layouts where particular attention to steel surface is required.

In heat-treating furnaces, many furnace parts, such as conveyors or rollers in continuous furnaces, radiant tubes for indirect firing and covers in coil and pack-

annealing furnaces, are of metallic construction since the temperature seldom exceeds the 800° to 1600° F range. Special alloy materials are utilized to reduce the maintenance of these parts to a minimum. In selecting the furnaces for a heat-treating plant, careful consideration must be given to the type of product to be heated, to the kind of treatment to be performed, and to the production rate required. The ensuing sections explain pertinent factors in design and describe the application of various furnace types.

Method of Heat Application—The character of the material to be heated and the type of treatment to be performed have an important bearing on the choice of method of heat application. Heat-transfer laws govern the flow of heat to the steel in heat-treating as in other heating furnaces. The surface of the material absorbs heat transmitted to it by radiation or by convection or both, and this heat is transferred through the body of the material by conduction. In heat-treating furnaces, the transfer of heat by convection is relatively more significant than in furnaces operated at a higher temperature level. In heat treating steel, the rate of heat transfer to the surface is usually low in order that each individual piece, as well as all pieces in the furnace, may be brought up uniformly to the required temperature level. However, in some furnaces which utilize induction heating or radiant-type open-flame burners, uniform

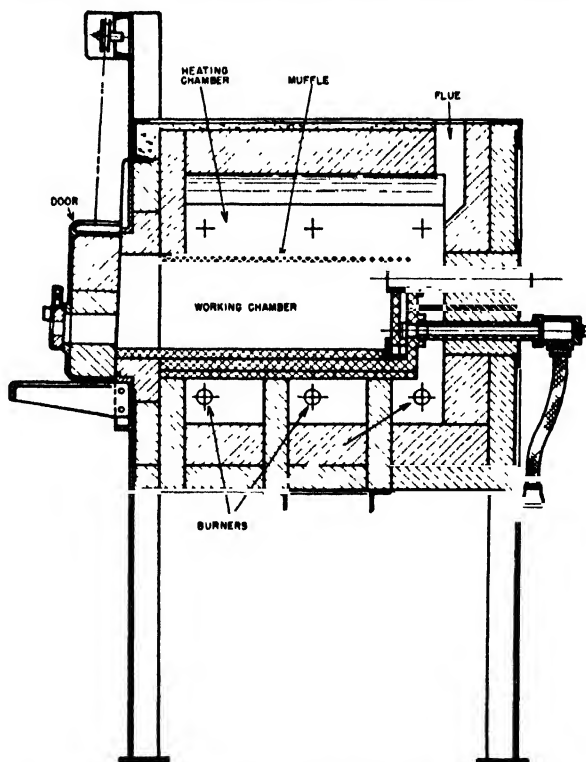
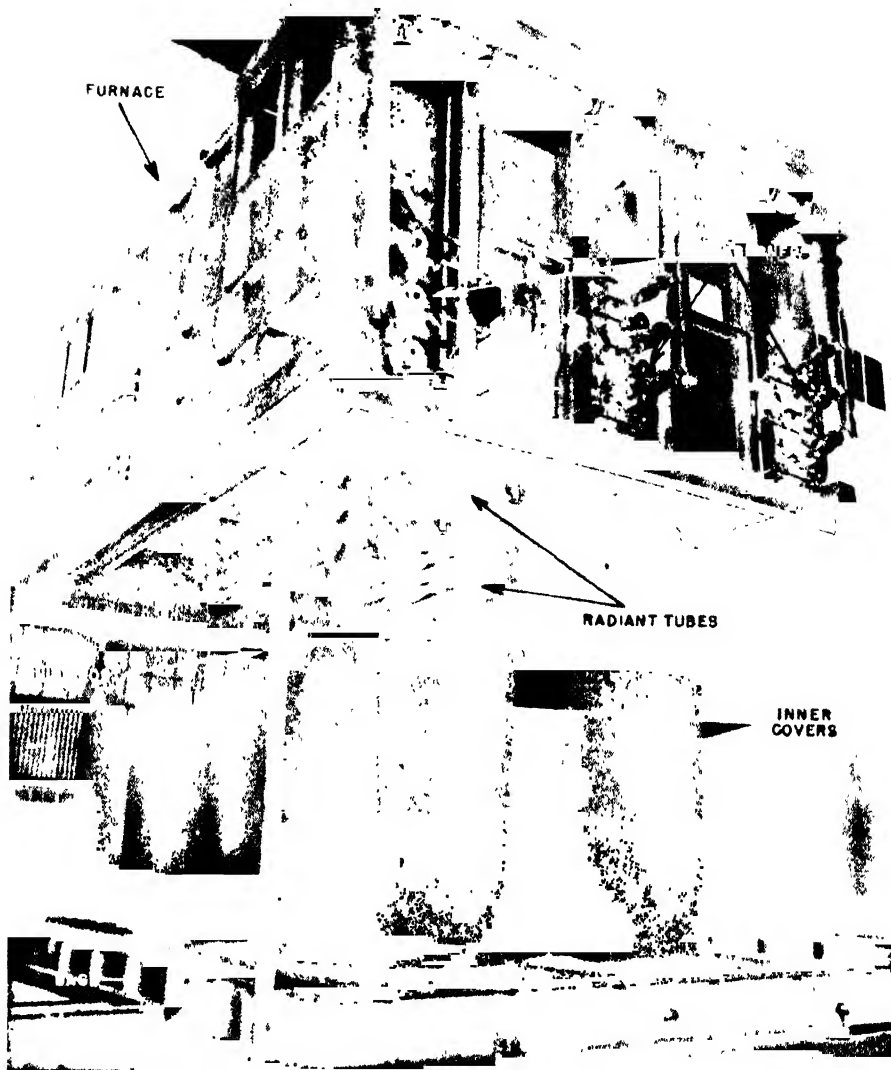


FIG. 21—15. Diagrammatic cross-section of an indirect-fired muffle type furnace equipped for use of a controlled atmosphere in the muffle. Openings in rear provide for insertion of control thermocouple and entrance of prepared atmosphere gas. (Courtesy, Surface Combustion Corporation.)

FIG. 21—16. Radiant-tube fired cover-type furnace employed in heat treating coils of flat rolled products enclosed in inner covers under which controlled - atmosphere gas is circulated. (Courtesy, Surface Combustion Corporation.)



heating can be accomplished rapidly with high heat-transfer rates.

Gaseous fuel and electric power are the two main sources of heat used in heat-treating furnaces. In some cases, fuel oil has been substituted for gas due to a shortage of the latter or for economic reasons. In gas heating, a number of variations in method of firing are used. These variations may be separated into two general classes, direct and indirect firing. Direct firing is used more generally. This method permits the products of combustion of the fuel to circulate about the material to be heated. In direct-fired furnaces, open burners may be used either in the furnace proper or they may be installed outside the work-heating chamber in the path of an external fan which circulates large volumes of hot gases through the furnace. Temperatures attainable in this type of furnace are limited by the materials of which the fan is constructed. The latter modification is used in convection-type furnaces. In indirect-fired furnaces the products of combustion do not enter the work-heating chamber. Indirect firing is used in muffle furnaces, an example of which is shown in Figure 21—15.

Another common application of indirect firing is ob-

tained with radiant tubes, an example of which is shown in the furnace in Figure 21—16. Furnaces using direct firing are relatively lower in operating cost and original capital investment. Furnaces using indirect firing generally are selected where the control of furnace atmosphere is of particular importance.

Electric power is used as a source of heat in many heat-treating furnaces due to the ease it affords for controlling temperature, to its suitability for use with protective-gas atmospheres, and to its cleanliness. Resistance-type heating elements, which either are imbedded in the furnace refractory lining or suspended from heat-resisting hangers, radiate heat to the furnace charge. The resistance units are positioned in the furnace to permit uniform heating. Furnaces with zone heating sometimes use electrical resistance units because of their easy adaptation to the control of temperature levels. Another method for heating electrically is by induction. Induction heating is done by passing a high-frequency alternating current through a coil surrounding the material to be heated. The coils are shaped to suit the material to be heated. The rapidly alternating electrical field, in which the material to be heated is held, causes the steel to heat very rapidly, due to eddy

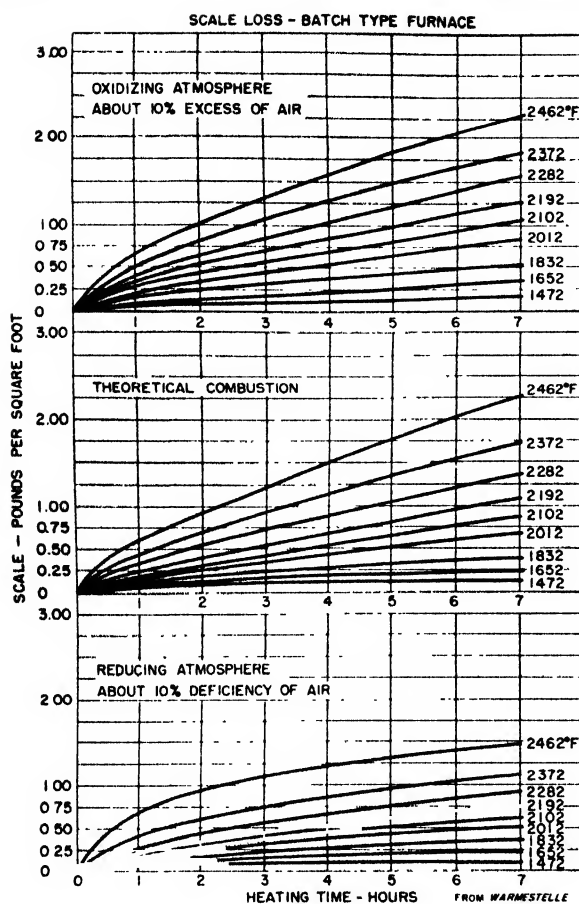


FIG. 21—17. Graphical representation of relative amounts of scale formed on steel heated in a batch-type reheating furnace with variations in time of heating, temperature level, and heating-chamber atmosphere. (From "Warmestelle.")

currents and hysteresis. Due to the rapidity of the process, the duration of heating is very critical, necessitating precise control.

Atmosphere Control.—The effect of a heat-treating operation on the surface condition of the work pieces is influenced by the time of heating, the temperature level maintained, and the atmosphere surrounding the material. Figure 21—17 shows graphically the relative amount of scale formed in a batch reheating furnace for variations of these conditions. While the temperature level in heat-treating furnaces is somewhat lower than that shown in the figure, the time is generally longer. By using the proper atmosphere in the working chamber, a clean scale-free surface is obtained. Such a surface is required for most sheet and strip material and other important steel products, such as wire and tubes. The three gases most injurious to surface condition are oxygen, carbon dioxide, and water vapor. The effect of each of these, as well as effects from other gases, follows:

Oxygen reacts with the iron of the steel to produce iron oxide. For this reason it must be excluded entirely for bright annealing. It also reacts with the carbon in steel to lower the carbon content of its surface; that is, it decarburizes the steel. In some types of controlled annealing, oxygen of the air may be caused to scale the steel faster than it decarburizes it. This results in a

product having a decarburized surface layer of minimum thickness and the scale, being flaky, is easily removed.

Nitrogen, in the molecular state, is entirely passive to iron and is entirely satisfactory for bright annealing low-carbon steels. If pure and very dry, it will be passive to high-carbon steel, but the presence of even slight traces of moisture will cause decarburization.

Carbon dioxide and carbon monoxide are considered together since the ratio of their concentration in the atmosphere plays an important part in their action on the steel surface. As an example, if the ratio of carbon dioxide to carbon monoxide is 0.6 or higher at 1500° F, the atmosphere will scale steel readily. If the ratio is reduced to 0.4, or lower, the atmosphere will no longer scale steel but will remain decarburizing to a 1 per cent carbon steel. For low-carbon steels, a ratio of carbon monoxide to carbon dioxide on the order of about two to one will be in equilibrium with the steel, and a gas of this composition is used to advantage for producing bright-annealed sheets. A higher carbon-monoxide content actually will carburize the steel.

Hydrogen is highly reducing to iron oxide, and, therefore, opposes the formation of a heavy, flaky scale and when present in the products of combustion, promotes formation of a tight scale that is hard to remove. At certain temperatures it is absorbed by the steel and is likely to result in embrittlement, more particularly in high-carbon steel. If the hydrogen is dry, it has no scaling effect on high-carbon steel at elevated temperatures, but it does cause considerable decarburization. A common prepared atmosphere which is used for bright-annealing work is composed of 75 per cent hydrogen and 25 per cent nitrogen. It is formed by cracking anhydrous ammonia. Hydrogen is highly explosive if mixed with air and particular precaution must be exercised to prevent air infiltration into furnaces in which it is employed.

Water vapor is oxidizing to iron and combines with carbon of steel to form carbon monoxide and hydrogen within the temperature range of the "water-gas reaction." It is reactive to a steel surface at temperatures even as low as 400 to 700° F, and is thus often the cause of formation of a blue oxide during the cooling cycle.

Hydrocarbons, more specifically methane, are carburizing gases. They are subject to thermal decomposition at annealing temperature, liberating hydrogen and depositing soot on the steel.

The most commonly prepared gases for control of atmosphere are formed by the partial combustion of hydrocarbon gases, contained in such fuels as coke-oven gas, natural gas, propane, or butane. Manufacturers of converters for the preparation of gases describe the various kinds under trade names, such as "DX" gas, "Drycolene," etc. The first step in making such a gas is to burn a mixture of the fuel gas with air. This provides a gas high in nitrogen, but containing other undesirable gases including water vapor, which must be removed. Other atmosphere gases are prepared by (1) cracking a non-combustible mixture of air and gas with a catalyst at high temperature, (2) by cracking anhydrous ammonia at high temperature, and (3) by passing air through a heated retort filled with charcoal.

Furnaces using controlled atmosphere have a number of construction features not incorporated in ordinary furnaces. These features are essential to prevent loss of gas and to minimize the entrance of air into the furnace which would upset the control established. Casings for furnaces with controlled atmosphere are welded gas-tight. Batch-type furnaces are provided with sand seals. Continuous furnaces charged and discharged from the

ends sometimes have flame curtains to burn out the oxygen from any possible air infiltration, or the furnace may be operated under sufficient pressure to prevent air infiltration. In the latter case, a small loss of the prepared atmosphere usually occurs through the small unavoidable openings in the furnace. Doors, where required, are fitted snugly by sloping fronts with ground surfaces or by wedging devices or clamps.

Batch-Type Furnaces—The five principal general types of batch furnaces are described below:

1. **Box furnaces** are constructed with a solid hearth. They are shaped, as their name implies, similar to a box and are charged through door openings by tongs or some mechanical charger. The furnace hearth may vary from a few square feet in area to over 30 square feet. Heating may be done by direct or indirect fuel firing or by electricity. Muffle and semimuffle type construction often is employed when control of atmosphere is required. This type of furnace is used frequently for individual-piece or small-lot heat treating, for laboratory test and shop work, and for general production work on a small scale. Box furnaces have been constructed for convection heating either with a fan underneath the roof or with one external to the furnace for recirculation. Furnaces of this type are used for annealing, normalizing, tempering and carburizing.

2. **The car-bottom furnace** consists of a furnace shell equipped with burners or heating units with the hearth built upon a separate car which runs in and out of the furnace shell to charge and unload the furnace. The car usually is moved into and out of the furnace by a toothed rack attached to the bottom of the car and a stationary pinion actuated by an electric motor, the car itself resting on rollers or wheels that move over a two-rail track. The doors of the furnace are of the vertically

lifting type, full width of the furnace, and are hydraulically or electrically operated. In order that the entire surface of the charge may be exposed to heat of the same intensity and to aid circulation, the charge is supported above the floor of the car bottom by heat-resisting alloy castings or on refractory piers. The car bottom is made to fit the furnace closely and the escape of hot gases around it is prevented by sand seals. Car-bottom furnaces have been constructed to process charges from a few tons to several hundred tons. They are used for heat treating of axles, bars, heavy plates, castings and miscellaneous shapes.

For operations involving heating, quenching and tempering, it is desirable that the quenching tank be located in proximity to the furnace to enable the charge to be placed in the tank in the shortest possible time. In some installations, less than a minute is required to transfer the charge from a closed furnace to the quenching tank. Car-bottom furnaces may be direct or indirect fired, and various designs have been developed to improve heat distribution in the working chamber. Electric heating also is employed in some car-bottom furnaces. Car-bottom furnaces sometimes are constructed of two chambers side by side, with a common division wall to facilitate annealing and tempering operations. In some installations, an auxiliary cooling system employing blowers is provided to accelerate cooling. Some car-bottom furnaces are known as **elevator furnaces** where the car is rolled under the furnace shell and then raised into the furnace by a motor-driven lifting mechanism. Those in which the shell is lowered over the car, as shown in Figure 21—18, are used to provide a more complete sand or water seal than is obtainable with the conventional car-bottom furnaces.

3. **The bell-type furnace** has a removable shell or

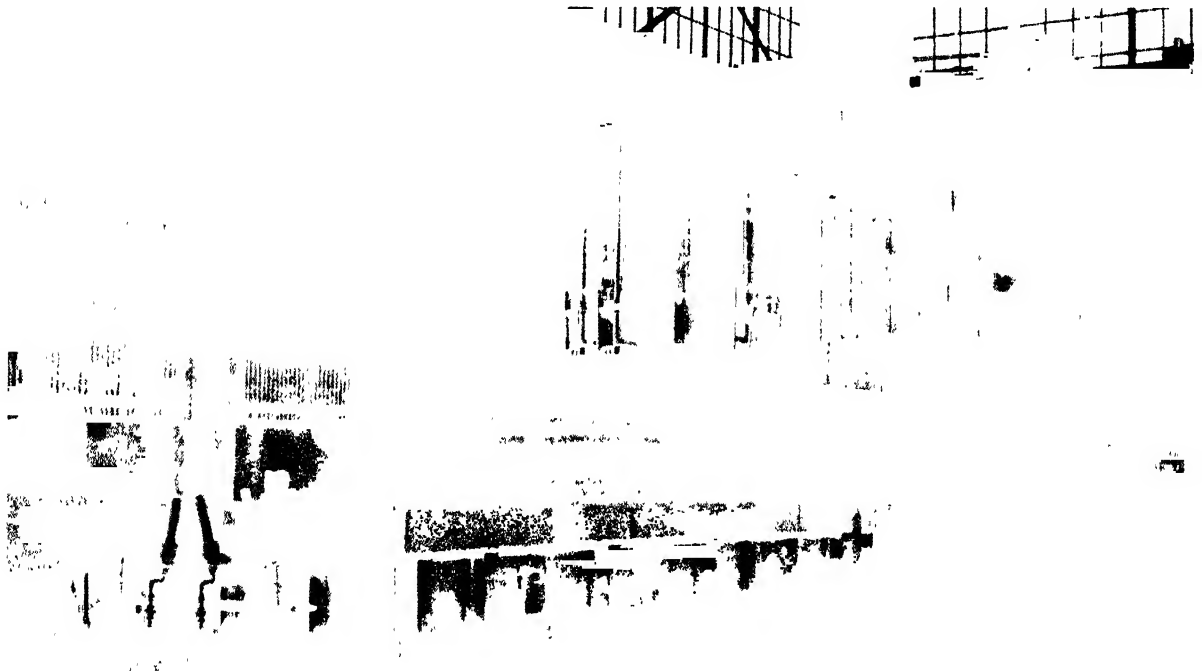


FIG. 21—18. Charge of mixed sizes of steel bars, supported above hearth by cast-alloy fixtures, ready to be rolled under bell-type furnace body, which then will be lowered over charge. Toothed rack above floor at lower left is driven by a pinion to move car. (Courtesy, Surface Combustion Corporation.)

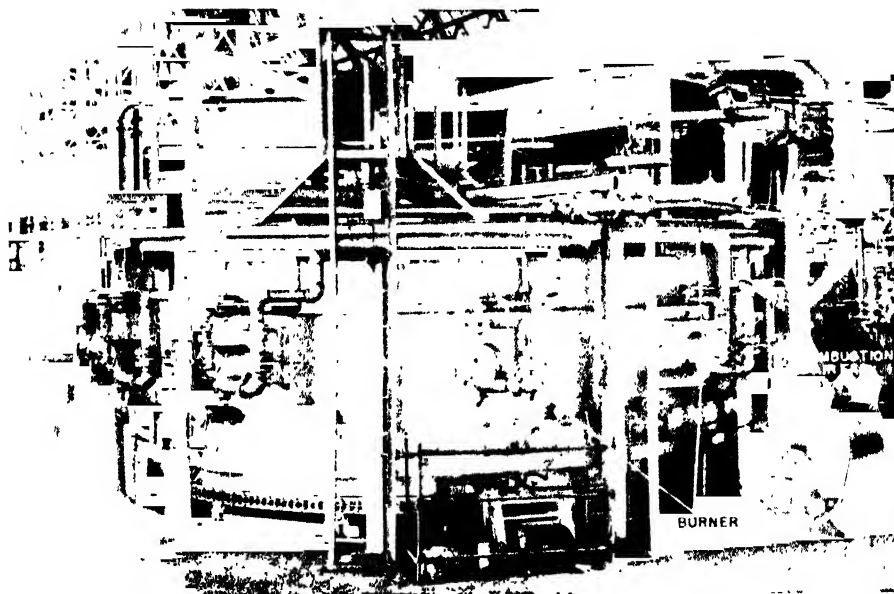


FIG. 21—19. Rotary-hearth furnace for heat-treating steel. Hearth is rotated by a chain-and-sprocket drive, seen in the foreground. (Courtesy, Surface Combustion Corporation.)

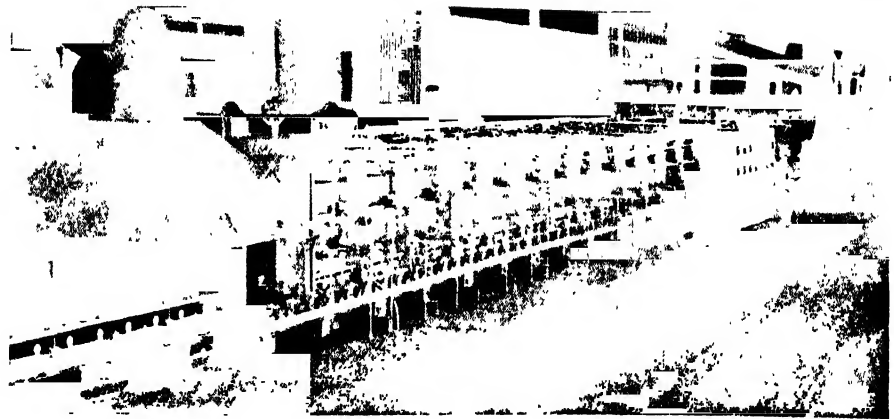
cover. The furnace usually is used for processing material which requires special surface protection from oxidation or decarburization. The furnace shell is removed by a crane and set aside while the hearth of the furnace is charged. The shell is then replaced, as was shown in Figure 21—16. Furnaces of this type, used for annealing sheet, strip, rod and wire, usually are called **box annealing, pack annealing, coil annealing, or cover annealing furnaces**. In these, the material is stacked on a permanent base or stand, a light inner cover is placed over the stack, sealed with sand at the bottom and provided with a constant supply of prepared gas atmosphere, and then the portable heating unit is lowered over the assembly. The heating covers are square, rectangular or cylindrically shaped. Loads vary from 35 to 400 tons per charge, distributed on one to eight stands per base. In most instances, a number of bases and inner covers are provided with one or more covers for heating. After heating of each charge is completed on a base, the heating cover is moved to another base, leaving the charge protected by the atmosphere under the inner cover which is left in place. The covers may be direct fired or equipped with radiant tubes for indirect firing, or they may be heated by electrical resistance units. The heating elements are attached to the inside of the heating cover, which is built of steel and lined with a refractory insulating material and braced substantially in order that it can be moved from base to base with an overhead crane. Many inner covers are made of heat-resisting alloys. All are sealed to the base at their bottom edges with a powdered refractory. The heating time in a cover annealing furnace for coils of sheet or tin plate range from 24 to 44 hours for the larger sized furnaces, depending upon the length of soaking period required. The soaking period usually is about 4 to 12 hours. In furnaces of 150 to 300 tons capacity, the average production is about 5.5 tons per hour, the fuel consumption about 1,000,000 Btu per ton, the maximum fuel-burning capacity about 12,000,000 Btu per hour, and the atmosphere-gas consumption about 1,200 cu. ft. per hour. In pack or box annealing furnaces, natural gas or some inert gas is used to surround the charge; the circulation of gases inside the inner cover is by natural or forced convection. In cover fur-

naces for annealing coils, separators are placed between each coil to aid in distribution of the gas inside the inner cover. The circulation of this gas in a number of modern installations is forced. The fan is located in the base below each stand. The trend in annealing sheet and tin plate has been towards the greater use of coils and heating by forced convection rather than by the former pack method of annealing, in order to obtain higher production and improved uniformity of heating.

4. **Pit furnaces** are furnaces of cylindrical or rectangular shape in which the material is charged and withdrawn through an opening in the furnace top. The larger furnaces are installed usually with at least part of their work chambers below floor level, while many of the smaller and shallower furnaces rest on the working floor, for convenience in handling material. The material to be processed usually is suspended by a fixture or loaded into a basket and set into the working chamber, which resembles a "pit." Pit furnaces employ either direct firing or electrical heating, in either case with natural or forced circulation. They may or may not be equipped with special facilities for atmosphere control. Pit furnaces are used for normalizing, hardening, annealing, tempering, and carburizing.

5. A **salt-bath or lead-bath furnace** is another type of heat-treating furnace. It is designed to hold a bath of molten salt or lead in which the material is immersed for treatment. These furnaces are usually small pot-like affairs used in batch operations, but some large furnaces have been constructed of rectangular shape with depths of 15 feet to suit the shape and size of the material to be handled, with conveyors or other means for carrying out continuous operations. They are equipped usually with a hinged cover or a ventilating hood for minimizing fumes. Such furnaces are used to obtain uniform temperature distribution and close temperature control of the work piece. The bath is heated and maintained at proper temperature either by electrical resistance or by combustion of a fuel. Furnaces with a molten bath for heat treating are called **pot furnaces** when the bath is contained in a pot or crucible constructed of a heat and corrosion-resistant metal, usually externally heated by suitable burners. Other bath-type furnaces may be heated by electric current passing through the (salt)

FIG. 21—20. Continuous disc-roller-hearth type of normalizing furnace, divided lengthwise into zones each having individual automatic temperature control.



bath between immersed electrodes, or by immersed resistance coils or fuel-fired tubes.

Continuous Furnaces—In continuous furnaces, the material moves through the furnace, and two basic types of these are recognized. In one, the furnace is circular with a rotating hearth which carries the charge. The walls and roof are stationary, and the furnace enclosure is made by contacting the walls with the periphery of the moving hearth through a sand or liquid seal. In the other type, the furnace is composed of a single, long straight chamber, or series of chambers, through which the material is moved. Differentiation of continuous furnace types may be made according to the way the material is moved, such as the rotary-hearth, the roller-hearth, the pusher, the conveyor, the walking-beam, the tunnel, the continuous-strand and the monorail types. Modern production methods, dealing with ever increasing tonnages of material of identical size and treatment, favor the continuous-furnace type best suited to the nature of material to be heat treated.

Continuous furnaces are designed with and without auxiliary equipment for atmosphere control. Heat may be applied by direct or indirect firing or electrically. They are especially suited for zone heating and cooling. A brief description of continuous furnace types and their application is given below.

1. **Rotary-hearth furnaces** are used generally for heating pieces that are to be handled individually. Typical applications are the heating of gears, shells, cylinders, billets, etc., that are to be fixture-quenched or handled individually for scale-free hardening without decarburization, for normalizing or drawing. This furnace type is used also for heating smaller parts loaded in lightweight trays, and for pack carburizing. Charging and discharging are accomplished at the same location. Rotary-hearth furnaces are built in a wide range of hearth sizes, to heat from a few hundred pounds up to 60 tons per hour. A typical rotary-hearth furnace for heat treating steel is shown in Figure 21—19.

2. **Roller-hearth furnaces** are high production, continuous-type units, especially suited for uniform treatment of large orders of identical material. This type of furnace is used widely for bright annealing of tubes, stampings, drawn parts, etc.; for normalizing, annealing, hardening and drawing steel bars; for annealing malleable iron, small steel and iron castings, and forgings; and for normalizing flat-rolled products. Roller-hearth furnaces are constructed as a single furnace or as a line of furnaces for zone heating and cooling, and sometimes have an intermediate section with a tank for quenching.

In some modern furnaces used for continuously treat-

ing short lengths of sheet, disc rollers made of heat-resisting alloys with polished surfaces are utilized to reduce the cooling effect of full contact with the ordinary type of roller and to avoid scratching of the piece. The discs are staggered and mounted upon water-cooled shafts, which are driven by variable-speed motors either through a chain and sprocket system or shafts and gears. A gas-fired normalizing furnace with automatic pyrometric control is shown in Figure 21—20. Furnaces of this type are built up to 100 inches in width and vary from 120 to 200 feet in length, the larger ones having capacities for normalizing up to 300 tons per 24-hour day. Sheets undergoing treatment in roller-hearth furnaces may be protected further from contact with rollers of whatever type by the use of rider sheets, which are placed on the rollers and support the work. Cover sheets on top of the work also may be used to further increase protection. Rider and cover sheets may be used repeatedly before they must be scrapped, since they generally are made of alloy steel.

3. **Pusher-type furnaces** are of two general types. In one type the parts are pushed against each other, as in the continuous reheating furnace. In the other type, the parts are loaded in trays or other types of carriers which are pushed through the furnace.

4. **Conveyor-type furnaces** are constructed similarly to roller-hearth furnaces except that belt conveyors are used to carry the material through the furnace. They are suitable for accurately heat treating small miscellaneous pieces which would not ride properly on a roller hearth. Belt conveyors are made of alloy material of sufficient strength to carry the load and are resistant to heat, oxidation, corrosion and abrasion. A number of different designs of conveyors are utilized to satisfactorily meet the requirements. Some conveyors consist of several individual chains held on constant centers by spacer bars provided with suitable projecting lugs upon which the load, such as sheet or plate cut to length, rests. Many other belts are constructed of open mesh or woven chain to permit free circulation of hot furnace gases or protective atmosphere, while another construction utilizes pans or trays connected to a roller chain to carry the material. The production rate or heating cycle in this furnace type is controlled both by the temperature setting and by varying the speed of the conveyor.

5. **Walking-beam furnaces** employ a special mechanism within the furnace, known as a "walking beam" to move the material through the furnace. The walking beam consists of a number of alloy supports or beams which are arranged in rows of two or more beams to the row, throughout the length of the furnace. The beams are staggered with the one immediately ahead, and are

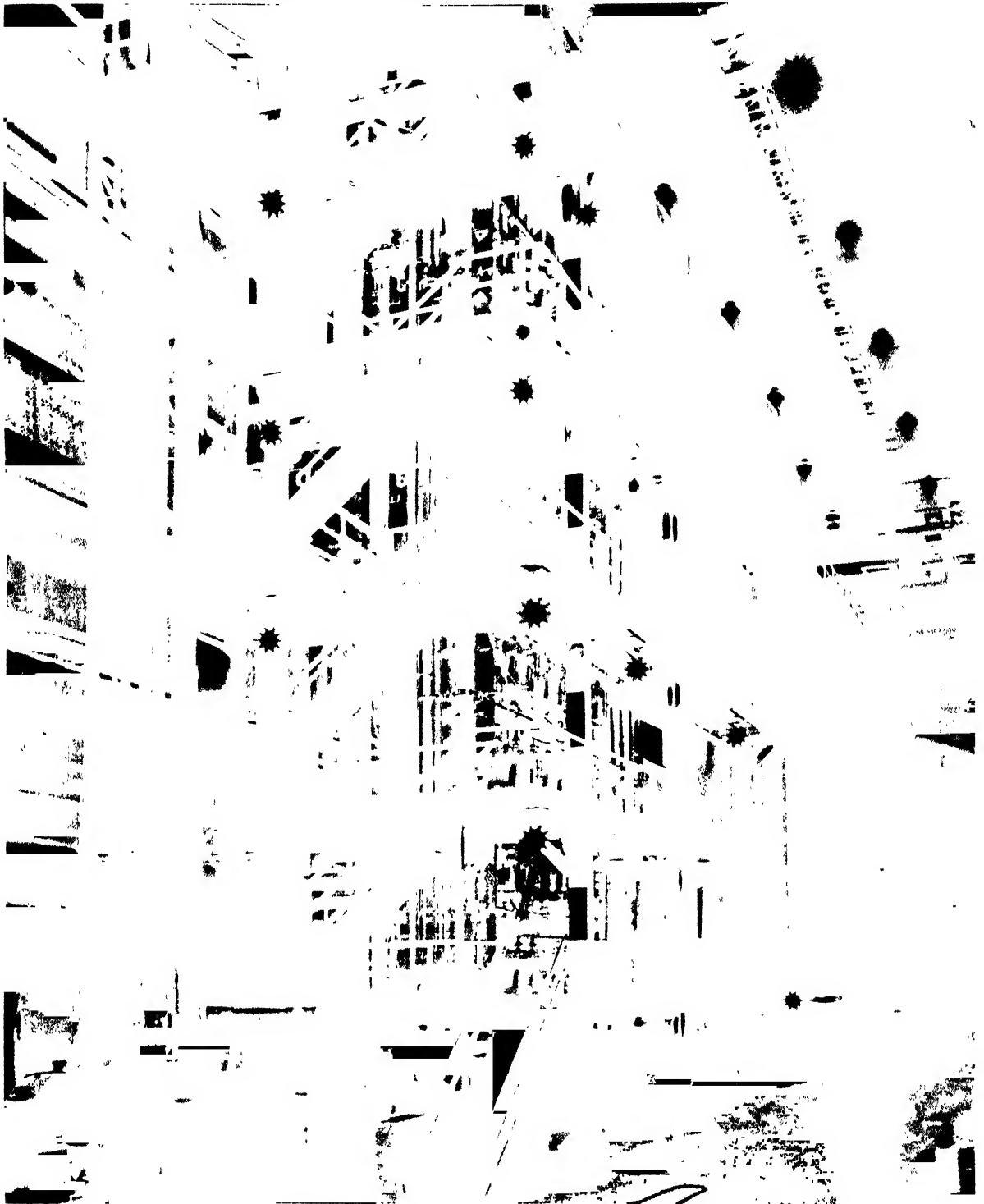


FIG. 21—21. Tower-type furnace in a continuous annealing line in a tin mill. The furnace proper, surrounded by structural-steel operating platforms, is 64 feet in height, and is heated by a combination of gas-fired and electric heating units. It has an annual rated capacity of approximately 136,000 tons.

placed in longitudinal slots in the furnace hearth. They are attached from below to toggles or cams that intermittently raise the beams, move them forward, and then lower them, thus depositing the material on the beams ahead. By this step action the material is moved

through the furnace at the desired rate. This furnace type is used commonly for tubes, bars, structural shapes or similar material.

6. In tunnel-type furnaces, the stock to be heated is placed upon cars which then are pushed or pulled

slowly through the furnace. This furnace type was used at one time for continuous box annealing of sheets. Furnaces used for this purpose sometimes reached 300 feet in length and were built in the form of a long tunnel, not necessarily straight, nor with a level bottom. The sheets were piled upon a base and sealed in a box for annealing. The box then was placed upon a small car which was pushed into the furnace. As the furnace was full of these cars, one car was removed as each car entered the furnace.

7. **Continuous strand-type furnaces** have been developed to reduce the extra handling and the long heating and cooling periods required in annealing sheet and tin plate in coil form. Heat treating uncoiled strip provides greater possibilities for the control of the time-temperature requirements for the entire piece and, therefore, a more uniform product. A coil can be processed in a matter of minutes or a few hours compared to the long cycle required in a batch furnace. Another special advantage of this furnace type is that other operations, such as cleaning or coating, may be combined with the heat-treating process to avoid extra handling and the expense of duplicated handling equipment for separate lines. Continuous strand-type fur-

naces are constructed either as horizontal or vertical units. Furnaces of the latter type of construction are sometimes referred to as **tower-type furnaces** and are used primarily to conserve floor space. Furnaces of this type utilize either electric or radiant-tube heating or both.

Figure 21--21 shows a modern tower-type furnace for annealing cold-reduced material at one stage in the manufacture of tin plate.

The horizontal type of continuous-strand furnace sometimes utilizes catenary suspension of the uncoiled strip, where neither rolls nor any other type of support are used throughout the heating zone. The heating zone of these furnaces may be from 20 to 50 feet in length. The preheating and cooling zones usually are constructed shorter than in the conveyor type and for some kinds of work are omitted entirely.

8. In **overhead monorail furnaces**, the material undergoing heating is suspended from rods that serve as hangers or even may be welded to the suspension rods. The suspension rods are attached at their upper ends to the carriers that operate on the monorail. If the rods are welded to the work pieces, they are removed after the assembly leaves the heating furnace.

Chapter 22

CONSTRUCTION AND OPERATION OF ROLLING MILLS

SECTION 1 TYPES OF MILLS

General Classification—The three principal types of rolling mills used for the rolling of steel are referred to as **two-high**, **three-high**, and **four-high** mills, shown schematically in Figure 22—1. As the names indicate, the classification is based on the manner of arranging the rolls in the housings, a two-high stand consisting of two rolls, one above the other; a three-high mill has three rolls, and a four-high mill has four rolls, arranged similarly. When rolling is in one direction only on **two-high** mills, and the piece is returned over the top of the rolls to be rerolled in the next pass, the mill is known as a **pull-over** or **drag-over** mill. This type of mill formerly was used mainly for production of light sheets and tin plate; it still is used by merchant mills for rolling of tool and high-alloy steels. On **two-high reversing** mills, the direction of rotation of the rolls can be reversed, and rolling is alternately in opposite directions, with work done on the piece while traveling in each direction. The long mill tables of reversing mills make it possible to handle heavy pieces in long lengths that would be impractical to roll on ordinary two-high mills, or to handle on the lift tables of a three-high mill (see below). The reversing two-high type of mill occupies an important position in the industry and, with the use of manipulators, it is possible to produce on it slabs, blooms, plates, billets, rounds, and partially-formed sections suitable for later rolling into finished shapes on other mills. In all three-high mills, each roll revolves continuously in one direction; the top and bottom rolls in the same direction and the middle roll in the opposite direction. The piece is lifted from the bottom pass to the return top pass by mechanically-operated lift tables, or by inclined approach tables. Usually the large top and bottom rolls are driven, while the smaller middle roll is friction driven. This latter roll is about two-thirds the size of the other two rolls, in order to permit removal through the housing windows. **Four-high** mills are used for rolling flat material, like sheets and plates, and represent a special type of two-high mill for both hot and cold rolling, in which large (idler) **backing-up** rolls are employed to reinforce the smaller (driven) **working** rolls. The use of four-high mills resists the tendency of long working rolls to deflect, and permits the use of small-diameter working rolls for producing wide plates, and hot- or cold-rolled strip and sheets of uniform gage. These mills often consist of a number of stands spaced closely together in one continuous line and are known then as **tandem** mills; the product passes in a straight line from one stand to the next. In **cluster** mills, each of the two small working rolls is supported by two (or more) **backing-up** rolls. This latter type of mill is used for the rolling of thin sheets.

ARRANGEMENT OF MILLS

A single stand mill, which may be either two-, three- or four-high, and either reversing or non-reversing, represents the most common arrangement for rolling a wide range of products, including blooms, slabs, plates, sheets, and various sections. **Guide**, **loop**, and **cross-country** mills are made up of several two- or three-high stands, or a combination of both, and are used for rolling of merchant-bar sections. **Guide** mills are small hand mills consisting of several stands of rolls in a **train**. Mills in train have the rolls of separate stands in the same line, the rolls of one mill being driven from the end of the rolls of an adjacent stand. Guide mills take their name from the metal guides which support the piece in the correct position during its passage through the grooves of the various passes. For example, it is possible to roll from an oval section to a round in one pass, provided the oval is supported by metal guides. In many guide mills it is the practice of the catchers, in order to save time, to start the piece through each of the passes before it is through the preceding one, thus forming a loop, resulting in this arrangement being called a **looping** mill. The layout of a mill of this type is shown in Chapter 30. There originated in Belgium the plan of setting up an independent roughing stand preceding the finishing train of the looping mill. This arrangement became known as a **Belgian** mill. On looping mills, it was found that the loop could be made mechanically by a tube or horse-shoe type trough, called a **repeater**, and thus dispense with the hand catchers. Prior to the looping mill, the piece was rolled throughout the entire length in one pass before it could be entered in the next pass. The looping arrangement eliminates the temperature difficulties encountered with long lengths. The shapes produced range from simple rounds and squares to intricate special shapes, but must be relatively small in cross-section. The **cross-country** mill is so named because of the scattered location of its roll stands, and was developed for rolling sections that, due to size or shape, are not adaptable to loop rolling. These mills involve the continuous idea, but the stands are placed so far apart that the piece must leave one set of rolls before entering the next (see Chapter 30). To save space and to avoid complicating the drives, the stands usually are arranged in two or more parallel lines, and the direction of travel of the piece is reversed during the rolling by employing transfer and skid tables. This arrangement results in a high-production mill of great flexibility, which may be used for a wide range of products, including structural shapes, rails, and splice bars. A continuous mill consists of several stands of rolls ar-

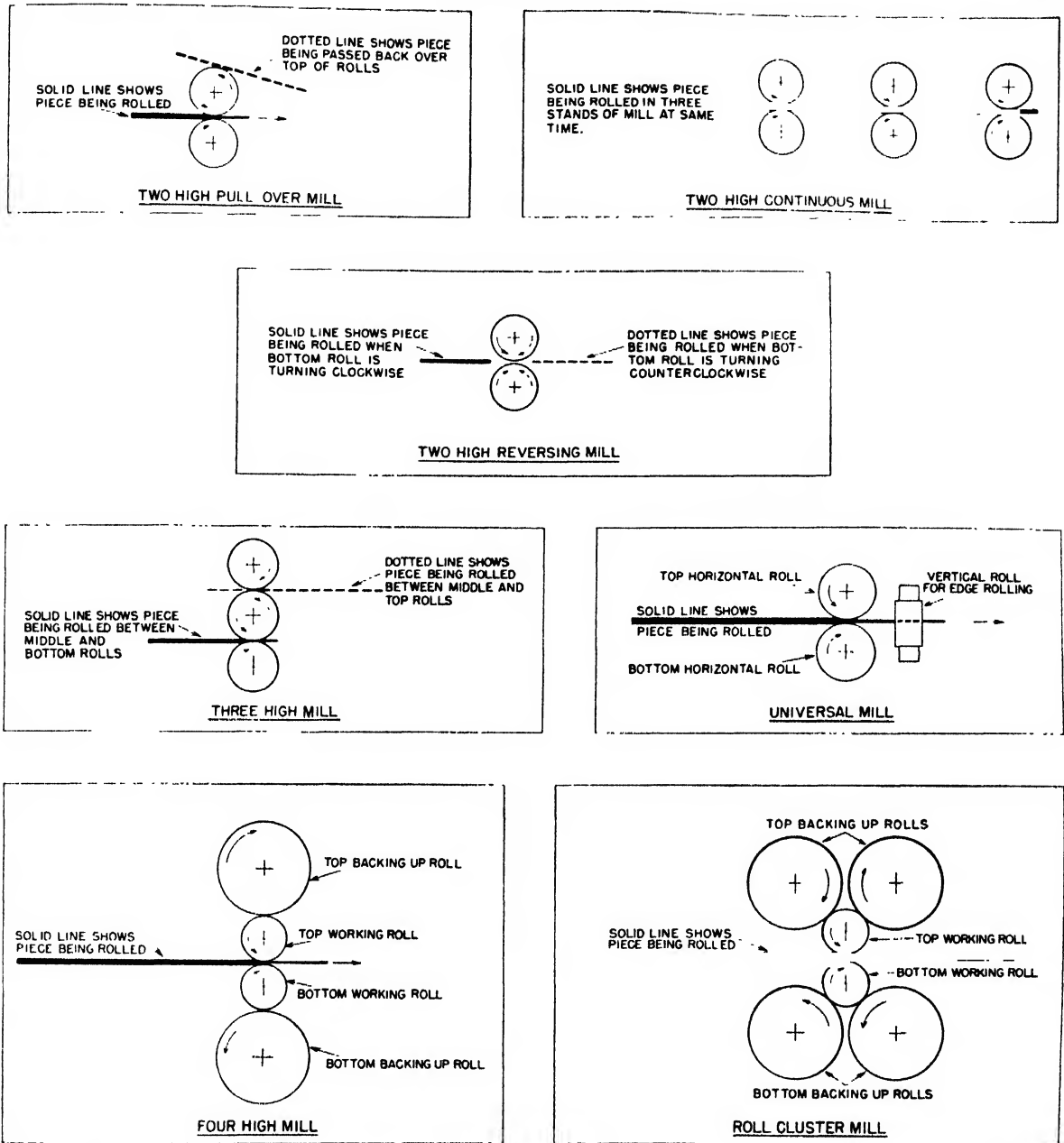


FIG. 22-1. Schematic representation of roll arrangements in the principal types of rolling mills.

ranged in a straight line (in tandem), with each succeeding stand operating with roll surface speed greater than its predecessor. Such a mill is illustrated diagrammatically in Part 2 of Chapter 25. Reduction takes place in several passes at the same time until the piece emerges as a finished shape from the last roll stand. This type of mill is in very common usage for rolling strip, sheet, billets, bars, rods, etc. A semi-continuous mill comprises also a four-high reversing roughing stand for reducing the piece prior to entering the continuous mill for reduction to the finished shape. This arrangement gives moderately high production with lower first cost than a continuous mill. Combination mills are those in which the roughing or major part of the reduction is

performed in a continuous mill, and the shaping in a guide or looping mill.

SPECIALTY MILLS

The universal mill is a combination of horizontal and vertical rolls, usually mounted in the same roll stand (Figure 22-1). The mill is made up of two-high (and occasionally three-high) horizontal rolls, with vertical roll sets on either or both sides of the horizontal stand. The vertical rolls also usually are driven. The direction of the piece is reversed after each pass in the mill. The universal mill is used to a limited extent also for plate product that requires rolled edges (see Figure 22-2). A special type of universal mill, known as the Gray

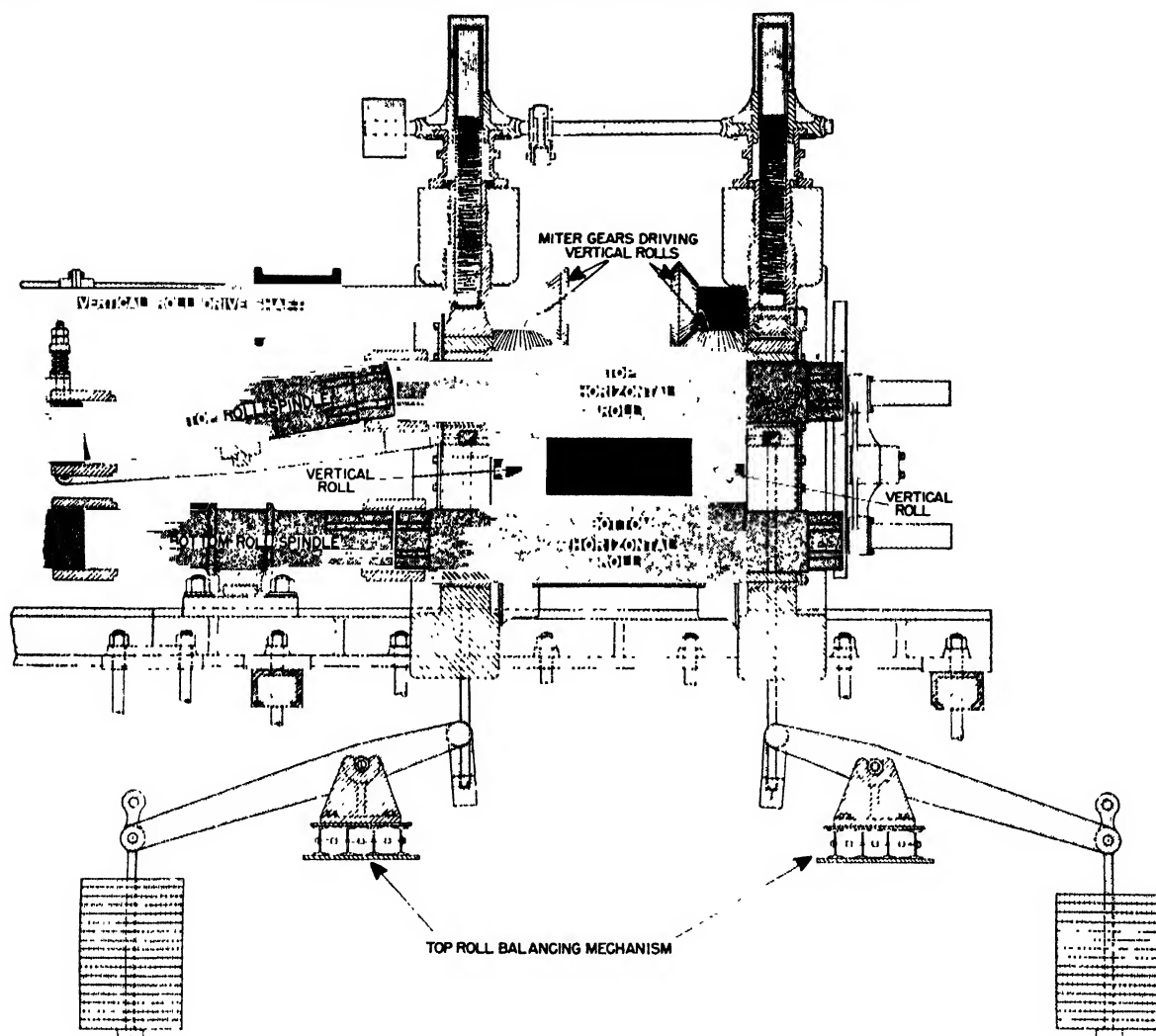


FIG. 22 -2. Diagrammatic layout of the principal parts of a universal mill for rolling plates.

mill is well adapted for rolling beams and H-sections of great width and depth without taper on flanges (see Figure 22-3). The horizontal rolls work on the web and flange thickness, while the idler vertical rolls in the same stand work simultaneously on flange thickness only. The roughing stands and intermediate stands are of the reversing type, and each has a separate stand of driven horizontal edging rolls which work on the flange height only. The finishing stand consists of the horizontal and vertical rolls in which the beams are given one pass only.

The **Wenstrom mill** is a similar modification of a universal plate mill, designed principally for rolling flats. Instead of acting upon the top and bottom and the two sides at different times, it does this simultaneously. The top roll can be adjusted vertically, and the bottom roll transversely, whereby pieces of different thickness and width can be produced with the same set of rolls. The **Sack universal mill**, designed principally for rolling cruciform sections, has horizontal and vertical rolls which act upon the piece simultaneously, the general arrangement being much like that of the Wenstrom mill. A somewhat similar principle is employed in the **Schoen mill** for rolling of railroad car wheels, whereby

the tread and flange are rolled simultaneously with the web, while rotating the forged wheel blank in a vertical position. This is accomplished by a pair of driven web rolls, and an idler tread roll in simultaneous contact with the wheel blank. A pair of idler rim rolls controls the width of rim (see Chapter 31).

For rolling of billets, rods, and narrow slabs, a continuous mill called a **Morgan mill** may be used (see also Chapter 40). It consists of a series of horizontal roll stands arranged one after the other, so that the piece is being rolled in a number of stands at the same time. The drive for each stand is through bevel gearing, with roll speed of each stand so proportioned that each set of rolls travels at a greater speed than necessary merely to compensate for the increase in speed due to elongation of the piece in the preceding pass, in order to keep the piece under tension at all times. Twist guides may be used to turn the piece 90° between passes. The **Garrett mill** (see also Chapter 40) is used for rolling small rods; it reduces the billet section in a roughing mill, which may be a looping train of three stands, or two or more stands arranged in tandem, followed by two trains of four or five stands each, lying end to end along two parallel lines in close prox-

ROLLING MILLS

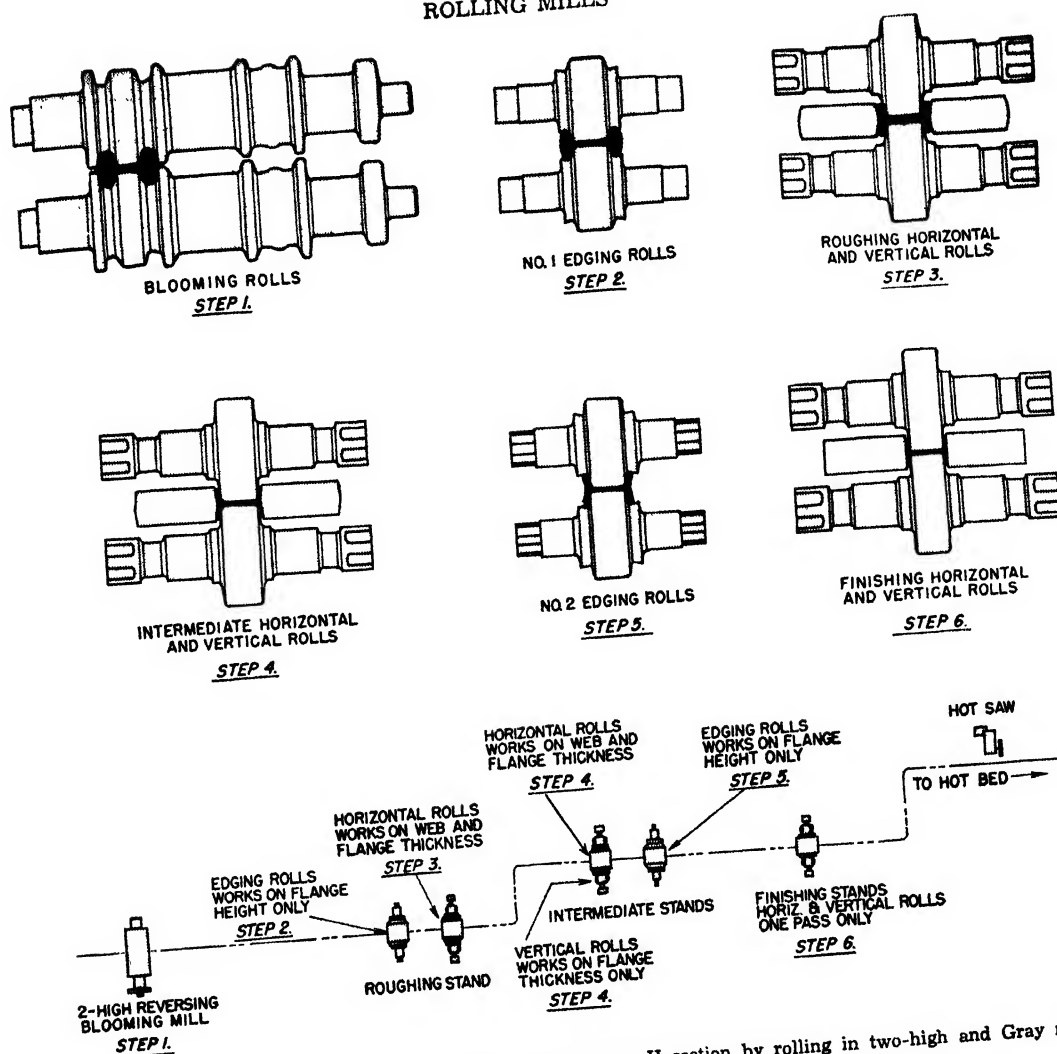


FIG. 22-3. (Above) The rolls and their functions in forming an H section by rolling in two-high and Gray mills. (Below) Flow diagram showing layout of the mill utilizing the rolls shown above.

imity to each other, and so placed that the first intermediate pass is in line with the last roughing pass, and far enough away to give proper clearance for the pieces. Generally, catchers are employed on the oval side, while repeaters are used only on the opposite (or square) side of the intermediate and finishing trains.

Another hot-rolling mill for billets, rounds and squares is the ingenious **Lamberton mill**, developed to avoid the need for a reversing motor. This mill is a two-high unit, with the motor turning in one direction only. The roll housing has two horizontal rolls in the usual position, but the shafts of the mill pinions are in the same horizontal plane. The mill housing is arranged to revolve through 180° , so that the rolls exchange positions after each pass, thus reversing the rolling direction. The bottom roll becomes the top roll, and vice versa, without changing their directions of rotation; thus, bars may be rolled in either direction without reversing the motor.

For the cold reduction of strip with a high degree of reduction in one pass, the **Krause mill** has had some application in the brass industry, and consists of a rolling head with the pinch taken between two small work

rolls, backed up by so-called cam plates for backing-up support. During reduction the strip is kept still, and the work rolls and the pinch are given the motion. The rolling head is reciprocated, taking a succession of short bites as the head goes back and forth, and as the thick stock is fed into the machine. A reciprocating type of machine for the incremental cold reduction of seam-rolled steel tubing is described in Chapter 41; this is the **Rockrite machine**. The **Sendzimir cold strip mill** for wide thin strip uses no backing-up rolls. The mill may have two, three, four or more caster shafts per work roll and may use one or more sets of intermediate rolls, between the casters and the small work rolls. The intermediate rolls transmit the pressure from the work rolls onto the casters, always in a plane perpendicular to the work rolls. The **Unitemper mill** is used as one means for temper rolling or skin rolling of finished cold-reduced strip. This mill consists of two stands of two-high mills, but with the second stand above the first in the same mill housing, to provide closer spacing between the two stands. With the motor of the first stand acting as a generator, and the motor of the second stand acting as a motor, the strip can be elongated a con-

trolled amount between passes with only a very slight reduction in each stand due to rolling.

Die rolling is the process of rolling a string of blanks, each of which has varying cross-sectional area produced by heavy reductions, and specified center to center length. When sheared to length the blanks are of identical shape. Products such as automobile axles and crankshafts are produced satisfactorily by die rolling. The blanks are rolled with or without flashing, depending upon the particular product section.

For producing seamless tubing, the types of piercing mills include the **Mannesmann** or parallel-axis barrel-type roll piercing mill, the **60° cone-roll** piercing mill, and the **Stiefel** or **180° disc** mill. For the operating principles of these piercing mills, refer to Chapter 41. A **continuous tube rolling mill**, built in 1948 by National Tube Division of United States Steel Corporation, consists of nine tandem individually powered stands of two-high grooved rolls. The rolls in the consecutive stands have their axes at 90° with each other. A cylin-

drical mandrel extends entirely through the pierced billet and passes through the mill with the work piece. The reheated tubes are processed in a tension reducing mill consisting of sixteen two-high roll stands, without the use of a mandrel. For producing butt-weld pipe the **Fretz-Moon tube mill** may be used. The skelp on leaving the furnace enters the first pair of rolls which form the piece downward, with the edges still apart. The second pair form the complete circle and bring the edges of skelp together, and the welding of the edges by their own heat takes place. The mill actually consists of three pairs of horizontal rolls and three pairs of vertical rolls. The last four pairs reduce and size the pipe, and furnish traction for pulling the skelp through the furnace and forming rolls.

Opportunity will be given in later sections to become better acquainted with the operating details of most of these mills. The types of rolls used in the more common mills, and their manufacture, will be discussed in Chapter 23.

SECTION 2

ROLLING-MILL ACCESSORIES

Many of the accessories of rolling mills are common to all types of mills, differing in design and operations to conform with the conditions in a particular mill. In addition to the rolls (described in Chapter 23), essential parts include the **mill drive**, **lead spindle**, **pinions** and their housings, **spindles** and **coupling boxes**, **chock bearings**, **screws** and **screw-down mechanism**, **edgers**, **front and back mill tables**, **manipulators** and **side guards**, **entering and delivering guides** and **roll-changing devices**. The newer mills may be equipped also with various control devices, such as pressure meters and automatic roll-setting devices. Many of these parts are discussed in the description of particular types of mills in later chapters. In the following, a discussion is presented of those parts essential to the operation of the rolls. To aid in locating some of the various parts described below, Figure 22-4 shows a cross-sectional diagram of a high-lift reversing blooming mill.

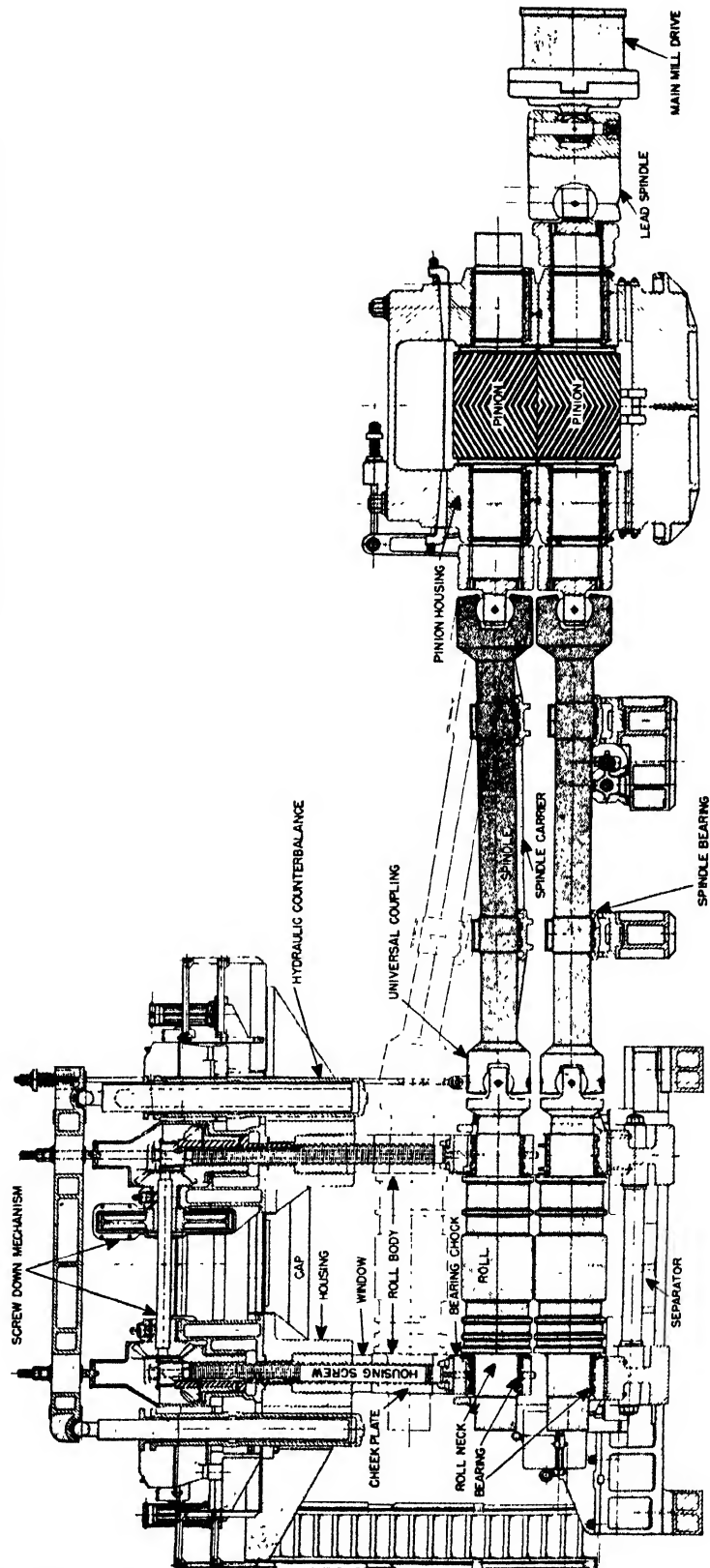
Lead Spindle—The lead spindle is used to connect the prime mover with the pinions, and may be of the universal type, either short-coupled, or long with carrier bearings, depending on the position of the motor in the layout. If short-coupled, standard flexible couplings can be used. The lead spindle is attached to the bottom pinion of two-high mills, and to the center pinion of three-high mills. If the pinions are of the universal type, they can be of generous proportions, as their diameter is not limited by the space available. Lubrication of all working surfaces is important. Grease usually is applied through a system of holes drilled in the jaws, and connected to various designs of spring-loaded cavities in the spindle body. The connection at either end of the spindle may be made by a **coupling box**, which is a hollow cylindrical casting corresponding in section to the wobbler, with one end fitting the spindle and the other the pinion (see Figure 23-1 of Chapter 23). The coupling box usually is made deliberately as the weak spot of the mill, either on the lead spindle or the spindles on the roll end. In the event of extreme overloading of the mill, the box breaks and disconnects the motor from the mill. The minimum length of spindle is slightly over twice the length of the coupling box that is used.

Pinions—The pinions are gears serving to divide the

power transmitted by the drive between the two or three rolls, driving the adjacent rolls in opposite directions. If twin-motor drives are used, no pinions are required, since the power is transmitted directly to each roll. The earlier pinions had either spur teeth or a divided face and staggered spur-type teeth, but the present practice is to use double helical teeth. The helical gears give a smoother drive, as some parts of the teeth are in contact at all times, making the transmission of power continuous. When rolling certain grades of materials, operation of the old type of pinion with spur teeth was characterized by jarring of the meshing teeth. This mechanical jarring was transmitted to the rolls and produced marks on the product being rolled. Pinions are made of cast or forged steel. They are mounted in babbit bearings and set in housings similar to those used for roll stands. The pinion stand must be of adequate strength to withstand the overturning effect of the full and maximum torque of the drive motor. Housings should be sealed to eliminate dirt and scale, and forced lubrication should be provided for the pinions. Except in plate, direct-drive and universal mills, the distance from center to center of pinions determines the size of the mill. The pinions absorb about 6 per cent of the power transmitted.

Spindles are used to connect the pinions with the rolls if the mill is not a direct-driven type; in the latter case the spindle is connected directly to the motors. Spindles are made of cast or forged steel and are fitted at each end with wobblers similar to those on the rolls (Figure 23-1), or with the universal couplings, depending upon the type of mill. The bottom spindle runs in spring-balanced carriers, and the top spindle is supported at its center of gravity by carrier bars balanced hydraulically or by counterweights (Figure 22-4). The spindles are supported by babbit or composition bearings, and should operate as nearly level as possible to prevent excessive power loss. It is very difficult to operate with a spindle more than 15 degrees out of level. This angle may be kept within the desired limits by increasing the length of the spindles. If the angularity is greater than 6 degrees or 7 degrees, a universal coupling should be used, in which case the ends of the wobbler are cut from a section of a sphere to give them the rounded form necessary to permit

FIG. 22-4. Schematic cross-sectional diagram of a hydraulically balanced high-lift reversing blooming mill, indicating location and nomenclature of principal parts. (Compare with the schematic diagram of a similar blooming mill equipped with mechanical counterbalances for the top rolls and spindles in Chapter 25.)



them to work at different angles, and the spindle is supported by saddle bearings in a carrier which rises and falls with the top roll and holds the spindle in place (see Figure 22-4).

The Bearings—The bearings, which support the roll necks and align the rolls, may be of two general types; roller bearings or the older chock-type bearings. One type of roller bearing is illustrated diagrammatically in Figure 22-5.

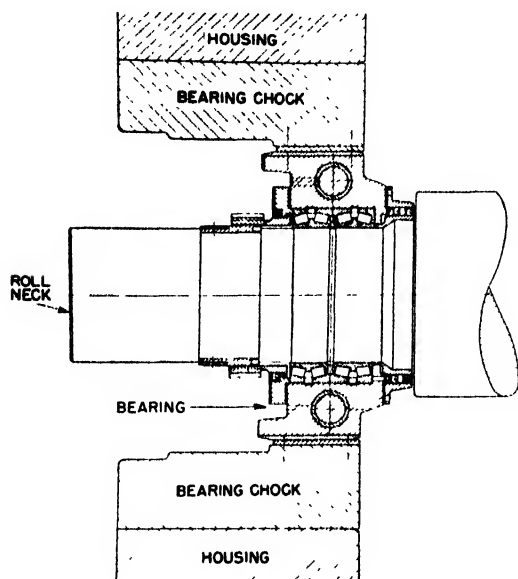


FIG. 22-5. Horizontal section through the center-line of a bottom work-roll chock on a plate mill equipped with roller bearings.

Roller bearings first were used for rolling mills in Czechoslovakia in 1921 and in Sweden in 1922, where they were applied to small hot-rolling mills. In America, use of the roller bearing accompanied the development of the cold-reduction mill for strip and sheet, and its adoption has been extensive since 1932. The development of this type of bearing was slow on account of the difficulties that had to be overcome; namely, (1) the roll necks had to be of adequate diameter to withstand the rolling loads; (2) the roll necks had to be kept free from wear; (3) the bearings had to permit quick roll changes; (4) the bearings had to permit adjustments of the roll laterally as well as vertically; (5) the bearings had to be self-adjusting to take care of slight changes in the shape of the rolls caused by heating and bending; (6) the bearings had to have ample carrying capacity; and (7) in case of roll breakage, provisions had to be made to keep the bearings themselves from being damaged. Roller bearings are now common in new mills, although they have not replaced entirely the chock type of bearing, which possesses the advantage of greater simplicity of construction.

Modern roller bearings are showing extremely long life, with resulting lower cost per ton of product. Mills equipped with roller bearings may be expected to use from 15 to 20 per cent less power for the main drives than mills using bronze or conventional bearings. As roller bearings require no adjustment for bearing wear, greater tonnages can be rolled with less variation in section from piece to piece, permitting requirements to be met with greater ease and improved mill yield. Roller bearings also result in a general reduction in

mill maintenance, and, as an automatic greasing system is used for lubrication, a clean mill operation is effected. Figure 22-5 is a horizontal section through the center-line of a bottom work-roll chock (sometimes called chuck) on a 160-inch four-high reversing plate mill.

Chock Bearings—The chocks usually are made in two parts, the bearing with the surface of its lining in contact with the roll neck, and the chock, which holds the bearing in place against the roll neck. The bearings may be lined with babbitt or may be made with babbitt, brass or bronze inserts or they may be non-metallic bearings of the phenolic-resin type.

The alloys used in the bearings vary considerably. Table 22-I gives an approximate composition of some of the more common alloys used for bearings.

Table 22-I. Compositions of Typical Bearing Metals (Per Cent).

Metal	Cu (%)	Zn (%)	Sn (%)	Sb (%)	Pb (%)
Red Brass	85	15	—	—	—
Yellow Brass	65	35	—	—	—
Bronze, No. 1	85	—	15	—	—
Bronze, No. 2	82	15	3	—	—
White Metals	—	—	10-15	12-20	65-80
Babbitts*	—	—	59-91	4-12	0.35-26

* As, 0.1 max.; Bi, 0.08 max.; Fe, 0.08 max.

The tin in babbitt bearing metals can be replaced satisfactorily in most instances by substituting lead-base babbitts for tin-base babbitts and making, in some cases, additions of silver or arsenic to increase the hardness of the alloy at elevated temperatures.

As bearings made of all of these metals are soft and not very strong, it is necessary to insert them in supporting castings which are set into the housings. These castings are box-like in shape, each one containing on one side a semicircular groove corresponding to, but larger than, the necks of the rolls. In order to reduce weight, the castings are cored out, and may be of either iron or steel.

Reference has been made above to the use of bearings of the phenolic resin type. These composition bearings of a laminated type operate with water as a lubricant and have replaced the bronze and babbitt types of bearings on a number of mills, including some blooming mills, plate mills, bar and billet mills, rod mills, strip mills, skelp mills, tube mills, three-high sheet mills, and structural mills. Bearings of hard wood, with water as a lubricant also have been used successfully for certain mills. The advantages of the composition and wooden bearings include longer life, freedom from greasing, and reduced power requirements. Failure of this type of bearing in some mills may be due to one of several causes. The heat conductivity of such bearings is very low, and heat generated by friction must be removed continuously with adequate cooling water. Roll necks for these bearings must be smooth to prevent excessive bearing wear.

Arrangement of the Chocks—In a two-high mill, a chock is placed under each of the necks of the bottom roll and above each of the necks of the top roll. In case the top roll is adjustable, light bearings must be placed also under the necks of this roll for support; the upward thrust of the top-roll balancing device is exerted against these supporting bearings. The top-roll balance is for the purpose of keeping: (1) the top-roll necks in contact with the upper chock bearings; (2) the chocks tight up against the screw points; and (3) the

screw-thread surfaces in contact. On heavy mills, the counterbalance may be of the overhead hydraulic-cylinder type connected to an accumulator system, or of the underneath counterweight-and-lever type; both types of counterbalances can be compared in Figures 22-4 and 25-8. On small mills, screw bolts extending through the housing serve the same purpose. The exposed half of the necks of the lower roll usually will be covered to protect them from scale, etc.

The arrangement of chocks in three-high mills is more difficult. The simplest way is to place double-groove chocks between the top and middle and the middle and bottom rolls, and then set them in the housings one above the other, so that all the adjusting made necessary by the wearing away of the bearings and the material of the rolls themselves may be made with large screws in the top of the housing that can be turned to move upward or downward. However, this arrangement causes the bottom bearing to wear down rapidly and increases the power required to drive the mill, owing to the additional friction induced on the bottom bearing by the weight of the upper two rolls and their chocks. This fault may be overcome in two ways: (1) by making the bottom roll fixed and supporting the weight of the upper two rolls and their chocks on the shoulders of the chocks themselves, the distance between rolls may then be regulated with shims, or "liners," by adding or removing the shims as the bearings wear down; (2) a better way, and the one most often employed in modern mills, is to make the middle roll fixed, in which case the bottom roll is raised or lowered by an adjusting wedge attached to a screw in the housing which permits it to be moved back and forth with a wrench from the outside of the housing.

In all mills, two-high as well as three-high, the top chocks are held down by two strong screws which work in threaded holes or nuts in the tops of the housings.

The functions of the chocks are not only to furnish vertical bearings for the rolls but also to prevent their movement laterally as well. This lateral displacement of the roll is prevented by the inner edge of the bearing which is formed to fit against the shoulder of the roll. Adjustment for wear in this direction is provided by adjusting screws which extend through the side of the housing and bear on the ends of the chocks. This lateral adjustment is a matter of great importance in rolling sections that require grooved rolls, the reason for which is self-evident.

Housings—There are two housings for each stand of rolls (see Figure 22-6). They may be made either of iron or steel, the choice of materials depending upon the size of the mill and the strength required. Generally they are of annealed cast steel, but housings may be

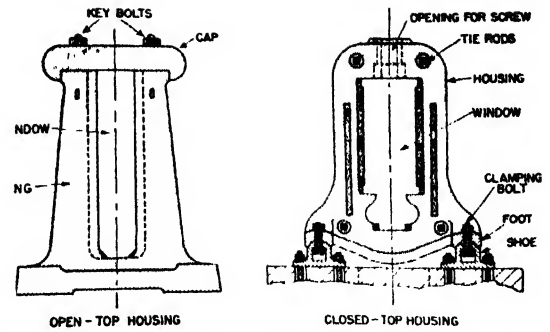


FIG. 22-6. Diagrams indicating principal parts and their names for (left) a closed-top housing and (right) an open-top housing.

of welded steel construction, heavy plates or slabs. The housings are of an "O" or "U" form, each having an opening called the **window**, which serves as a receptacle for the bearings. Housings may be either closed top (O) or open top (U). In the former, the base, the two legs and the top are all in one piece, while in the latter, the top will form a separate part which can be removed. The base of the housing is cast with a projection on each side to form the feet. In the bottom of each foot is cut a groove which fits over a **shoe**, running parallel to the rolls. Suitably shaped bolts then serve to clamp the foot of the housing to the shoe, which is fastened firmly to the foundation by long bolts. This method permits the housing to be moved laterally, and facilitates the lining up of the mill. The tops of the two housings in a set are prevented from spreading apart by suitable tie rods or, in the case of open-top housings, the top for both housings may be cast in one piece. Similarly, tie rods usually will be placed at the bottom. Recesses or other openings are cast in the inside face of each housing to receive the supports for the guards and guides, these supports being usually in the form of square rest bars which extend from housing to housing in front of the rolls. The immense pressure applied to the rolls between the top and bottom of the housing acts as a stretching force on the uprights of the housings; the degree to which the housings resist this force is an important factor in determining the reduction that can be effected in one pass and also the exactness with which the thickness of the piece is controlled. The uprights of the housings sometimes are referred to as posts.

A screw-down mechanism is used on mills to position the top roll for each pass through the mill, except on continuous and three-high mills where fixed passes are

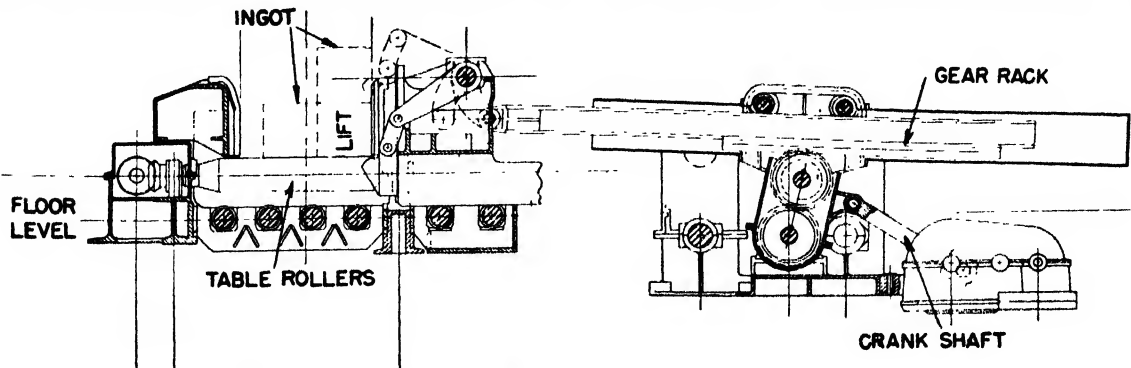


FIG. 22-7. Schematic diagram of the main operating parts of a typical manipulator, showing motion of the lifting fingers that raise ingot by a corner in the process of turning it 90 degrees.

used. The top roll is adjusted by screws which extend through the top of each housing. The screw-down bearings are of roller or antifriction type. The transmission of power was first accomplished by hydraulic pressure, but in modern mills the speed of the operation has been greatly increased by electric drives. On one recent installation, the maximum rate of raising or lowering the top roll is 38 feet per minute, compared with 2 or 3 feet per minute on the older mills. On small mills where the adjustment is only occasional, the screws will be operated by hand with spanner bars. In all cases, compression of these screws is unavoidable and, combined with the stretch of the housings, produces the spring of the mill, which in some cases is surprisingly large.

Edgers or edging rolls are used to give a universal or rolled edge to the product. The edging unit may be a roll stand separate from the horizontal stands, but in the universal-type mill the edging unit usually is attached to the main-roll housings. The edging unit consists of two vertically-mounted rolls, each manipulated by screws of identical size, so that they move in unison in opposite directions with respect to the centerline of the mill. Each roll is operated through a screw-down mechanism like that used on the top horizontal roll of a reversing or universal-type mill, with the exception that the motion is horizontal instead of vertical.

Front and back roller tables feed and receive the piece during each pass. Their speed should be matched closely to that of the rolls, and the width should be the same. On reversing mills these tables are subject to heavy usage due to impact from turning the piece, and should be of rugged construction. Forged steel generally is used today in making the rollers, which in new mills are mounted in roller bearings and equipped with automatic lubrication. In the older mills, bronze or babbitt bearings are lubricated by hand. The rollers usually are electrically driven by a line shaft through miter gears.

The manipulator consists of a pair of side guards, stroking laterally over the rollers of the front or back mill tables, usually on both sides of the mill (Figures 22-7, 22-8 and 22-9). The manipulators operate to turn the piece between passes, to move it to another pass, and to straighten it when necessary. Most manipulators are constructed so that both horizontal and vertical motion is possible. In most mills, these units are operated by electric or hydraulic power. Depending upon the size of ingot or section to be manipulated, the height of side guards may run up to 48 inches. Lifting fingers, always on the front manipulator and sometimes on the back one also, are incorporated in the guard or guards toward the drive side. These have a vertical or nearly vertical stroke and serve to lift the piece by a corner in the process of turning it 90 degrees.

Guides and Guards—In order to prevent collaring and to insure that the piece enters and leaves the pass in the correct position, guides are employed. These guides vary in form and size to fit the conditions. In some cases, they are merely grooved fore-plates; in others, they are blunt-edged plates set up in front of the collars, dividing the space in front of the rolls into a series of pigeon holes; in large mills rolling heavy sections, they may take the form of grooved rollers; in the smaller mills like the guide mills, they are trumpet-shaped castings that fit close up to the roll and have exit openings to conform to the shape and size of the section of the entering piece; in continuous mills they may be constructed so as to twist and thus turn the piece between two successive passes. Guides may be

ACTION OF A MANIPULATOR TURNING A SLAB 90°

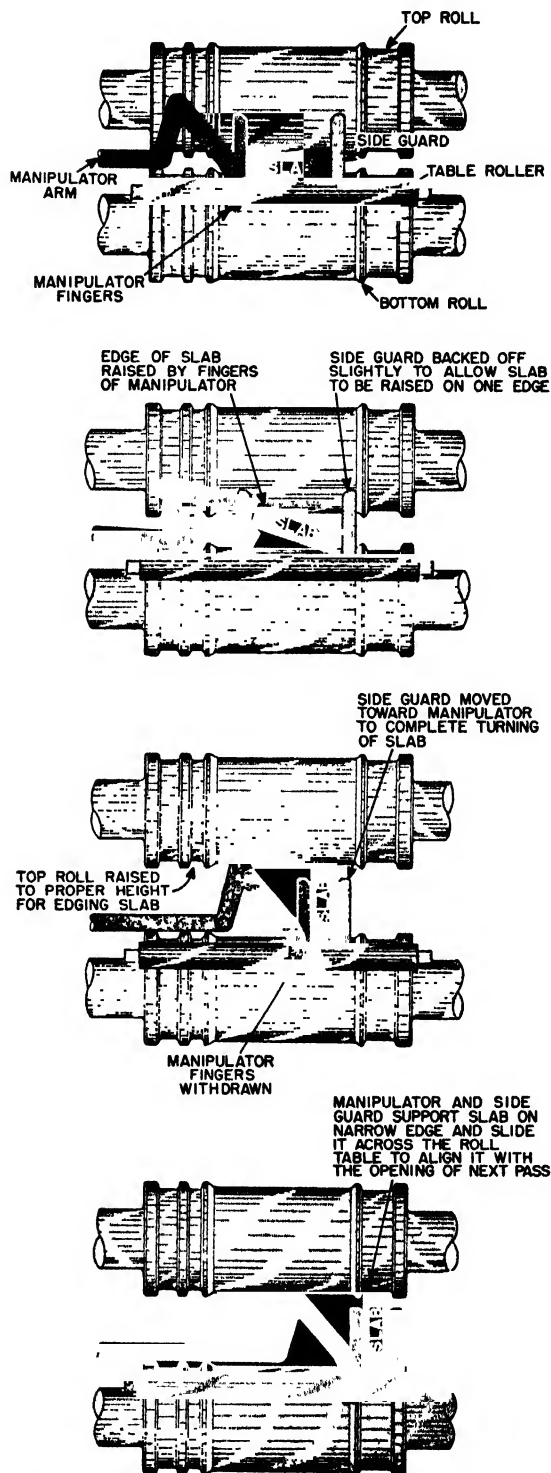


FIG. 22-8. Schematic representation of the actions of the principal parts of a manipulator in turning a slab 90 degrees. The mechanisms have been simplified and exaggerated in size for the sake of clarity.

FIG. 22-9. Blooming mill in operation rolling a round section, showing location of manipulator with respect to stand.

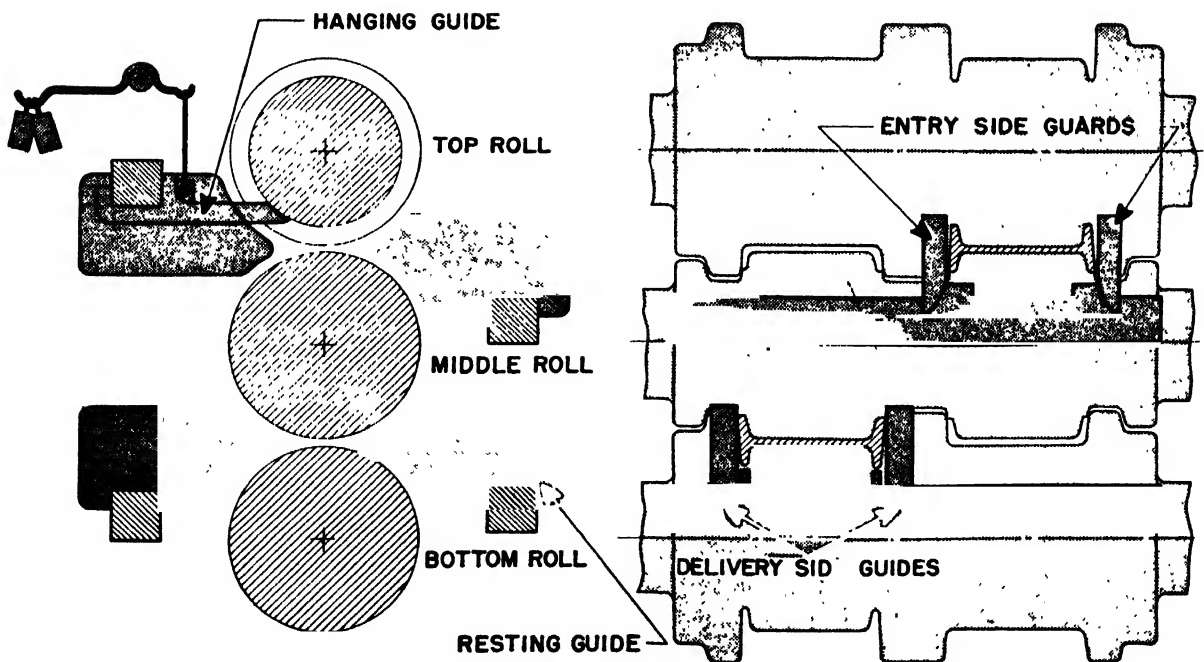
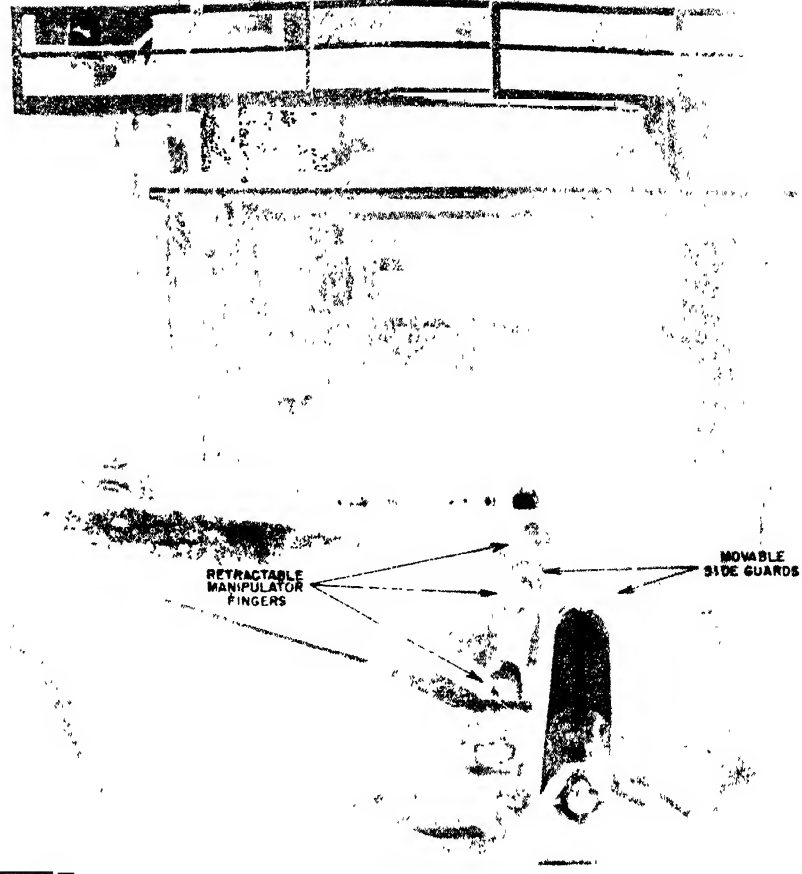


FIG. 22-10. Schematic representation of guides and guards applied to a three-high mill.

employed on both sides of a pass, in which case they are designated as entering guides and delivery guides. They are held in place by means of the rest bars previously mentioned in connection with the housings. Guards are devices employed mainly on the delivery side of the mill to control the direction of the piece after leaving the pass. Reversing and three-high mills are provided with guards on both sides of the mill (see Figure 22—10).

Roll-changing devices are dependent upon the construction of the mill housing. The rolls in an open-top housing are changed by removing the housing cap and picking out the rolls separately or collectively with crane slings. Roll changing in closed-top housings is accomplished through the window of the housing

either: (1) by using a counterweighted porter bar, (2) by a sleeve utilizing the ingoing roll as a counterweight, or (3) by a "C" hook. All of these methods require overhead crane service to remove the rolls one at a time. The rolls are pulled out of the housing by attaching a socket fitting over the protruding roll necks. In some large modern mills, a roll-changing rig is used to remove worn rolls and install new ones. The rig is placed in a permanent mounting, level with the housing sill, and consists of a rack-and-pinion motor-operated crosshead mounted on a rail frame. It withdraws both top and bottom rolls with their respective bearings, all at one time. The old setup is removed from the rig by the crane and the new rolls and bearings are mounted on the rig to be pushed back into the mill.

Chapter 23

ROLLING-MILL ROLL MANUFACTURE

SECTION 1

ROLLING-MILL ROLL DESIGN

Principal Parts of Rolls—Of the essential parts of the rolling mill, the rolls are of the greatest interest, as they control the reduction and shaping of the metal. There are three parts to a roll; namely, the body, or the part on which the rolling is done, the necks which support the body and take the rolling pressure, and the wobblers, where the driving force is applied through loose-fitting spindles and boxes which together form a sort of ingenious universal coupling. These parts are shown in Figure 23—1.

A plain-surface or cylindrically-bodied roll is used (in pairs) for rolling sheets and plates, while for bars and shapes, grooves of suitable design are turned in the roll bodies. Such grooves are called *passes*.

Figure 23—2 shows two examples of grooved rolls:—a three-high set of rolls with 98 degree diamond passes, with the roughing-down operation from a billet indicated by the hatched lines, and a two-high set of rolls with grooves for rounds. Rolls are turned as required for a multitude of shapes. Figure 23—3 shows some of these grooves or passes and their nomenclature. The dotted lines over each pass show the cross-section of the piece leaving the preceding pass and entering the pass in question.

Procedure in Designing—When a new section has been approved for rolling, a detailed drawing of the section is sent to the roll designer, together with instructions to proceed with the design.

In most cases, though not all, a cold-finishing templet of the section is made which is an exact cross section of the bar to be produced by rolling. This is the starting point in the design. The templet work is done by skilled artisans called *templet filers* who must work to very close tolerances. As the bars are finished hot, a certain amount of shrinkage takes place in cooling. To allow for this shrinkage it is necessary to prepare a hot-finishing templet to which the rolls are turned. The shrinkage varies with different finishing temperatures but, on the average, is $\frac{1}{64}$ inch to the inch.

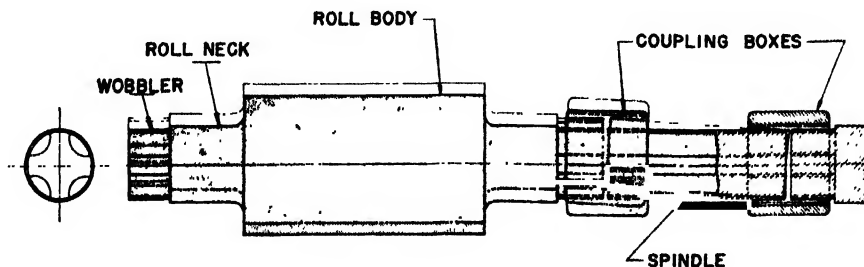


FIG. 23—1. Nomenclature of the parts of a rolling-mill roll and the units that connect it to the motive power driving the mill. The broken lines indicate position of the parts when the roll is raised.

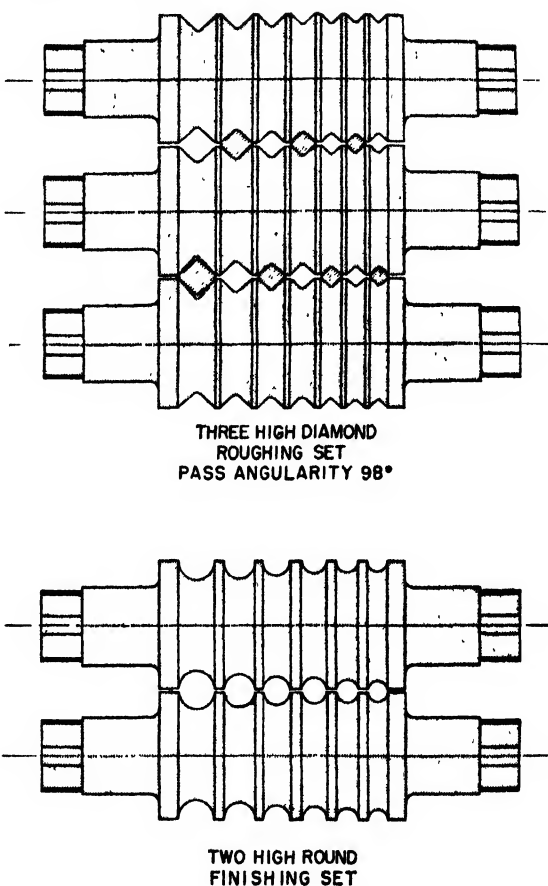


FIG. 23—2. (Above) Set of three-high rolls with diamond passes, showing the seven passes used in roughing down a billet to a round-cornered square section. (Below) Two-high set of rolls with grooves for finish rolling of rounds.

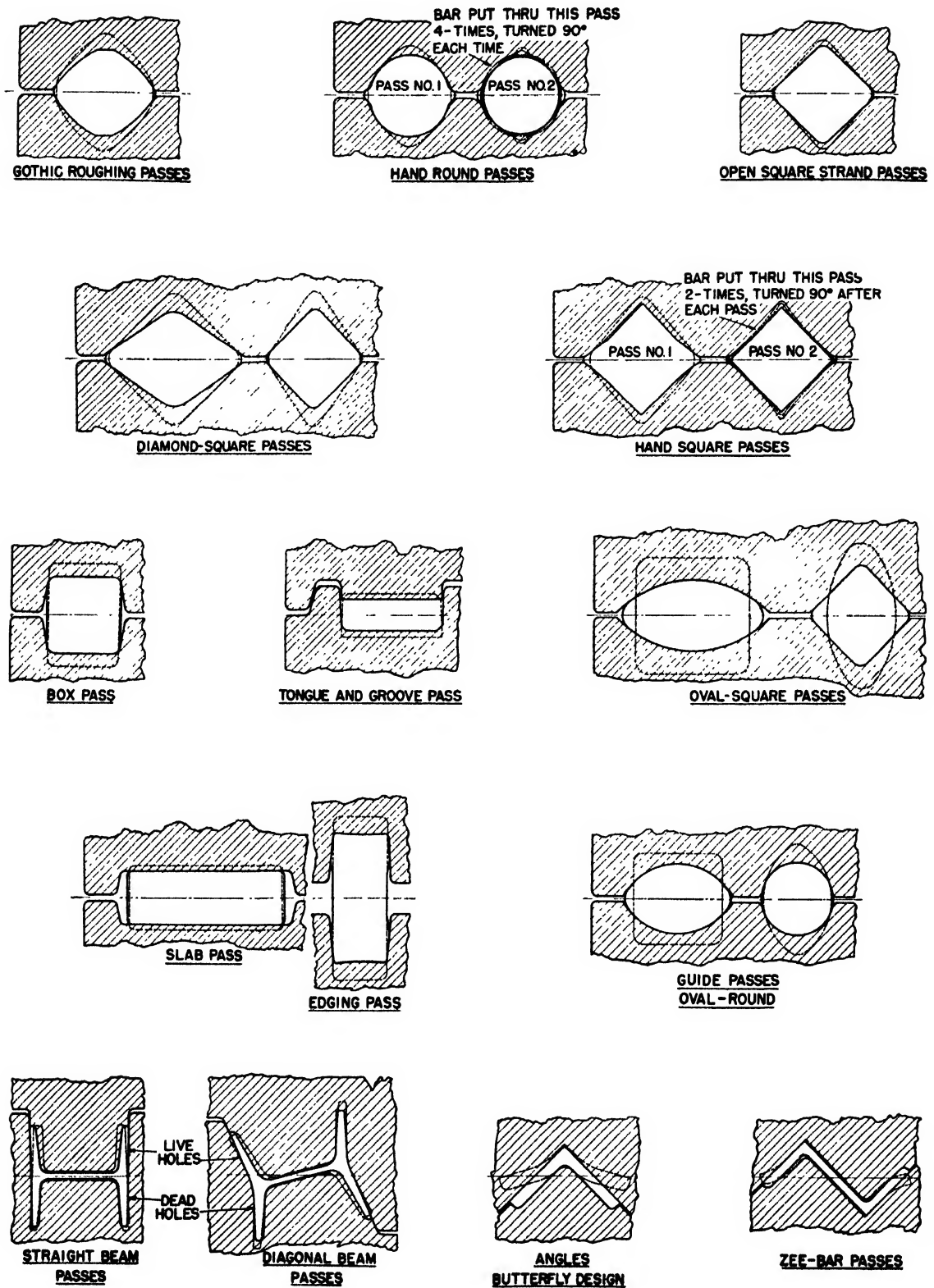


FIG. 23-3. Diagrammatic representation of some common types of roll grooves or passes. The cross-hatched portions are axial sections of the roll bodies. Dotted lines indicate cross section of the piece entering each pass.

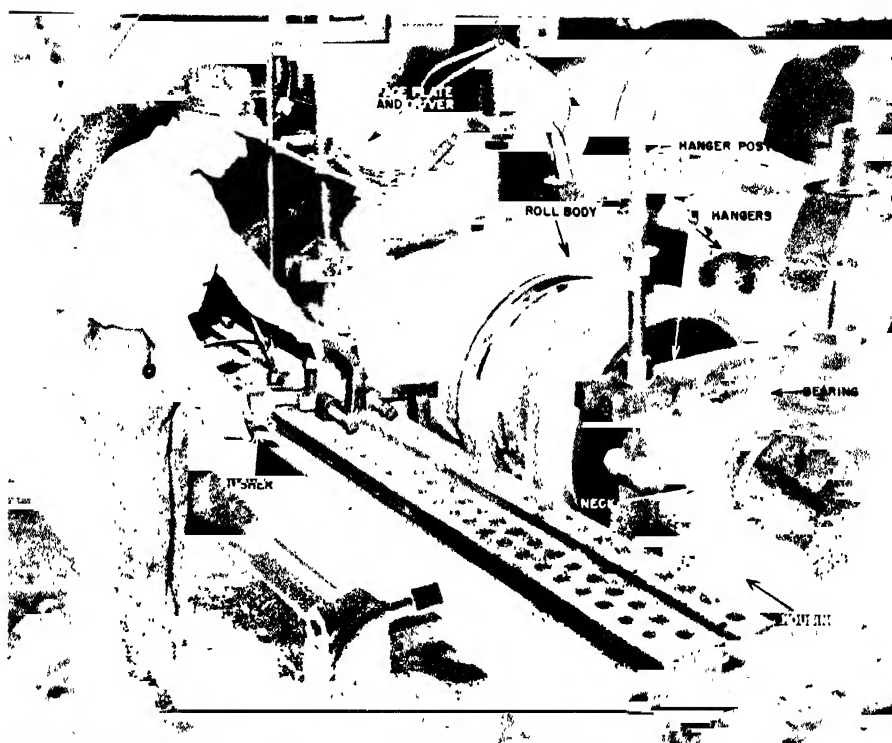


FIG. 23—4. This illustration shows the operation of a roll lathe, turning grooves in a shape roll.

The passes in the design are drawn very accurately from the finishing pass to the starting billet, bloom or ingot. Where it is necessary to use a definite size of billet or bloom, the design must be shaped so that the available billet or bloom will be satisfactory.

Plants having primary blooming mills have an advantage in that special sized billets, blooms, or shaped blanks may be made exactly to the size required.

In designing the passes, it is customary to draw each preceding pass, from the finishing pass to the initial pass, over the succeeding one, and the roll designer judges the correctness of the design by this method, drafting the passes in the form which his knowledge and experience has shown to be proper. This part of roll design has been referred to as intuitive ability in design and is a faculty not readily transferred to others.

When the templets have been prepared, drawings are made and roll castings ordered from the drawings. When the castings have been received, the rolls are turned on special lathes to match the drawings and to an exact fit of the templets. As will be learned in Section 2 following, rolls are cast from either iron or steel, with or without alloying elements, depending upon the use to which they are to be put.

Roll lathes are different from the usual machine lathes. They are very rigidly constructed and all tool movements are made by screws and wrenches. Figure 23—4 shows a roll turner at work on a shape roll.

The pass guides usually are designed by the roll department and are a very necessary part of any successful roll design. Many shaped sections are so cut in the rolls that they would have a tendency to wrap around the rolls unless stripped from the pass by guides. Entering and delivery pass guides also serve their purpose in guiding and delivering bars into the proper pass in a straight line. Figure 22—10 illustrated the guide principles.

Elements of Good Roll Design—In the designing of rolls there are many points to be observed if the design is to be satisfactory.

In determining the number of passes, one point to be remembered is that the fewer the passes, the smaller the roll expense. Drafts (amounts of reduction per pass), however, must be suited to roll diameters and plant heating capacity so that roll breakage will be small, and excessive use of power will be avoided.

The passes must be designed with liberal tapers on the various parts so that they may be dressed to templet with the minimum possible reduction in diameter.

A successful design also takes yield into consideration. A minimum of section variation, giving a bar uniform in size from the front to middle and back, insures good yield. Yield is defined as a measure of good product made from a given quantity of starting material.

The tonnage which may be obtained from the rolls also is important. The design which permits the greatest tonnage to be rolled before it is necessary to scrap the rolls because of their reduction in size, due to dressing to eliminate the effects of wear, gives the lowest roll cost.

In roll design, the fact that material being rolled elongates and also spreads in proportion to the draft must be given the most careful consideration, as this is the fundamental principle in rolling, and the proper directing of this elongation and spread is vital to the success of a roll design.

Spread of the various kinds of material must be taken into consideration, as there is a great deal of variation in spread of steels. Roll speeds must be considered in the design. High speeds in many cases restrict spread and low speeds increase it. Low speed permits heavy drafts, while high speed requires lighter drafts.

Temperature of the steel is also a factor. High temperatures permit heavy drafting, while low temperatures call for lighter draft.

Roll diameter also is important. A roll of large diameter is strong and permits heavy drafts without roll breakage, and also, because of the large diameter, the area of contact of roll on bar is large, permitting easy entrance (see Chapter 19). Rolls of small diameter re-

quire less power to drive, reduce spread and increase elongation, but are more easily broken, and heavy reductions cannot be taken readily on them as the area

of contact between the work-piece and roll is small. All these points and many more must be given full consideration in any successful roll design.

SECTION 2

CASTING OF ROLLING-MILL ROLLS

The selection of the proper grade of roll to be used in any of the various stands of the rolling mills is most important. In general, primary mills such as blooming or slabbing mills require a roll in which strength is paramount. Such rolls are subject to tremendous shock and extreme pressures in the rolling of large ingot masses with heavy reductions. The heat from the ingots transmitted to the roll also has a tendency to cause surface or fire cracks through differential expansion of the surface, and the strength and toughness of the rolls must be such as to resist the further development of these cracks. The original plain-carbon steel roll containing 0.30 per cent to 1.50 per cent carbon, used in the past for such applications, now has been supplanted largely by the alloy-steel roll of greater strength and durability.

From the primary mills through the secondary and so on to the finishing mills, the required rolls generally become smaller and harder and any of a variety of roll grades may be used; carbon-steel or alloy-steel rolls for roughing, iron or alloy-steel rolls for intermediate work, and the various grades of iron-base rolls for the finishing stands. Specifically, however, each roll must be custom made to suit the requirements of the individual mill by a procedure within the limits of the manufacturer's equipment. For this reason, no two manufacturers use exactly the same procedure for the same roll and each must apply his own specifications for a given requirement.

Steel-base rolls, as now produced, generally are cast from steel made in acid open-hearth furnaces, while iron-base rolls are usually made from metal melted

in reverberatory-type air furnaces. In some cases, both types of rolls are made of material melted in electric-arc furnaces, either acid or basic. Although the fundamentals for making steel and iron rolls are similar in many respects, important differences exist.

STEEL-BASE ROLLS

In the casting of steel rolls, as indeed in the casting of any shape, the greatest importance must be attached to the control of directional freezing of the molten metal in the mold after casting. To insure soundness in the casting, solidification must occur progressively from the smallest to the largest cross section. Shrinkage occurring in each step of solidification must be compensated for by still-molten metal drawn from adjacent unfrozen regions. Shrinkage of the interior of the largest section, which is last to freeze, is compensated for by a feed-head or sink-head of large volume provided to act as a reservoir for molten metal (see Figure 23-7). To obtain the advantage of gravity feed during this freezing-feeding process, as well as to satisfy mechanical requirements, the feed-head is always placed on top of a roll mold which has been set vertically on end, as shown. The effective cross-sectional pattern thus obtained should approach the shape of an inverted cone. Due, however, to the required design of the roll and the general rule that the drive end is best placed down, this optimum arrangement is often distorted or even inverted. In these cases provision must be made to control the freezing pattern by artificial means. This may be done in several ways, one of which is "padding" or adding extra thickness of metal to

FIG. 23-5. Illustrating use of a sweep in preparing a mold for casting a steel roll. (Courtesy, Pittsburgh Rolls Corporation.)





FIG. 23-6. Assembling the two halves of a mold for a steel roll. (Courtesy, Pittsburgh Rolls Corporation.)

sections which are too small to solidify in the proper manner in relation to the balance of the casting. Increasing the rate of freezing by inserting heavy metal blocks, rings, or segments, called chills, in the mold, close to the surface contacted by the molten metal, is still another means for controlling the freezing pattern. Consideration must be given also to the fact that, in addition to increasing the freezing rate, chilling in the manner described also promotes formation of a dense, refined outer skin of the roll at the contact areas, which

in most cases is very desirable. Although rolls are made with meticulous care in both molding and pouring, particles of the mold sand frequently are entrapped accidentally on the surface and, in order that these or other surface defects will be removed properly during subsequent machining, rolls are cast slightly oversize. The extra stock thus cast on a roll must be of such thickness that, in those instances where heat treatment precedes the other processing, the tough, heat-treated skin will not be removed during subsequent machining.

The initial step in the manufacture of steel rolls is that of examining the blue print of the roll; the composition and heat treatment are determined from the requirements indicated. The pattern is designed after applying the metallurgical and mechanical considerations discussed above. This pattern is generally in the form of a sweep; that is, a flat board carved on one edge to conform to the longitudinal contour of the roll as modified for casting, with provision on the other edge for fastening the sweep securely to a spindle or axle (Figure 23-5). Then the plan of manufacture is given to the operating section.

The steel flasks or containers for roll molds are cylindrical in shape, and designed to separate into two longitudinal halves. Proper molding sand is first rammed into each half, after which the sweep mounted on the spindle is attached to the half flask and rotated, cutting the sand before it to the shape of the sweep contour. The same procedure is used on the second half-mold. Figure 23-5 shows a mold used for casting a large-diameter universal-beam mill roll. After molding, the half-molds are baked in drying ovens at temperatures somewhat above 400° F to drive out moisture and bond the mold thoroughly. Figure 23-6 shows the two halves of a mold being put together. Figure 23-7 shows the vertical section of a mold for an alloy-steel blooming-mill roll in position for pouring. Rolls are bottom poured and the metal enters the bottom neck through a gate tangent to the periphery of the neck. This is to set up a swirling motion of the metal as it enters the mold. This centrifugal action concentrates dirt and foreign particles in the center of the rising molten metal in the roll mold, and as the metal continues to rise, they are carried to the top into the sink-head. The dotted lines seen in Figure 23-7 show the outline

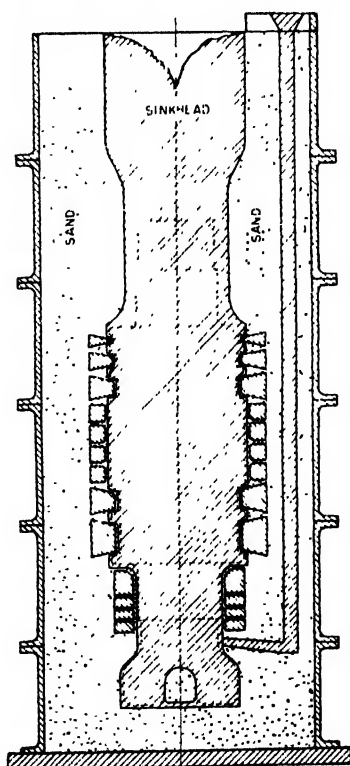


FIG. 23-7. Schematic section of a mold for an alloy-steel blooming-mill roll.

FIG. 23—8. Inserting chilling rings in a mold for a steel roll. (Courtesy, Pittsburgh Rolls Corporation.)



of the finished roll. The excess of metal at the top is either burned off before, or cut off during, the machining operation. The cavity in the sink-head is formed by feeding molten metal in the head to the roll body to compensate for shrinkage during solidification.

Figure 23—8 shows cooling or chilling rings being placed in a roll mold and Figure 23—9 shows the mold being smoothed over and near completion. Figure 23—10 shows some structural-mill rolls having the chills removed. In this type of mold, the chill rings located in the passes play their most important part by imparting a fine-grained structure to those parts of the roll which form the major wearing surface.

After the alloy-steel rolls are taken out of the molds,

by an operation called shaking out, they are ready for heat treatment. An example of this heat treatment is that used for a chromium-molybdenum alloy-steel roll for a blooming mill, containing 0.80 to 0.90 per cent carbon, 0.70 to 0.80 per cent manganese, 0.25 to 0.30 per cent silicon, 1.00 per cent chromium, 0.25 to 0.30 per cent molybdenum, and a maximum each of phosphorus and sulphur of 0.04 per cent. After shakeout, the roll is brought to the heat-treating furnace and placed in the furnace in a manner to allow full access of heat to all parts. It is brought up slowly to 1700° F and allowed to cool in the furnace to 1000° F, and then is reheated to 1550° F. At this point, the roll is removed from the furnace and air-cooled to 1000° F,

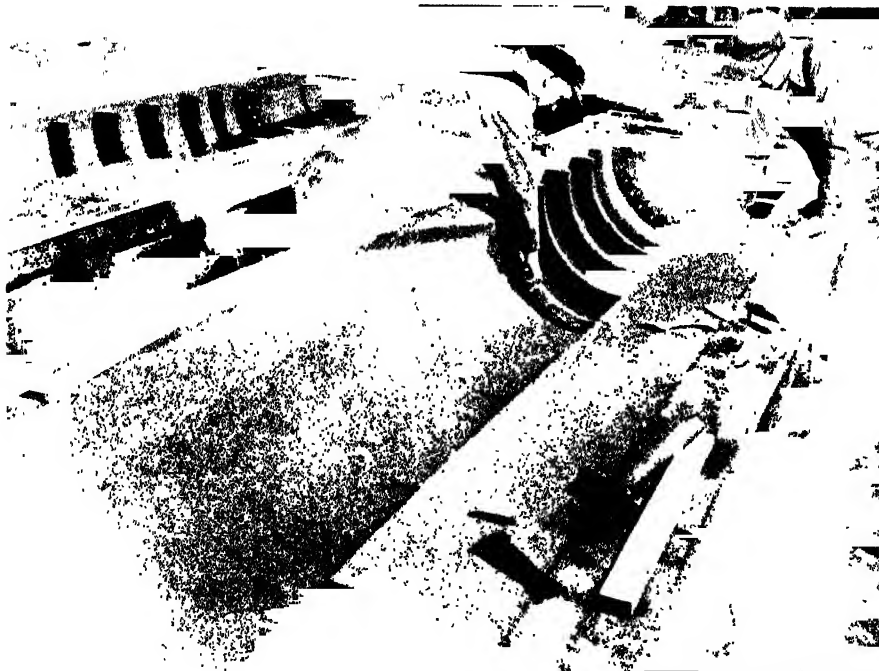
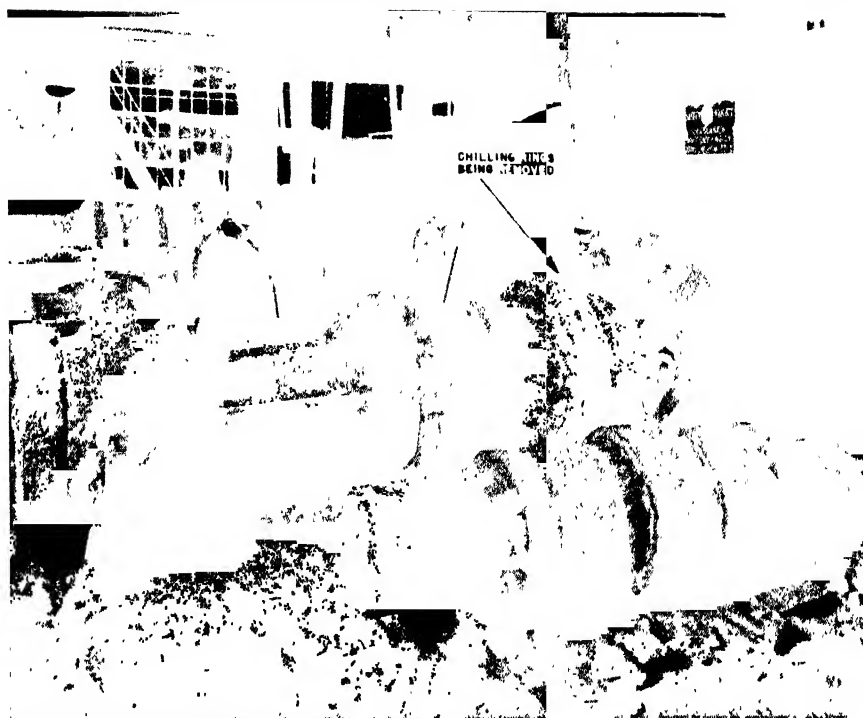


FIG. 23—9. Smoothing over the face of one-half of a roll mold, after insertion of chilling rings. (Courtesy, Pittsburgh Rolls Corporation.)

FIG. 23—10. Removing the chilling rings from steel-roll castings after they have been shaken out of the molds. (Courtesy, Pittsburgh Rolls Corporation.)



replaced in the furnace and brought back up to 1250° F, after which it is allowed to cool to 300° F and removed from the furnace. The heat treatment varies with the size and chemical composition of the roll.

After heat treatment, the roll goes to the roll lathe for removal of extra stock on the top neck and thorough testing of the roll body. Any defects which cannot be removed by the rough turning result in rejection and scrapping of the roll.

A special type of alloy-steel roll is the **built-up roll**. These are used for backing-up rolls on four-high plate and strip mills. The solid backing-up roll as used on large four-high mills is a difficult one to cast. The large volume of the casting induces strain during cooling due to the unequal cooling between the roll surface and its center. Due to this unequal cooling, cracks are very apt to occur in the casting. In the built-up roll, the center arbor or mandrel is relatively small in diameter and easier to cast. It also cools more uniformly than a large-diameter solid roll, and is less liable to strain and crack-

ing. The outer shell also is easy to manufacture and, due to its construction, can be heat treated to be much harder than the arbor. Figure 23—11 shows a built-up roll of special construction. In some cases, solid rolls which have been dressed down repeatedly in service are reduced further and fitted with sleeves and returned to service.

Another type of roll is the **forged-steel roll**. This roll is forged from an ingot of suitable size and, after being normalized in a heat-treating furnace, is turned close to the finished size. A hole is then bored longitudinally through the center of the roll to facilitate hardening by quenching after heating to the proper temperature. This type of roll is used mostly for the cold rolling of flat material. One of the favored compositions for a roll of this kind is: carbon, 0.85 per cent, manganese, 0.25 to 0.30 per cent; phosphorus and sulphur, below 0.05 per cent; silicon, 0.25 to 0.30 per cent; chromium, 1.60 to 2.50 per cent; molybdenum, 0.25 per cent; and a trace of vanadium.

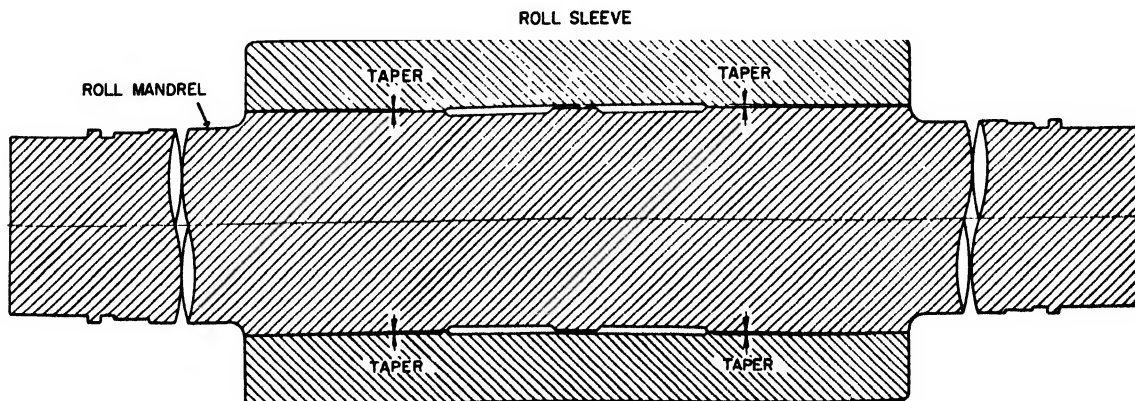


FIG. 23—11. Cross-section of a built-up roll.

IRON-BASE ROLLS

The iron-base roll differs from the steel-base roll principally in per cent of carbon, the steel roll containing from 0.30 to 2.50 per cent carbon, whereas iron rolls contain from 2.50 to 3.50 per cent. The material for iron rolls usually is melted in an air furnace using powdered coal as fuel. The flame is less oxidizing than that used in open-hearth operation and there is less loss of elements.

In the manufacture of iron rolls, a wide range of composition is employed to obtain the desired characteristics such as hardness, strength, resistance to spalling, and depth of dense, refined grain structure. Table 23—I summarizes the general range of elements for various types of rolls, and rolling-mill applications.

The normal charge for the melting furnace in the manufacture of iron rolls contains 25 per cent pig iron and 75 per cent worn-out rolls, heads and gate scrap. If lower carbon content is desired, a certain percentage of low-carbon-steel scrap makes up a part of the charge. Alloys, such as nickel, chromium and molybdenum, if required, are also added with the charge.

When the iron is melted and sufficiently hot, two samples are taken, one for chemical analysis and the other for a test block which is cooled and broken. From the chemical analysis and the test coupon, the roll metallurgist determines the necessary additions of alloys for the type of roll being cast. The metal is tapped in a ladle and, when the correct temperature is reached (2425° to 2650° F), poured into previously prepared roll molds. After cooling from one to four days, the rolls are shaken out, heat treated if necessary, and machined to specifications.

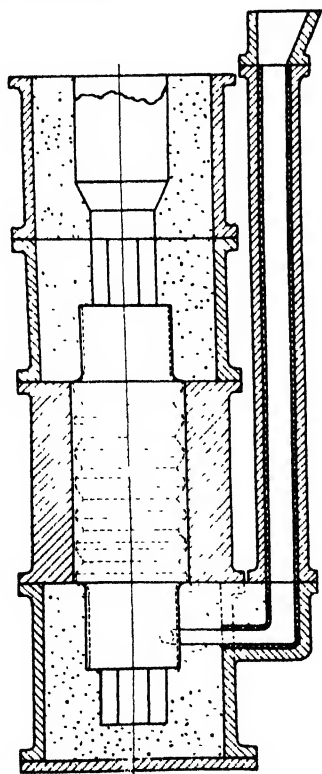


FIG. 23—12. Vertical section of mold for casting a plain-bodied roll, or a shape roll in which grooves are machined from the solid (dotted lines).

Chill Rolls—Figure 23—12 shows a roll-mold arrangement for casting a plain-bodied roll, or a shape roll in which the grooves are machined from the solid. The wobblers and necks are cast in sand but the body of the roll is formed by heavy-walled cast-iron cylinders, known as chills. The purpose of these chills is to cool the molten iron quickly after it has been poured into the mold, while the necks and wobblers, being molded in sand, are not subjected to fast cooling. The carbon in molten iron is in solution. If the molten iron is allowed to cool slowly, the carbon in excess of 0.82 per cent separates as graphitic carbon and is distributed throughout the mass. But if iron of the proper composition is cooled rapidly from the liquid state, the carbon combines with the iron to form cementite (Fe_3C), which is very hard and white in color. The depth of case on the body of the roll, induced by the fast cooling of the molten iron in contact with the chill mold, depends upon the chemical composition of the iron (see Chapter 18). As the mold loses its chilling effect, carbon starts to come out of solution to form a mottled area immediately below the hard case. This mottled area consists of a mixture of graphite, cementite and iron. Since the interior of the roll cools slowly, gray iron forms therein. Another factor controlling the depth of chill is the diameter of the roll being cast. As the diameter increases, the chilling rate is retarded due to greater cross-sectional area. On a 30-inch diameter roll, the following composition would give a clear chill about $\frac{3}{4}$ inch in depth, and a mottled area about $\frac{3}{4}$ -inch deep before reaching the portion composed uniformly of gray iron:

Element	Per Cent
Carbon	3.00
Silicon	0.65
Phosphorus	0.40
Manganese	0.25
Sulphur	0.08

To improve the finish of the mill product and reduce spalling and wear, various alloys are added such as: nickel, chromium, molybdenum and vanadium in balanced proportions to control chill depth and roll structure.

Since the case or chill on the rolls is hard and brittle, there is a practical limit to the depth a roll can be chilled without causing failure through breakage. In general, chill specifications will vary between $\frac{1}{2}$ and $1\frac{1}{2}$ inches in depth, depending upon the product to be rolled and the amount of diameter reduction through dressing expected to be permissible before the roll is scrapped.

Grain-Iron Rolls—For certain applications, such as the intermediate stands of some hot-strip, plate, rod and merchant mills, where shock, temperature and extremely heavy loads frequently result in spalling, fire-cracking and breakage, chill rolls are not practical. This applies also to rolls for billet, rail and structural mills, with deep grooves or passes. To meet these requirements, an iron composition commonly referred to as grain iron is used. Sufficient alloys, such as chromium and nickel, are added to control hardness and to increase strength while silicon or other graphitizers are used to resist formation of a definite chill. While these rolls do not give the quality of finish obtained from a true chilled-type roll, they have deeper penetration of the refined structure, with increased strength. The grain-type rolls become softer and the gray iron coarser progressing inward from surface to center, and this less desirable structure is encountered as the deeper

Table 23—I. Types, Compositions and Uses of Iron Rolls

Type	Carbon (%)	Manganese (%)	Silicon (%)	Phosphorus (%)	Sulphur (%)	Chromium (%)	Nickel (%)	Molybdenum (%)	Scleroscope Hardness	Uses
Plain Chill	2.90-3.30	0.20-0.25	0.60-0.70	0.45	0.08	0.25	58-70	Sheet, Merchant, Plate and Rod Mills.
Nickel Chill—Regular	2.90-3.30	0.18-0.25	0.40-0.60	0.35	0.08-0.12	0.25-0.50	2.50-3.00	0.25	65-70	Finishing; Bar, Rod, Skelp and Intermediate Stands of some Strip Mills.
Nickel Chill—Hard	3.30	0.18-0.25	0.40-0.50	0.35	0.08-0.12	0.75-1.00	3.50-4.50	0.25	80	Finishing Stand; Strip and Band Mills.
Sand Iron (Low-Alloy Grain)	2.50-2.80	0.40-0.50	0.80-1.25	0.15-0.25	0.08-0.12	0.50	0.50	0.25	45-55	Roughing and Intermediate; Merchant, Structural and Rail Mills.
Medium Alloy Grain	3.00-3.25	0.40-0.50	0.90-1.25	0.10-0.20	0.08	0.70-1.20	0.50-1.00	0.25	55-65	Intermediate, Finishing; Merchant, Bar, Rod, Structural and Pipe and Skelp Mills.
High-Alloy Grain (Regular)	3.15-3.40	0.40-0.50	0.80-1.10	0.10-0.15	0.06	1.50-1.75	4.00-4.50	0.25	65-75	Intermediate Stand; Hot Strip Mills.
High-Alloy Grain (Hard)	3.40	0.90-1.50	0.80-1.00	0.10-0.15	0.06	1.50-2.00	4.50-5.00	0.25	80-90	Finishing Stand; Strip Mills and Cold Reduction Mills.
Ductile Iron	3.20-3.40	0.30-0.50	1.50-2.25	0.04-0.15	0.01-0.02	0.75 max.	1.50-2.50	0.25-0.50	40-65	Roughing or Intermediate Stands; Rod, Structural and Bar Mills.

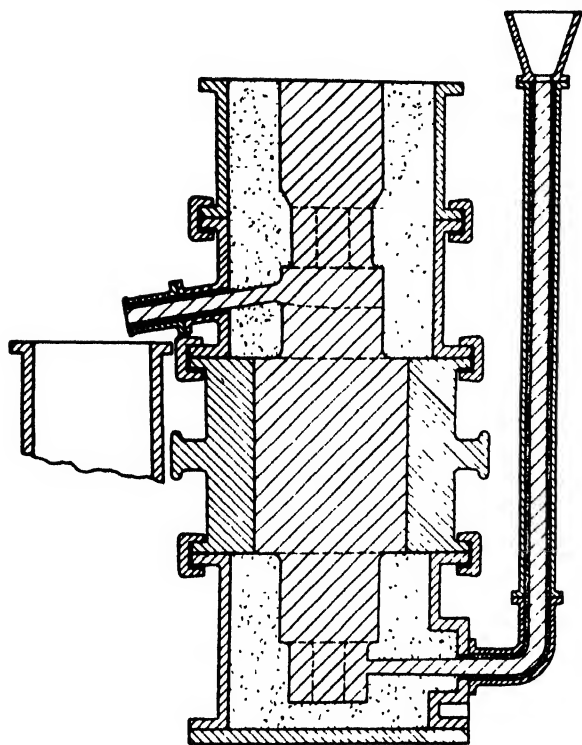


Fig. 23-13. Schematic representation of the vertical section of a mold for casting a composite iron roll.

passes are cut in a roll. To overcome this characteristic, rolls calling for deep working passes are molded similar to steel rolls, as shown in Figures 23-8, 23-9 and 23-10 in Section 1 on steel rolls. The inserted iron rings promote a finer structure and better wearing metal in the passes.

Composite or Overflowed Rolls—High-alloy rolls (1.50 to 2.50 per cent chromium and 4.00 to 5.00 per cent nickel with 75 to 90 Scleroscope hardness) used in cold-reduction mills and finishing stands of some hot-strip mills, would, if cast solid, have hard, unmachinable necks. Casting strains are quite severe in this type of

roll. To overcome these objectionable features, a different method of pouring has been developed as shown in Figure 23-13. The mold is made in the regular manner, except that a spout is connected to the top neck cavity a few inches above the body of the roll. Metal of high-alloy content is poured until the metal reaches the run-off spout. Pouring is stopped and a small amount of metal of a composition that produces much "softer" iron than the first high-alloy metal is poured down the runner to keep the ingate from freezing. After a predetermined time, pouring of the "soft" iron is continued, washing out through the spout connected to the top neck and into a "nugget pot" the still-molten hard roll metal that has not been chilled and solidified by the iron mold which forms the body of the roll. When sufficient "soft" iron has been introduced into the mold to wash out the still-fluid iron in the center of the roll, the pouring is stopped, and a plug inserted in the overflow spout. The same iron used to flush out the molten hard iron from the interior of the roll is introduced into the top neck either by pouring directly down through the neck or through a gate attached to the top neck just above the run-off spout.

A roll poured by this method will have an extremely hard fine-grained structure for a depth of 1 to 2 inches. The necks will be machinable and the central part of the roll will be strong, the hard brittle iron having been replaced or diluted by the soft "flush iron."

Ductile-Iron Rolls—With the recent development of "ductile" or "nodular" cast iron, some manufacturers have produced rolls of this material which is made by the addition of magnesium or rare earth compounds to iron of restricted analysis.

The remarkable strength and toughness of this iron results from the nodular shape of the free graphite in the structure as contrasted with the flake graphite common to gray iron. This iron, if properly made and heat treated, develops properties which approach the strength and ductility of steel.

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Chapter 24

MILL DRIVES AND POWER REQUIREMENTS

SECTION 1

INTERRELATION BETWEEN ELECTRIC-POWER SUPPLY AND THE DEVELOPMENT OF MAIN MILL DRIVES

During the first part of the nineteenth century, rolling mills were driven by water wheels and low-pressure steam engines. The power was transmitted to the mills by direct mechanical connection, through suitable shafts, gears and couplings. Because of the low steam pressures used, short steam lines were necessary. As a result, many small boiler houses were built adjacent to the mills. Most of the engines were of the simple non-condensing type and this factor, together with the small, hand-fired boilers used, resulted in high-cost operation. Coal had to be delivered to the boiler houses, ashes disposed of, and steam lines maintained. In some plants, compound condensing engines were used, requiring a supply of cooling water and a sewer system. When the mills were shut down over the weekend, the steam lines had to be kept hot, as the contraction set up by cooling the line resulted in leaks at the joints. Nearly all of these early mills have been replaced, during the last twenty years, with modern electrically-driven mills that, while fewer in number than the mills they replaced, have a much greater combined capacity.

With the development of the electric generator and motor in the latter part of the nineteenth century, a new and more efficient method of driving rolling mills became available. Instead of direct mechanical connection of the mills with the power source, generators could be placed in a convenient central location, and electric power could be transmitted over wires to motors attached to the mills. The generators could be driven either by gas engines, steam engines, or steam turbines.

Shortly after 1890, internal combustion engines were developed to operate on blast-furnace gas. This by-product fuel was available at all blast-furnace installations and at this period either was used for making steam or was wasted. The gas engine could use this fuel direct and thus eliminate the investment for a boiler house required for a steam engine. These gas engines were used as the prime movers for blast-furnace-blowing equipment and also for driving electric-power generators. This type of drive was selected for both the blowing units and the generators at the Gary Steel Works when this plant was designed in 1908. At that time, gas-engine-driven generators were available in sizes having outputs up to 2000 kw, and fifteen such units were installed. A few years later, the size was increased to 3000 kw, which equalled the largest generator driven by a reciprocating steam engine at that time. The gas-engine-driven generator installation at Gary was the first sizeable steel-mill power house in the United States, and made possible the use of 6000-horsepower motors to drive the mills. At a later date, the maximum size of the gas-engine-driven generators was increased to 6000 kw and three units of this

size were installed in the South Works, Chicago, Illinois.

While the development of the gas engine was progressing, another prime mover had entered the field—the steam turbine. Both the steam engine and the gas engine were slow-speed machines and, when used to drive electric generators, the speed limitation resulted in a physical limitation of the generators. The 5000-kw generators, which were about the largest built, had a diameter of over 30 feet. On the other hand, the steam turbine is inherently a high-speed machine and the early generators of 5000-kw capacity that were driven by steam turbines required only a fraction of the space needed by other types of units.

Developments in boiler practice whereby steam pressures could be increased to 250 pounds at first, then 500 pounds and then, by 1948, over 2000 pounds per square inch, all favored development of turbine designs and the size of turbine units increased until 275,000-kw generators now are available. While these developments

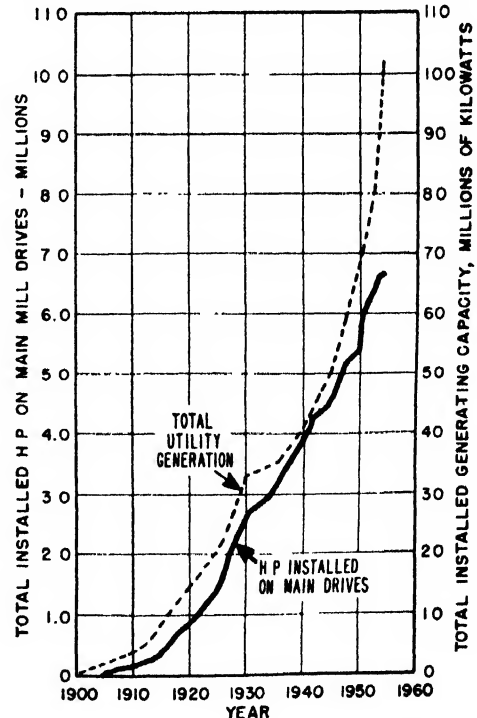


FIG. 24—1. Relationship between growth of electric-power generating capacity and installation of main mill drives (1904-1954). Note difference in vertical scale used in plotting the curves.

in electric-power-producing facilities were taking place, iron-silicon alloy steel in sheet form for transformer cores was being developed and improved and better insulating materials developed, with the result that larger quantities of electric power could be produced and, further, it could be transmitted economically over greater distances to the motors that used it.

Such improvements in both the generation and distribution of power were required before electrification of mills became possible. Figure 24—1 shows the relation between the growth of electric-power generation and the installation of main-drive motors in steel mills. The installed generating capacities were obtained from Edison Electric Institute statistics and show a growth from 160,000 kw in 1905 to 102,000,000 kw in 1954. The main drive figures were taken from the Proceedings of the Association of Iron and Steel Engineers, and show a growth from zero in 1905 to 6,700,000 horsepower in 1954.

Many steel plants, especially the smaller ones, are not self-contained in that they have no coke plant or blast furnaces. Such plants buy their electric-power supply from the power companies and, since they may impose large loads such as strip mills create, the power producer has the problem of providing sufficient gen-

erating and distribution capacity to serve such loads.

If a map showing the electric transmission lines in the United States is studied, it will be found that the various utility generating sources are interconnected by high-voltage lines to provide a pooled power supply. These interconnecting systems have many advantages, and are favorable to steel mills in that heavy peak loads caused by electric furnaces or strip mills do not necessarily have to be absorbed by a single generating station but, usually, are distributed among many units that are connected to the system. The growth of these interconnecting systems has followed the growth of electric-power consumption and today, with forecasted arrangements, it is possible to purchase power for all types of steel-mill electric loads.

The growth in generating capacity plus the interconnecting systems have made possible the large motor-driven installations in modern mills. Prior to about 1930, it often was a problem to obtain 6000 kw to drive a proposed blooming-mill reversing motor. Today, purchased-power contracts often cover the supply of power for loads of 100,000 kw, and additions to the supply are limited only by the time required to install additional generating capacity.

SECTION 2

DEVELOPMENT OF MAIN MILL DRIVES

In 1905, two 1500-horsepower, 230-volt, 100/125 r.p.m., direct-current motors were installed on the light-rail mill at the Edgar Thomson Works in Braddock, Pa. They were supplied with power from two 1000-kw, steam-engine-driven, direct-current generators in a nearby power house. In the same year, the first reversing direct-current main-drive motor was installed on the 36-inch universal-plate mill at South Works in Chicago. Power for this latter motor was furnished at 575 volts, direct current, from a 2200-volt, 25-cycle, motor-generator set. Both of these installations were, with some modification, still in operation in 1954.

At about the same time, another type of main drive, suggested by European developments, was installed on the light-rail mill at the South Works. This consisted of two 1500-horsepower, 2200-volt, 25-cycle, 80/120 r.p.m., variable-speed, wound-rotor motors. The rotor windings of these motors were connected to the stator of a second wound-rotor motor on the same shaft; this is known as a cascade connection. By varying the resistance in the rotor circuit of this second motor, the speed of the main drive could be varied between 80 and 120 r.p.m.

About the same time (1908-1910), the Gary Works, designed to be the largest steel plant in the world, planned to use slow-speed, wound-rotor induction motors on the heavy rolling mill drives. Some of these had windings that, by external contactors, could be connected to change the number of poles and thus provide what is known as a two-speed motor. Other

than this, the main drive installations were designed to operate at constant speed, except for the variations that could be obtained by changing the resistance in the secondary circuits of the wound-rotor motors.

This type of drive was satisfactory for some types of mills but, for the merchant mills in particular, greater ranges of speed variation were needed and this led to the development of variable-speed controls. With these systems, speed variations up to 50 per cent were made possible.

The demand for wider speed ranges, better regulation and a more simplified control led to the development of large, variable-speed, direct-current motors, which are now the preferred drives on practically all new installations.

While these developments were progressing, another problem confronted the steel industry—the alternating frequency of the power systems. In the period 1905-1910, practically all steel companies generated their own power and were not interconnected with utility systems. Slow-speed, direct-connected drives were needed, since high-efficiency gears such as now used were not available. For this purpose, 25-cycle motors offered an advantage and were adopted in the mills and, as late as 1910, all main drive motors in the steel industry had a 25-cycle source of power. Today, while there are many 25-cycle drives representing many thousands of horsepower still operating, new installations using 25-cycle power are very few and 60 cycles have become the standard frequency of the industry.

SECTION 3

POWER REQUIREMENTS IN THE STEEL INDUSTRY

According to a report of the American Iron and Steel Institute, the iron and steel industry used 40.7 billion kwh of electric power in 1953. Of this amount, about one quarter was generated in power houses in the industry and the remainder was purchased from utili-

ties. With an ingot production of 111,600,000 tons, the average power consumption per ton of ingots in 1953 was 364.8 kwh.

Power Requirements for Various Operations in the Production of Steel—When new mills are contemplated

in existing works or new plants planned, one of the questions to be answered is the quantity of electric power required to operate the new facilities.

Data on existing mills are the best source for this information. Actual power consumption by certain operations such as coke plants, blast furnaces and open hearths are found to be in accord with estimates based on established data from pre-existing similar units. Requirements of heavy mills such as blooming, slab and rail mills also can be estimated closely if the predicted output of product in tons per hour is matched by actual production. Finishing mills, however, require additional data as to the sections to be rolled and the pass design of the rolls.

In the total demand load of a plant, there are two loads that have the greatest effect on power peaks: those from the electric melting furnaces and the wide-strip mills. There are available data for power requirements of both of these operations, and the total power required for new installations has to be estimated by using these data in conjunction with proposed operating schedules.

The following tabulation gives the average kwh consumption per ton of steel for some of the major operating units in a modern plant.

Operating Unit	Kwh Consumed per Ton of Product
Coke Plant	17, based on coal charged
Blast Furnaces	10, based on pig iron
Open Hearth	8, based on ingots
Bessemer (with motor-driven blowing equipment)	92, based on ingots
Blooming Mill	13-15, based on blooms
Slabbing Mill	10-12, based on slabs
Plate Mill (Reversing) ..	30-40, based on plates
Merchant Mills	40-80, based on product
Wide Strip Mills	45-65, based on product

The preceding tabulation for mills covers the power used by the main drive motors, and does not include that used for auxiliaries such as tables, fans, lighting, and so on. On some mills, this auxiliary power amounts to as much per ton as the power used by the main drive.

The following tabulation indicates how power requirements are increased as the finishing operations are continued. These figures are averages taken from the records of a modern strip mill whose end products are tin plate and sheets:

Facility	Kwh per Ton
80-Inch Mill (including all auxiliaries) ..	63.0
Continuous Pickling	6.3
5-Stand Cold-Reduction Mill (including auxiliaries)	86.0-100.0
Electrolytic Cleaning	8.8

Tin Mill Annealing	20.0
Tin Mill Temper Rolling	18.0
Electrolytic Tinning	100.0
Hot-Dip Tinning	38.0
3-Stand Tandem Mill	35.0
Sheet-Mill Galvanizing	30.0

In addition to the above operations, such a plant has pumps, shops, air compressors, yard lighting and other miscellaneous loads that increase the total power consumption.

Estimation of the total electric-power requirements for a steel plant with the predicted fifteen-minute demand is part of a study required to get a fuel balance for new installations. (The predicted fifteen-minute demand is an estimated figure representing the highest possible peak load expected ever to be encountered for an arbitrarily-selected period of 15 minutes duration.) It is also necessary before arriving at an estimated amount to be used in negotiating purchased-power contracts. The tabulation following is an example of the estimation of the annual kwh requirements for an integrated plant planned to produce 1,500,000 tons of open-hearth ingots per year.

Mill or Department	Annual Production (Tons)	Kwh/Ton of Product	Total Kwh Per Year
Coke Plant	826,000	17	14,042,000
Blast Furnaces	1,000,000	10	10,000,000
Open Hearth	1,500,000	8	12,000,000
Blooming Mill*	1,235,000	26	32,110,000
Billet Mill*	458,000	28	12,824,000
Wheel Mill*	34,000	110	3,740,000
12-Inch Bar Mill*	221,000	65	14,365,000
16-Inch Bar Mill*	200,000	55	11,000,000
Structural and Rail Mill*	353,000	60	21,180,000
Plate Mill*	244,000	87	21,288,000
Splice Bar and Mine Tie. Miscellaneous (Pumps, Air Compressors, Lighting, Etc.; due load 7000 kw x 8760 hours)	37,000	70	2,590,000
Total			216,399,000

*Including auxiliaries.

The above represents an average consumption of 18,000,000 kwh per month or, in a 720-hour month, of 25,000 kw per hour. In this example, however, it is not the average power consumption in any hour but the peak hour that has to be determined, as this latter quantity determines the size of the generators required if a power house is to be built, or the peak demand that will be made on the utility system if power is to be purchased.

SECTION 4

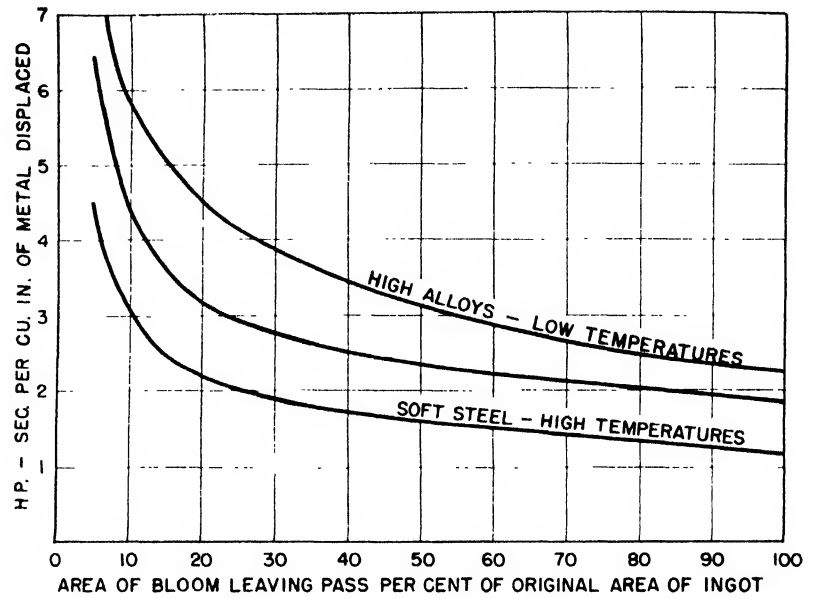
FACTORS WHICH AFFECT THE SIZE AND TYPE OF MAIN-DRIVE MOTORS

The selection of the proper motor to be used on mill drives is a very important item in the design of a mill. Its size and characteristics must be based not only on tonnage requirements and rolling schedules, but also on displacement, reductions, temperatures, composition of product, speed of rolling, and finish. The accepted methods for calculating loads when rolling a product at

a specific speed and definite temperature, with specified reductions per pass, are based on data taken from tests on existing mills.

The most common main-drive motor is that used on reversing primary mills. For these drives, the size and shape of the ingot are known, also the size of the finished bloom or slab, the composition and temperature of the

FIG. 24-2. Curves showing amount of work for displacement of one cubic inch of metal including all increases in friction during the pass in a single-motor-driven reversing blooming mill. (13, 34)



steel, the roll size, and the efficiency of the machinery. It has been determined from existing reversing primary-mill units that the total power required is divided approximately into the following components:

Power Required for Rolling	59.00%
Friction of Pinions and Mill	5.90
Loss in Reversing Motor	10.84
Loss in Electrical Connections	0.75
Loss in Generators	11.44
Loss in Flywheel	1.32
Loss in Slip Regulator	2.69
Loss in Induction Motor	6.40
Loss in Exciters, Blowers, Etc.....	1.66

Total 100.00%

The part of the total torque required of the motor shaft of a single-motor drive for a reversing blooming mill that is used in (1) deforming the metal being rolled; (2) overcoming the increase in roll-neck bearing friction during rolling; and (3) other mechanical losses, may be approximated from the following formula (see References 13 and 34 on Page 462):

$$(1) \quad T = 275 \times C (A_1 - A_2) \times D \frac{A_2}{A_1}$$

where T = Motor torque in pounds-feet
 C = Factor for cubic inches of metal displaced (ordinate of Figure 24-2).
 A_1 = Area of section in square inches before pass
 A_2 = Area of section in square inches after pass
 D = Diameter of roll at base of pass in inches.

From this formula, which applies only to the type of mill mentioned, the rolling schedule can be calculated pass by pass, and to the figures so obtained must be added the idling friction torque for the mill which may amount to 20,000 to 25,000 pounds-feet.

To obtain the maximum torque requirements for the motor, it is necessary to add the torque required for acceleration. The inertia of the mill parts and the armature for a 7000-horsepower motor is about 4,250,000 lb.-ft.². If the rate of acceleration is 20 r.p.m. per second, and the time required for acceleration is 2 seconds, then from the formula:

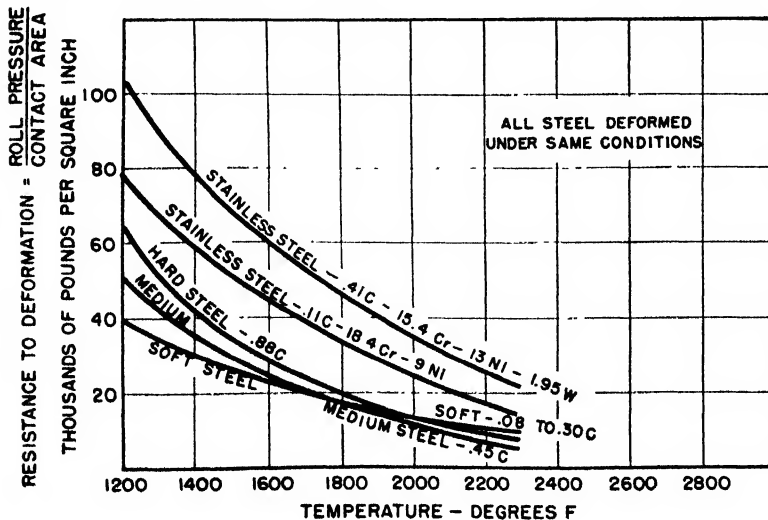


FIG. 24-3. Effect of temperature of steel on resistance to deformation. (After Trinks.)

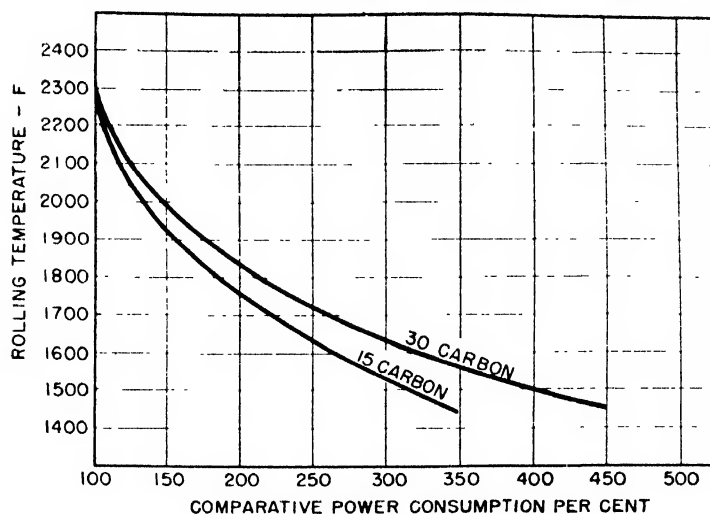


FIG. 24-4. Effect of temperature on power consumption during rolling steels containing 15 and 30 points (0.15 and 0.30%, respectively) of carbon. (Courtesy, Iron and Steel Engineer.)

$$(2) \quad T_a = \frac{0.003255 \times WR^2 \times \text{r.p.m.}}{t}$$

where

T_a = Torque required for acceleration (pounds-feet)

t = Time required for acceleration (seconds)

WR^2 = Inertia of moving parts,

$$T_a = \frac{0.003255 \times 4,250,000 \times 20}{2} = 138,000 \text{ pounds-feet}$$

Another factor to be considered is selection of a motor of ample capacity to prevent excessive heating of the motor during operation.⁽¹³⁾

Figure 24-3 shows a group of curves for five different steels in which the resistance to deformation is plotted against temperature of the steel in degrees F. Figure 24-4 shows similar curves for two grades of carbon steel, where rolling temperatures are plotted in relation to power consumption. From these curves it may be seen that a drop in temperature from 2300° F to 1750° F will double the power requirements for rolling. This indicates the necessity for considering steel temperature when calculations of power for rolling are being made.

Another factor to be considered is the number of horsepower-seconds required per cubic inch of metal displaced during the various passes. Temperature again

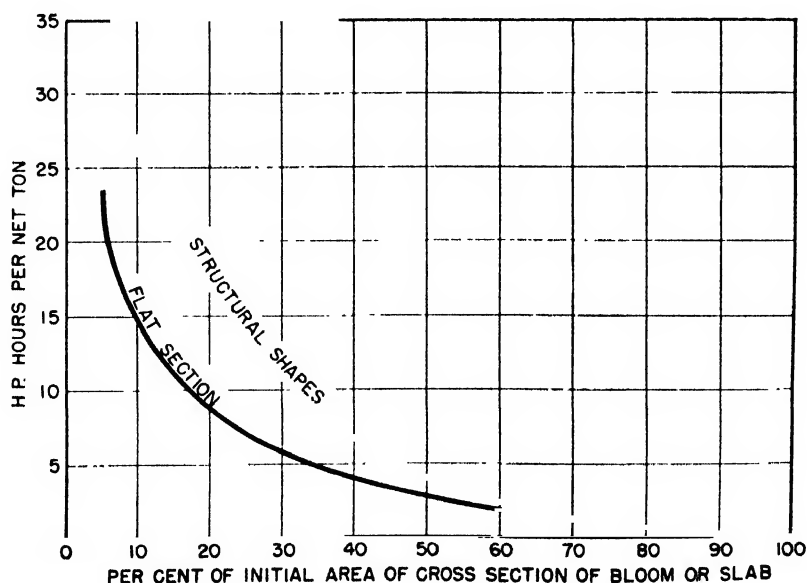


FIG. 24-5. Curves showing relation of horsepower-hours per net ton of steel rolled to per cent reduction of cross-section for shapes and flat sections.

must be considered in this case as the bloom or slab cools off during the rolling operation. Figure 24-2 shows curves giving the specific power for motor torque for displacement of metal plus all increases in friction during the pass. From these curves, it will be noted that as the bloom becomes smaller the horsepower-seconds increase rapidly, especially after 25 per cent of the original area is reached. The constant "C" used in formula (1) was taken from this chart.

Another form of these power curves is shown in Figure 24-5. These curves, for flat and structural sections, show that power requirements do not increase in direct proportion to per cent reductions, and that the smaller the section the larger the increase in power in terms of reduction.

The speed at which the piece is rolled affects the

power requirements in two ways. Theoretically, the power requirements should be directly in proportion to speed; however, with increase in speed the temperature of the piece being rolled has less time in which to drop; therefore, the higher the rolling speed, the higher the finishing temperature and, consequently, the lower the power required by the mill. For this reason, it is sometimes possible to increase the speed and not increase the motor loading to any great extent.

In order to meet the surface-quality requirements on certain products, it is necessary to roll them in the finishing pass at relatively low temperatures—sometimes as low as 1400° F. On mills where this practice is to be used, the horsepower requirements for the motor should be determined on the basis of the reduction and temperature required for such sections.

SECTION 5

TYPES OF MOTORS FOR MAIN DRIVES

There are four general classifications of motors that can be used on mill drives: synchronous, squirrel-cage, wound-rotor, and direct-current. All but the last-named operate on alternating current. Each of these has characteristics which make it suitable for definite applications. The cost of each type also differs and is often the factor determining the type to be used.

The synchronous motor, if an exciter is not required, usually costs the least and, in certain speed ranges, may be used interchangeably with the squirrel-cage design. Wound-rotor motors cost more than either the synchronous or squirrel-cage, and direct-current motors are the most expensive of all. With the latter, a supply of direct current is needed, which necessitates the use of a motor-generator set or mercury-arc rectifier to convert the alternating-current power supply of the plant to direct current, and these have to be considered as a part of the installation.

In the following paragraphs, short descriptions of the four general classifications of motors are given, along with suggestions for possible applications. It is assumed that the reader is familiar with the general design of such units; if not, details of their design and construction are available in the literature.

Synchronous Motors—The synchronous motor has the stator or stationary part wound with core and coils in the same manner as the squirrel-cage and wound-rotor motors. However, the rotating part or rotor has salient poles, positive and negative, wound for direct current. The power supply for these poles usually is furnished by a small direct-current generator called the exciter. This may be mounted on the main motor shaft, be belt driven by same, or be driven by a separate motor in a motor-generator set. In addition to the direct-current coils on the pole pieces, another winding which is buried in the pole faces is required. This is called the squirrel-cage winding and controls the necessary starting and pull-in torques of the motor and also the uniformity of the accelerating torques between starting and pull-in. When driving reciprocating machinery, the squirrel-cage winding also develops a damping torque.

This winding consists of a number of round or rectangular rods or bars that pass through the pole shoes in the axial direction. The material used may be copper, brass, or bronze, of such electrical conductivity as to suit best the particular requirements of starting and pull-in torques. The ends of these rods or bars project sufficiently beyond the poles to allow making connections to the short-circuiting end-ring segments to which

they are joined securely by brazing. In some cases, where both high starting torque and high pull-in torque are required, it is necessary to use double cage windings. These consist usually of two sets of damper bars but only one set of end connections.

To conduct the direct current to the field coils, collector rings mounted on the main shaft are required, the terminals for the field circuit being connected to these rings and the direct-current power supply from the exciter is connected to the brushes that contact the rings.

In order to apply a synchronous motor correctly, it is necessary to give consideration to the starting torque, pull-in torque, pull-out torque, operating temperature, power factor and method of starting.

The time required for a synchronous motor to attain full speed depends upon the load, and the power required to pull into step from this speed depends upon the inertia of the revolving parts, so that the pull-in torque cannot be determined without knowing the inertia of the external load as well as the torque required to overcome it.

The permissible starting-power input will vary with different localities, being affected by local power company regulations, capacity of feeders, or the capacities of individual, isolated power stations.

These motors can be designed with considerable range of starting, pull-in and pull-out torques. Some applications require a starting torque of 150 to 200 per cent, a pull-in torque of 110 per cent, and a pull-out torque of 175 per cent.

The efficiency of a synchronous motor usually is higher than that of a comparable induction motor, is less expensive in first cost, and has the ability to correct power factor. They have been built in sizes up to 10,000 horsepower for mill drives. It must be remembered that they are a constant-speed motor and are locked in step with the frequency of the power system. For this reason, there is no cushion afforded by a drop in speed for a sudden peak load, and the pull-out torque must be high enough to handle the highest peak load encountered. Synchronous motors often are used to drive motor-generator sets, air compressors, pumps and other types of equipment where constant load conditions exist.

Squirrel-Cage Motors—Squirrel-cage induction motors are the most simple with respect to design, and can be built in a wide range of sizes and torques. By varying the design resistance in the rotor winding, the

Table 24-I. Main Drive Motors Installed 1939-1954.

Year Installed	Alternating-Current Motors		Direct-Current Motors	
	Number Installed	Total Horsepower	Number Installed	Total Horsepower
1939	9	18,900	54	81,500
1940	13	6,850	72	112,000
1941	31	39,350	93	101,100
1942	14	20,000	97	263,650
1943	7	10,000	5	10,750
1944	14	14,200	17	34,100
1945	15	12,700	123	155,700
1946	16	41,650	191	286,650
1947	20	31,850	125	201,325
1948	11	29,900	93	107,900
1949	11	33,500	78	160,100
1950	32	77,750	236	451,450
1951	30	34,950	174	304,250
1952	8	7,200	41	64,600
1953	8	28,700	67	92,550
1954	13	16,100	106	159,605
Totals	252	422,800	1572	2,582,230
	(13.82%)	(14.05%)	(86.18%)	(85.95%)

starting torque can be low, normal, or high, as desired. It has become customary to start these motors "across the line" in the usual sizes, which necessitates strong bracing for the stator coils to withstand the powerful magnetic forces involved.

In the larger sizes, these motors have been designed for use in certain speed ranges in capacities up to 10,000 horsepower, and for speeds as low as 100 r.p.m., operating on 60-cycle power.

This motor has the lowest initial cost per installation, but its efficiency is less than that of the synchronous induction motor, and at the lower speeds the power factor is lowered. For instance, a 500-horsepower, 60-cycle, squirrel-cage motor has the following efficiencies and power factors at 900 compared with 514 r.p.m.:

R.p.m.	Efficiency (%)		
	½ Load	¾ Load	Full Load
	900 514	91.1 91.3	92.9 92.3
	Power Factor		
	½ Load	¾ Load	Full Load
	900 514	82.0 75.0	88.0 83.0
	½ Load	¾ Load	Full Load
	900 514	82.0 75.0	88.0 83.0

This type of motor is used on fans, pumps, compressors and shears, to name a few common applications, but is not used normally on main drives.

Wound-Rotor Induction Motors—This type of induction motor has a wide range of use in the steel industry. The difference between it and the squirrel-cage type is that the rotor winding consists of coils with the phase connections brought out to collecting rings mounted on the motor shaft. Connections can be made from these rings through external resistance boxes or a slip regulator; hence, the secondary resistance can be varied. By this means, the starting-power input and accelerating time can be controlled and, if the motor is used in connection with a flywheel, various divisions of the load between the motor and the wheel can be obtained.

These motors are available over the complete output and speed ranges required for mill drives, for operation on voltages up to 13,200. As in the case of squirrel-cage

Table 24-II. Total Horsepower of 25- and 60-Cycle Motors Installed on Main Mill Drives

Year	25-Cycle Motors		60-Cycle Motors	
	Horsepower	Per Cent of Total A.C. Units Installed	Horsepower	Per Cent of Total A.C. Units Installed
1905	4,900	100	0	0
1906-1910 ..	110,060	97	1,950	2
1911-1915 ..	99,675	57	72,610	42
1916-1920 ..	179,540	38	282,265	60
1921-1924 ..	67,500	34	131,740	65
1939-1945 ..	9,300	6	141,700	94
1946-1952 ..	4,500	2	245,100	98

Table 24-III. Synchronous Speeds of 25-Cycle and 60-Cycle Motors

Number of Poles	Speed	
	25-Cycle (r.p.m.)	60-Cycle (r.p.m.)
2	1500	3600
4	750	1800
6	500	1200
8	375	900
10	300	720
12	250	600
14	214	514
16	187½	450
18	166⅔	400
20	150	360
22	136	327
24	125	300
26	115	277
28	107	256
30	100	240
32	93.75	225
34	88.23	212
36	83⅓	200
40	75	180

motors, the efficiency and power factor decrease with the base speeds.

Wound-rotor induction motors are standard for flywheel-equipped motor-generator sets, and usually have been designed for a speed of 360 r.p.m. in medium sizes and 300 r.p.m. for the largest sets. However, several recent installations have been designed for 514 r.p.m. One such unit consists of an 8000-hp motor with a 65-ton, 215,000 hp-sec. flywheel, driving four 2500-kw and two 1750-kw direct-current generators.

Direct-Current Motors—Due to the demand for wide speed ranges for mill drives, many large direct-current motors are being used for this purpose. They are designed with flat speed characteristics, to operate usually on 600 volts. The largest size, about 8000 horsepower, has been used on reversing mills. These motors require forced ventilation due to the heat generated by operating conditions involving frequent reversals. The normal speed range is 2:1, but higher ranges can be obtained. On mill drives, it is not uncommon to have a maximum torque of 240 per cent of the full-load torque.

The trend toward the use of variable-speed direct-current motors on main drives began in the 1930's. The operating advantages of this type of drive overbalanced the added cost. Table 24—I, taken from the Proceedings of the Association of Iron and Steel Engi-

neers, shows this trend in motors (over 300 horsepower) installed on main drives from 1939-1954.

Table 24—II shows the trend from 25-cycle to 60-cycle motors.

The speeds of synchronous and induction motors are a function of the number of poles in the motor windings. These speeds are determined by the formula:

$$N = \frac{60 \times F}{\frac{1}{2} P}$$

where,

N = Speed (r.p.m.)

F = Frequency

P = Number of poles

Table 24 -III shows the possible speeds that can be obtained in both 25-cycle and 60-cycle motors. These are synchronous speeds and apply to synchronous motors. For induction motors, the full-load speeds are a few per cent less due to slip.

Although it is possible to build motors with more than 40 poles, they become very expensive, and the above table shows how it was advantageous to build the slow-speed, 25-cycle motors used in some of the early mills and why it would have been necessary to use gear sets if 60-cycle frequency had been specified.

SECTION 6

FLYWHEELS—HOW THEY WORK AND WHERE THEY ARE USED

Many main-drive applications are on mills that have short-time peak rolling loads. Blooming mills, slabbing mills, and reversing plate mills are in this classification. For such mills, it is often economical to use a wound-rotor induction motor having an adjustable secondary resistance in conjunction with a flywheel, to drive the generators that supply direct current to the main mill motors that are subject to peak loads. By this means, it is possible to select a motor having a capacity $\frac{1}{4}$ to $\frac{1}{2}$ of the value of the maximum load requirements of the mill, and so control its "slip" that the flywheel energy supplies the greater proportion of the energy to meet the peak loads. Some typical questions related to motors and flywheels of mill drives, and the methods of finding their answers, are given in the following examples.

1. Energy Stored in a Flywheel—Assume a large flywheel having a momentum (WR^2) = 15,000,000 lb.-ft.² (that is, an effect of 15,000,000 lb. at a radius of gyration of 1 ft., or 3,750,000 lb. at a radius of 2 ft.), and a wheel speed of 83 r.p.m. What is the total amount of energy stored in the wheel?

The energy (E) possessed by any body of weight W lb. moving with a velocity of V ft. per second is:

$$(1) E = \frac{W \times V^2}{32.16 \times 2} \text{ ft.-lb.}$$

In this expression, 32.16 ft. per second per second is the rate of acceleration due to gravity, usually expressed by the symbol, g. The rotating flywheel with a given WR^2 and running at N r.p.m. may be considered as a body with weight W moving with the same velocity as the end of its radius of gyration R; this velocity in feet per second for the case under consideration is:

$$(2) V = \frac{2 \times 3.1416 \times R \times N}{60}$$

Substituting this value of V in (1), it becomes:

$$(3) E = \frac{WR^2 \times N^2}{5865} \text{ ft.-lb.}$$

Then, for the present example,

$$(4) E = \frac{15,000,000 \times (83)^2}{5865} = 17,500,000 \text{ ft.-lb.}$$

This energy is equal to the amount of work required to bring the flywheel from rest to a speed of 83 r.p.m.

Since 1 horsepower-second = 550 ft.-lb.

E = 31,900 horsepower-seconds.

From the foregoing formulas, it may be seen that the stored energy in a flywheel is proportional to the WR^2 of the wheel and to the second power of its speed.

2. Amount of Energy Available for Regulation—The energy calculated above is the total energy stored in the wheel at a certain speed. How much of this energy may be used for load equalization in case this flywheel is installed on a mill drive?

If it is assumed that a speed variation of more than 15 per cent is not permissible, the wheel speed could be varied from 83 r.p.m. to 70.5 r.p.m. At 70.5 r.p.m., the stored energy in the wheel would be:

$$E_2 = \frac{15,000,000 \times (70.5)^2}{3,230,000} = 22,950 \text{ horsepower-seconds.}$$

Thus, the energy given up by the wheel in slowing down from 83 to 70.5 r.p.m. would be:

31,900 — 22,950 = 8,950 horsepower-seconds, or approximately 28 per cent of the total stored energy.

From this it is evident that the per cent of energy given up by the wheel is not directly proportional to the per cent of speed reduction.

Suppose the speed of the flywheel is lowered from N_1 to N_2 r.p.m. The speed reduction (S) expressed as a fraction of the maximum operating speed (N_1) is:

$$S = \frac{N_1 - N_2}{N_1}$$

or, the minimum speed is:

$$N_2 = N_1 \times (1 - S)$$

Now, the total amount of energy stored in the wheel at speeds N_1 and N_2 , respectively, are:

$$E_1 = \frac{WR^2 \times N_1^2}{3,226,000} \text{ horsepower-seconds}$$

$$E_2 = \frac{WR^2 \times N_2^2}{2,226,000} \text{ horsepower-seconds}$$

Hence, the energy given out comprises the following portion of the total:

$$\frac{E_1 - E_2}{E_1} = \frac{N_1^2 - N_2^2}{N_1^2} = \frac{N_1 - N_2}{N_1} \times \frac{N_1 + N_2}{N_1}$$

$$\frac{E_1 - E_2}{E_1} = S \times \left[\frac{N_1 + N_1(1 - S)}{N_1} \right]$$

$$= S \times (2 - S) = 2S - S^2$$

Giving S various values from 0 to 100 per cent, the corresponding values of energy given up by the wheel in slowing down may be calculated. For instance, if the speed reduction is 12 per cent ($S = 0.12$), the flywheel will give up 22.5 per cent of the total energy available at the maximum speed. Further calculations would show that at 50 per cent speed reduction, 75 per cent of the total energy is given up by the wheel.

3. Acceleration and Retardation of the Wheel—Assume that the wheel should be accelerated at a uniform rate from N_2 r.p.m. to N_1 r.p.m. in t seconds; then, the rate of acceleration (a) would be:

$$a = \frac{N_1 - N_2}{t}$$

If the inertia of the wheel is WR^2 lb.-ft.², then there will be required during this period a uniform accelerating torque (T_0), equal to:

$$T_0 = \frac{WR^2 \times a}{308} \text{ ft.-lb.}$$

For instance, if the same wheel as was considered above ($WR^2 = 15,000,000$ lb.-ft.²) is to be brought from rest ($N_2 = 0$) to $N_1 = 83$ r.p.m. in 10 seconds, then the required torque, T_0 , equals:

$$T_0 = \frac{15,000,000}{308} \times \frac{83}{10} = 405,000 \text{ ft.-lb.}$$

If the flywheel is brought up to this speed by an 83-r.p.m. motor, the latter will have to develop during the acceleration a torque (in addition to frictional torque) which corresponds to a load of:

$$\frac{405,000 \times 83}{5250} = 6440 \text{ horsepower}$$

4. Induction-Motor Characteristics—The majority of all mills equipped with flywheels are driven by induction motors. Within the operating range, the speed-torque or the slip-torque curves for these motors are very nearly straight lines. This is the basis for the assumption that the motor torque is proportional to the slip. If the secondary resistance of the motor is changed, the slope of the speed-torque curve is changed, but the breaking-down torque for the motor is not affected.

The torque developed at the motor shaft is proportional to the speed, and the difference in power between the input power and this torque goes into losses in the rotor windings, the core, and external secondary resistance.

The total power, P , can be expressed as:

$$P = \frac{T \times N_s}{5250}$$

where N_s is the synchronous speed and T the torque.

Starting from rest and drawing normal current from the line, the motor at first does not develop any power at the shaft because the speed is zero, but the secondary losses are high. With increasing speed, the losses are reduced and the mechanical power increases.

If the motor is running and more resistance is introduced in the secondary, maintaining constant line input, it will be observed that the speed, N , and the mechanical shaft power, P , both decrease in the same proportion, which means that the torque, T , is not changed.

This is an important conclusion, since it shows that when an induction motor drives a mill without any flywheel, and is called upon to develop a certain torque, nothing is gained by increasing the secondary resistance as long as the torque requirements are the same.

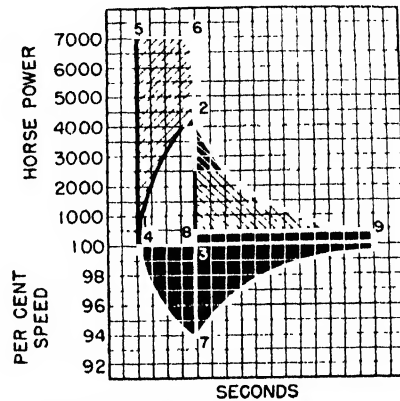


FIG. 24-6. Effect of flywheel energy on load and speed, showing division of load between motor and flywheel. Each horizontal division equals 1 sec. (Courtesy, Iron and Steel Engineer.)

5. Motor Load Curves—Assume a mill with a flywheel being driven by an induction motor of P horsepower, at a speed of N_s r.p.m. (synchronous speed). The mill is running light and the motor is developing only frictional torque, T_0 . Then the metal enters the rolls, and this imposes a combined torque, T_1 . The torque developed by the motor will increase gradually along some curve (Figure 24-6). What will the motor torque be after " t " seconds?

According to an expression called Gasche's formula:

$$T = T_1 - \frac{T_1 - T_0}{e^{At}}$$

where

T = motor torque to be determined

T_1 = combined (external) torque

T_0 = frictional torque

e = 2.718, the base for Napierian or natural logarithms

A = a constant for a given motor and flywheel (provided the secondary resistance is constant), and

equals $\frac{308 T_s}{WR^2 N_s S_n}$, in which latter expression:

T_s = normal torque

S_n = normal slip

N_s = normal speed

W = weight of the flywheel, and

R = radius of gyration of the wheel.

Figure 24-6 shows curves of the division of load between a motor and flywheel, calculated according to the

Electronic Control Elements are used in steel-mill regulating systems when the characteristics of the application are favorable to this type of control. Electronic control is used where great accuracy and speed of response are needed. It is also used when the input signal from the drive must be controlled from a very low power level. On very small drives, electronic equipment may be considerably less expensive, and sometimes there is no other way to solve the problem. In several recent installations, electronic control has been used as an addition to rotating-regulator or magnetic-amplifier equipment. A recent development in electronic controls is the transistor, a tiny simple device that employs the unique electrical properties of germanium crystals for rectifying and controlling the flow of current and that promises someday to replace the electron tube.

Rotating regulators have a broad field of application. The main and auxiliary drives of blooming mills are good examples of applications requiring a large number of rotating regulators. They are used at the Fairless Works, for example, for regulating the drives for the skin-pass or temper mill, a cleaning line, the finishing stands of the hot-strip mill, the main drive and auxiliaries for the 45-inch slabbing mill, and the main drive for the 40-inch blooming mill. On the main drive of a

blooming mill, rotating regulators are used to regulate and reverse the generator voltage, force fast response of the motor fields, limit the main armature currents to predetermined values, and for load-balance control where two or more armatures are operated in parallel.⁽²⁰⁾

The magnetic-amplifier regulating systems show promise of being applied in considerable numbers in the steel industry. Operating experience is somewhat limited since this type of control has begun to be applied only relatively recently. It has been used for controlling tandem cold-reduction mills, reel drives for cold-reduction mills, speed control for high-speed side-trimming line tension reels, speed regulation for the bridle rolls on cleaning lines, tension-reel drives for processing lines, tensiometer-type tension regulators for skin-pass or temper mills and as a voltage regulator for ignition rectifiers.⁽²⁰⁾

Electronic regulators have been used for applications where the regulator has to "see," such as edge control for slitting and cleaning lines, and where very precise and fast-acting regulator components are required, such as rod-mill drop regulators and electric flying-shear regulators. To obtain the required speed regulation, such as required on tube and rod mills, combination electronic-rotating regulating equipment has been used.

SECTION 8

REVERSING-MILL DRIVES

On motor-driven primary mills, such as slabbing, blooming, or single-stand plate mills, it is customary to use a reversing mill drive. About 135 mills of this type are in operation in the United States as this is written, and about 30 per cent of them still are driven by steam engines. These latter are the older mills which retain their engines for economic reasons. Since the installation of the first motor-driven reversing mill in 1915, very few engine drives have been installed and, today, only

motor drives are given consideration. Electric-motor and steam-engine drives for primary mills are compared in some detail in Chapter 25.

For modern reversing-mill service, the main-drive DC motor is of rugged design, with a welded frame of rolled steel, horizontally split. The pedestal for the bearing next to the mill is cast steel and is provided with a thrust bearing. These motors, being connected directly to the mill pinion stand, necessarily must be of slow

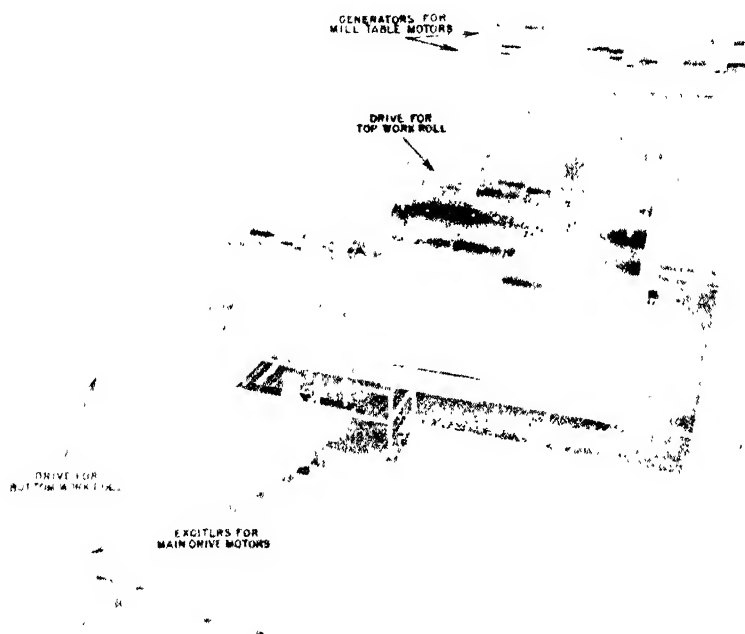


FIG. 24-8. Twin-motor drive for a 44-inch slabbing mill (Edgar Thomson Works) consisting of two 5000-horsepower, 40/80 r.p.m., direct-current motors.

Table 24—IV. Typical Modern Installations of Reversing Mill Motors*

Company	Location	Size Mill	Motor Horse-power	Motor Speed (r.p.m.)	Motor Voltage Rating
Armco Steel Corp.....	Butler, Pa. Middletown, Ohio	48"	7000	50-120	700
Bethlehem Steel Company	Steelton, Pa. Sparrows Point, Md. Sparrows Point, Md. Lackawanna, N. Y.	44" 54" 40" 54"	6000 7000 7000 7000	40-120 50-100 50-100 40-80	600 650 650 750
Great Lakes Steel Corporation	Ecorse, Mich.	46"	7000	40-100	700
Inland Steel Company.....	Indiana Harbor, Ind.	46"	7000	50-120	700
Jones & Laughlin Steel Corp. ...	Pittsburgh, Pa.	44"	7000	50-120	750
Republic Steel Corporation	Warren, Ohio Canton, Ohio Gadsden, Ala.	36" 35" 40"	6500 5000 7000	50-120 40-120 50-120	600 600 700
United States Steel Corp.	Braddock, Pa. Homestead, Pa. Homestead, Pa. Gary, Ind. Gary, Ind. South Chicago, Ill. South Chicago, Ill. Fairfield, Ala. Morrisville, Pa.	44" 44" 45" 44" 40" 54" 40" 46" 45"	2-5000 7000 2-5000 8000 6000 2-5000 7000 7000 2-6000	40-80 50-120 40-80 40-100 40-90 40-80 40-120 50-100 40-80	700 700 700 800 600 700 700 700 700
Wheeling Steel Corp.....	Steubenville, Ohio	45"	6000	50-120	700
Youngstown Sheet & Tube Co.	Indiana Harbor, Ind. Indiana Harbor, Ind.	46" 45"	8000 2-5000	50-120 40-80	700 750

*Courtesy of "Iron and Steel Engineer."

speed. One of the normal speed ratings is 0/40/80, which represents speed control from zero to 40 r.p.m., which is the constant-torque range, on voltage control, and from 40 to 80 r.p.m., which is the constant-horsepower range, on field control.

One of the original design problems on these motors was commutation and, in sizes above 4000 horsepower, they were built with two armatures on a single shaft. In recent years, however, the designs have been developed so that an 8000-horsepower, 0/50/100 motor has been built with a single armature. Sizes and speeds of some modern installations of these motors are shown in Table 24—IV.

This table indicates that the most popular size is about 7000 horsepower. Some of the characteristics of such a motor are:

Frame Diameter.....	197 inches
Armature Diameter.....	148 inches

Length of Shaft.....	295 inches
Diameter of Shaft.....	32 inches
Weight of Complete Motor.....	416,000 pounds
Approximate Reversing	

Cycle.....[50-0-50 r.p.m.—2 seconds
[120-0-120 r.p.m.—5 seconds

A motor of this type was installed on the reversing roughing stand of a 132-inch semi-continuous plate mill at Columbia-Geneva Steel Division of United States Steel Corporation. This 7000-horsepower, 0/25/60 r.p.m. motor is really two 3500-horsepower motors mounted as a single unit. The armatures are of unusual construction in that they have no conventional shaft. The spiders are bolted to stub flanged shafts at each end and the complete unit assembled at the mill. This design solved some of the shipping problems for the heavy pieces.

When the power requirements for a reversing mill exceed 7000 horsepower, problems in motor and pinion

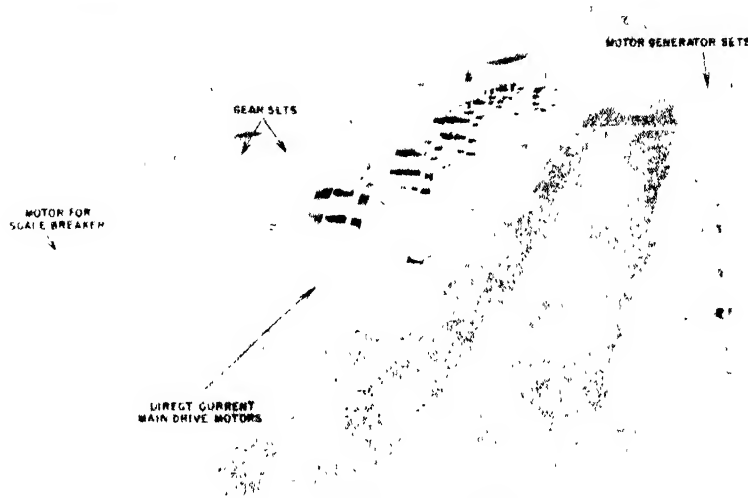


FIG. 24—10. Interior partial view of a motor room showing the drive motors and some auxiliaries for the finishing stands of an 80-inch hot-strip mill (Irvin Works).

SECTION 10

CONTINUOUS-MILL DRIVES

Wide Hot-Strip Mills—The major development of continuous-mill drives was associated with the wide strip mills. There were in 1954 about forty of these mills in the United States, 36 inches and larger, whose combined production was about 40,000,000 tons.

The conventional layout for these mills has a long, narrow motor room paralleling the hot-strip mill, between the mill and the slab yard. This motor room houses all of the main-drive motors, the motor-generator sets, the gear-reduction units, control and switching equipment. Figure 24—10 shows the motor room in one of the modern mills.

The drive motors for continuous mills follow a fairly uniform pattern, with the scale breaker and four roughing stands being driven by wound-rotor induction motors, and the six finishing stands driven by variable-speed DC motors.

The roughing stands are spaced so that the slab is in only one stand at a time.

The time the slab is in the rolls in the scale breaker and in Stands Nos. 1 and 2 is less than three seconds, offering ideal conditions for use of a regulator and flywheel combination.

On the older mills, it was customary to use flywheels on the third and fourth stands also, but with the increased lengths of slabs now being rolled on modern mills, the time the piece is in the rolls often exceeds four seconds, which makes the value of the flywheel questionable. As indicated earlier, the amount of momentum given up by the flywheel decreases rapidly with time, while the proportion of load taken by the motor increases with time. Hence, after three or four seconds, the motor may be driving both the mill and the flywheel. Modern installations of this type use 22-foot long slabs which are in the roughing passes for seven seconds. For this reason, flywheels were omitted and synchronous motors used for the main drives.

The DC motors driving the finishing stands present more interesting features. In the earlier mills, it was thought to be of advantage to have these motors of the same size and speed to simplify stocking of replacement parts. This necessitated gear sets on each stand which

were not only expensive but introduced chatter and extra inertia in the drive. The latest designs call for direct drive on all finishing stands except Nos. 5 and 6, where the speed is too low for economical use of direct-connected motors.

The basic data for calculating the horsepower requirements per stand for a continuous mill are based upon actual test. Composition of the steel, its temperature, speed of rolling, and width are variables in these data. In addition, allowance has to be made for the judgment that has to be exercised by the roller, who is given a certain size slab to be reduced in ten stands to a product meeting close dimensional tolerances and high surface-quality standards. To do this, the theoretical reductions per pass may have to be varied, and it may be necessary to increase or decrease the draft on certain passes as compared with the calculated reductions.

One of the measures used to compare the driving powers of hot-strip mills is the horsepower per inch of roll face. Since the main-drive motors may have different maximum torques or overload ratings, it is customary to use both the normal and overload ratings in such a comparison. Table 24—VI compares the horsepower per inch of roll face for eleven of the hot-strip mills operated by major steel companies.

The DC finishing-mill motors are designed to operate on 600 volts. At this voltage, they will run at the base speed. If lower speeds are required, the DC generator voltage is reduced, which also decreases the horsepower output but does not change the torque. Higher speeds are obtained by weakening the DC fields in the motors by the field rheostats or, in the more modern installations, by changing the field voltage by means of the rotating type of control. The standard speed range for these motors on field control is 2:1 and they are designed to have very flat speed characteristics.

When operating, the six finishing-stand motors must function as a unit and maintain their speed relationship regardless of load. Usually, a master control is installed so that, once this relationship is established, the speed of the six stands can be raised or lowered as a unit.

Table 24—VI. Main Drive Motor Horsepower Per Inch of Roll Face—Continuous Hot Strip Mills.

Stand No.	Motor Rating Used	United States Steel Corp. Dranesburg 80" Cont. Hot Strip Mill	United States Steel Corp. Gary 80" Cont. Hot Strip Mill	Republic Steel Corp. Cleveland 98" Cont. Hot Strip Mill	Jones & Laughlin Steel Corp. 98" Cont. Hot Strip Mill	Bethlehem Steel Corp. Lackawanna 79" Cont. Hot Strip Mill	Great Lakes Steel Corp. Ecorse 98" Cont. Hot Strip Mill	Youngstown S. & T. Co. 79" Cont. Hot Strip Mill	Inland Steel Co. 76" Cont. Hot Strip Mill	United States Steel Corp. Youngstown 43" Cont. Hot Strip Mill	Ford Motor Co. 56" Cont. Hot Strip Mill	Bethlehem Steel Corp. Sparrows Point 56" Cont. Hot Strip Mill
Scale Breaker	O.L.R. N.	39.06 15.62	39.0 15.62	20.4 8.17	26.1 10.4	31.6 12.65	26.1 10.4	47.4 19.0	32.8 13.5	29.1 11.6	35.6 14.3	26.8 10.71
R-1	O.L.R. N.	153.0 43.75	109.3 43.75	76.6 30.6	78.0 31.2	95.2 37.9	78.0 31.2	94.8 38.0	98.5 39.4	145.5 58.2	111.5 44.6	133.8 53.7
R-2	O.L.R. N.	153.0 43.75	109.3 43.75	76.6 30.6	78.0 31.2	95.2 37.9	78.0 31.2	94.8 38.0	98.5 39.4	145.5 58.2	111.5 44.6	133.8 53.7
R-3	O.L.R. N.	153.0 43.75	109.3 43.75	76.6 30.6	78.0 31.2	95.2 37.9	78.0 31.2	94.8 38.0	98.5 39.4	145.5 58.2	111.5 44.6	133.8 53.7
R-4	O.L.R. N.	153.0 43.75	109.3 43.75	76.6 30.6	78.0 31.2	95.2 37.9	78.0 31.2	94.8 38.0	98.5 39.4	145.5 58.2	111.5 44.6	133.8 53.7
Scale Breaker	O.L.R. N.	10.0 6.25	16.0 10.0	9.78 6.13	8.34 5.2	10.1 6.3	8.34 5.2	10.1 6.3	10.5 6.57	18.6 11.6	11.4 7.16	14.3 8.94
F-5	O.L.R. N.	112.5 56.25	90.0 56.25	73.6 46.0	75.0 47.0	70.8 44.3	75.0 47.0	70.8 44.25	73.7 46.0	111.5 69.7	85.7 53.6	85.7 53.7
F-6	O.L.R. N.	112.5 56.25	90.0 56.25	73.6 46.0	75.0 47.0	70.8 44.3	75.0 47.0	70.8 44.25	73.7 46.0	130.2 81.4	85.7 53.6	85.7 53.7
F-7	O.L.R. N.	150.0 62.5	90.0 56.25	73.6 46.0	75.0 47.0	70.8 44.3	75.0 47.0	70.8 44.25	73.7 46.0	130.2 81.4	85.7 53.6	85.7 53.7
F-8	O.L.R. N.	150.0 62.5	90.0 56.25	73.6 46.0	75.0 47.0	70.8 44.3	75.0 47.0	70.8 44.25	73.7 46.0	130.2 81.4	85.7 53.6	85.7 53.7
F-9	O.L.R. N.	150.0 62.5	90.0 56.25	73.6 46.0	75.0 47.0	70.8 44.3	75.0 47.0	70.8 44.25	73.7 46.0	130.2 81.4	85.7 53.6	85.7 53.7
F-10	O.L.R. N.	62.5 31.25	60.0 37.5	57.2 35.7	50.0 31.2	50.6 31.6	50.0 31.2	70.8 44.25	73.7 46.0	111.5 69.7	71.6 44.6	85.7 53.7
TOTAL	O.L.R. N.	1398.56 528.12	1002.6 519.3	763.0 403.0	773.0 407.0	870.0 450.0	773.0 407.0	928.0 468.0	880.0 454.0	1330.0 697.0	908.0 459.0	1090.0 557.0

NOTE—O.L.R. = Pull-Out Torque Rating on Induction Motors and Overload Rating for Frequently Repeated Peaks on D.C. Motors.
N. = Normal Motor Rating.

Table 24—VII. Installed Horsepower and Generating Capacity—Tandem Cold-Reducing Mills

	Weirton Steel Co. Weirton, W. Va. 42" Mill 5-Stand	Jones and Laughlin Steel Corp. Pittsburgh, Pa. 42" Mill 5-Stand	Bethlehem Steel Corp. Sparrows Point, Md. 56" Mill 5-Stand	United States Steel Corp. Gary, Ind. Two 42" Mills 5-Stand	United States Steel Corp. Gary, Ind. 56" Mill 4-Stand	United States Steel Corp. Drewsburg, Penna. 48" Mill 5-Stand	United States Steel Corp. Pittsburgh, Calif. 56" Mill 6-Stand	United States Steel Corp. Morrisville, Penna. 80" Mill 4-Stand	United States Steel Corp. Morrisville, Penna. 48" Mill 5-Stand
Maximum Rolling Speed (Ft./Min.)	5000	6000	4125	4600	3500	4100	4000	3110	7000
Stand No. 1—Motor Horsepower.....	1750	1750	2000	1250	2500	800	1500	3000	1750
Motor Speed (r.p.m.)	70-210	80-240	70-175	70-210	90-255	300-750	80-240	90-270	110-310
Stand No. 2—Motor Horsepower.....	3500	3500	2500	2500	4000	2000	3000	6000 ^ω	3500 ^ω
Motor Speed (r.p.m.)	125-312	150-360	130-260	125-300	125-350	100-250	150-350	150-350	165-420
Stand No. 3—Motor Horsepower.....	3500 ^ω	2-1750 ^ω	3000 ^ω	3000	4000	2500	3500	6000 ^ω	2-2250 ^ω
Motor Speed (r.p.m.)	225-450	175-350	200-450	200-400	200-470	175-350	225-450	200-500	200-415
Stand No. 4—Motor Horsepower.....	2-1750 ^ω	2-1750 ^ω	3000 ^ω	3000	4000	2500	3500	5250 ^ω	2-2250 ^ω
Motor Speed (r.p.m.)	348-696	200-400	200-500	300-600	250-580	275-550	330-660	250-565	300-625
Stand No. 5—Motor Horsepower.....	2-2250 ^ω	2-2000 ^ω	3500 ^ω	3750	—	3000	4000	—	2-3000 ^ω
Motor Speed (r.p.m.)	500-900	275-535	300-685	450-850	—	400-700	250-465	—	345-435
Reel-Motor Horsepower	800 ^ω	600	800 ^ω	700	1000	300	800	1400 ^ω	1400 ^ω
Motor Speed (r.p.m.)	300-1100	300-1450	200-800	200-1000	150-600	200-800	200-1000	150-525	350-1700
TOTAL HORSEPOWER ON DRIVES ...	17,550	16,850	14,800	14,200	15,500	11,100	16,300	21,650	21,650
MG Set No. 1—Motor Horsepower	9900	10,000	11,200	10,000 ^ω	8500 ^ω	12,000	6800 ^ω	15,000	10,500
Generator No. 1—kw.....	3500	1500	4000	1000	2000	4000	1250	2400	1400
Generator No. 2—kw.....	3500	3000	4000	2000	3200	4000	2500	2400	2400
Generator No. 3—kw.....	—	3000	—	3000	660	—	700	2400	2400
Generator No. 4—kw.....	—	—	—	600	—	—	—	2400	1200
Generator No. 5—kw.....	—	—	—	—	—	—	—	1200	—
MG Set No. 2—Motor Horsepower	9900	10,000	4900	8400 ^ω	8500 ^ω	—	13,000	10,500	14,000
Generator No. 1—kw.....	3500	3000	2800	2500	3200	—	3000	2400	2800
Generator No. 2—kw.....	3500	3200	640	2500	2800	—	3000	2400	3600
Generator No. 3—kw.....	—	500	—	—	—	—	3200	2400	3600
TOTAL MOTOR HORSEPOWER—									
MG SETS	19,800	20,000	16,100	18,400	17,000	12,000	19,800	25,500	24,500
TOTAL GENERATOR CAPACITY—kw.	14,000	14,200	11,440	11,600	11,860	8000	13,650	18,000	17,400

^ω Twin Drives^ω Triple-Armature Motor^ω Double-Armature Motor^ω One Generator per Drive Motor

These motors are started by the Ward-Leonard system, the generator voltage being increased gradually until the motors are at the base speed. The motor-generator sets to supply these finishing-mill motors have a synchronous motor as the driving unit and two generators mounted on a common bed plate for each stand. Due to the time the strip is in the mill, flywheels would be a detriment and seldom are used. The synchronous motors usually are operated at leading power factor and correct the lagging power factor caused by the inductive load on the other drives.

The top rating for these 600-volt, DC generators is about 4000 kw, due to commutator limitations. The number of these sets required for a mill varies from two to three, and the ratio of generator capacity to total motor-drive horsepower ranges from 0.60 to 1.50. The synchronous motors driving these sets must have a high pull-out torque capable of absorbing the sudden, high power peaks from the finishing stands.

The generators must operate in parallel and receive their excitation from separate exciter sets. It is essential that the voltage on the drive motors be kept constant, which is accomplished by sensitive voltage regulators. The load on the generators varies from frictional load to 200 per cent of the full-load rating. Since the synchronous motors on these sets are used for power-factor correction, it is sometimes necessary to use automatic regulation on the shunt fields to prevent wide fluctuations in the phase angles of the corrective power.

Tandem Cold-Reduction Mills—Another type of continuous mill is the cold-reduction tandem mill. Mills of this type have three, four, or five stands, with a reel to wind up or coil the cold-reduced product. The electrical problem here is more difficult than on the hot-strip mill, because tension maintained in the product between the stands must be controlled within close limits. The speeds on this type of mill have been gradually increased, several mills having a top finishing speed of 6000 feet per minute. Direct-connected motors are used on all stands, and there are a few installations using direct-connected, twin-motor drives on Stands Nos. 4 and 5 on five-stand mills. Table 24—VII gives motor and generator data on some modern tandem cold-reduction mills.

Continuous Billet Mills—Modern continuous billet mills are provided with individual motor drives at each stand. On the 30-inch mill at Fairless Works, each stand (three vertical and three horizontal) is driven through gearing by a 1750-horsepower, 300/600 r.p.m., 600-volt, direct-current shunt motor. They are supplied with power from a motor generator set consisting of four 2000-kw generators driven by an 11,200-horsepower, 514 r.p.m., 13,800-volt synchronous motor. The 21-inch billet mill at the same plant consists of two vertical and two horizontal stands, each individually driven by a 1250-horsepower, 400/600 r.p.m., 600-volt, direct-current shunt motor through gears. The power for these drives is supplied by a motor-generator set consisting of two 2000-kw generators driven by a 5600-horsepower, 514 r.p.m., synchronous motor.

Continuous Bar Mills—Individual motor drives are also provided for the separate stands of modern continuous bar mills. At the 10-inch mill at Fairless Works,

there are 18 two-high stands, all in line. Ten have horizontal rolls and eight have vertical rolls. Each stand is driven by a 600-volt, direct-current motor, in sizes ranging from 400 to 700 horsepower. Power is supplied by a motor-generator set composed of three 2500-kw generators, one 150-kw exciter, and a 10,500-horsepower, 514 r.p.m., 13,200-volt synchronous motor.

Continuous Rod Mills—Continuous rod mills consist of numerous stands arranged, for drive purposes, in groups with an individual motor driving the two or more stands in each group. The selection and distribution of motors in such a mill is dependent upon roll speed, roll diameter, amount of reduction in each stand, the horsepower required, and the degree of flexibility of control of the mill as an operating unit. The new straight-line continuous rod mill at Cuyahoga Works of the American Steel and Wire Division of United States Steel Corporation will roll 3¼-inch square billets, 30 to 34 feet long, into coiled rods from about ¼-inch to 1½-inches in diameter. The motors with their speed ranges were selected as shown in Table 24—VIII.

Table 24—VIII. Motor Drives for Modern Continuous Rod Mill.

	Horse-power	Speed (r.p.m.)
Roughing Train		
Drive No. 1—Stands 1 and 2	600	300–950
Drive No. 2—Stands 3, 4 and 5	1250	225–675
Intermediate Stands		
Drive No. 3—Stands 6, 7, 8 and 9	1750	200–550
Finishing Stands		
Drive No. 4—Stands 10 and 11	1500	225–580
Drive No. 5—Stands 12 and 13	1250	225–620
Drive No. 6—Stands 14 and 15	1250	225–620
Drive No. 7—Stands 16–23, incl.	4500 ^(a)	690–960
Drive No. 8—Stand 16a ^(b)	700	200–690
Drive No. 9—Stand 17a ^(b)	700	250–790

^(a) Motor set consists of one double-armature and one single-armature motor.

^(b) Looping stands 16a and 16b are used for rolling ¾-inch to 1½-inch diameter rods. In this case, Stands 16 to 23 are not used.

Continuous Seamless Tube Mill—In the continuous seamless tube mill at Lorain Works of the National Tube Division of United States Steel Corporation (separate mills of which are described in Chapter 41), the piercing mill is driven by a 4500-horsepower, 225 r.p.m., 13,800-volt, synchronous motor through a reduction gear having a ratio of 2.25:1. The nine continuous stands of the mandrel mill are individually driven by direct-current motors with a combined rated capacity of 8500 horsepower. The product from the mandrel mill, after reheating, is further processed in either the stretch-reducing mill or the sizing mill. The stretch-reducing mill is made up of twelve two-high, overhung roll stands; each roll stand is individually driven by a 200-horsepower, 850/1700 r.p.m., 600-volt, shunt motor. The sizing mill consists of twelve overhung roll stands, each with an individual drive consisting of a 76-horsepower, 850-1700 r.p.m., shunt motor.

SECTION 11

MOTOR-ROOM VENTILATION

A very important feature of the motor rooms for large electrical installations is the ventilating system.

It is necessary to supply sufficient clean air to maintain normal operating temperatures for the equipment.

Several recent installations have been designed with down-draft recirculating systems, where the air which passes into the motors and generators is discharged into the basement, pulled or blown through surface-type water coolers, and sent back to the motor room. Since there is always some leakage, a certain amount of make-up air is required, and this is introduced into the basement through a cleaning system.

Several installations have allowed for only 10 per cent make-up air, which may have been adequate when the building was new but, as cracks developed and loose joints appeared in the structure, this quantity of make-up air was too small and, as a result, outside air infiltrated into the building. These recirculating systems also have the detrimental effect of picking up the carbon dust from the brushes of the motors and generators and, in the recirculating process, this dust is deposited in the windings since no cleaning equipment is included in the indoor system. If a closed system is specified, the make-up air provided for should be at least 25 per cent.

The other type of ventilating system supplies 100 per cent new air which is blown into the basement through a cleaning system, then forced up through the motors into the motor room, where a positive pressure is maintained that prevents infiltration. This system requires larger fans and motors than the recirculating

system, but requires no cooling-water system and is less costly to maintain. Usually, there are three or four fans operated as needed so that, in winter time, the heat from the motors can be used to warm the room.

In the recirculating system, the cooling system has to be designed so that the difference in temperature between the cooling water and the outgoing air is not less than 10° F. Since the air delivered to the machines should not be above 104° F, the cooling water should not be warmer than 94° F. The following table indicates the amount of air required for cooling per kw loss in each machine:

Cubic Feet of Air Per Minute Per KW Loss	Rise in Air Temperature (Degrees F)
100	32.9
110	29.9
120	27.4
130	25.3

For cleaning the air, various types of filters have been used. The oil-immersed traveling-screen type, one of the early designs, did not remove the fine particles of soot and smoke from the air, and the oil was a fire hazard. Later installations favor the electrostatic type of air cleaner that eliminates even the finest foreign particles from the air.

SECTION 12

AUXILIARY DRIVES

The auxiliary drives on mills are as important as the main drives, and require proper application and co-ordination of types and sizes of motors. Both AC and DC motors are used, the former being the accepted ones for driving fans, pumps, and, occasionally, shears and run-out tables. The majority of the motors, however, are mill-type DC units, because of their ruggedness and the many operating characteristics that can be built into them. These mill-type motors were developed by the steel industry in cooperation with the electrical-equipment manufacturers for use in the rough service of driving mill tables, heavy-duty cranes, manipulators, etc. Their ratings and principal dimensions have been standardized by the Association of Iron and Steel Engineers into eleven frame sizes, ranging from 5 horsepower to 200 horsepower, based on the 1-hour rating. They can be obtained with several designs of windings from straight series to shunt, and with various types of ventilation. The designs are very rugged, and the armatures have low inertia in comparison with general-purpose motors. Table 24—IX describes motors for various auxiliary-drive applications.

Table Rollers—The early strip-mill run-out table rollers were equipped with variable-speed AC motors, one per roller, and speed variations were obtained by varying the frequency of the power supply by a variable-speed motor-generator set. Since 1937, the DC mill-type motor has been used for this application.

Schloemann rollers, a German design, in which the movable part of the table roller is the rotor of the motor and the stationary part wound to form the stator of an induction motor, have been used on some mills. When such rollers handle hot products, they must be water cooled. Speed variations are obtained by frequency control from a motor-generator set.

In modern mills, the rollers may be driven by individual, small direct-current motors of 3 to 4 horsepower, or they may be arranged in consecutive groups with

each group driven separately by a direct-current motor of ample size through a line shaft and gearing. Variable-voltage control is used to regulate the speed of the table rollers. The current generally is supplied by a motor-generator set.

Screw-Downs—Proper selection of motors for screw-down drives on the various mills is very important. In some mills, such as blooming and slabbing mills, a maximum roll lift of 66 inches may be required, with lifting speeds of up to 645 inches per minute. On strip mills, a maximum operating lift of only a few inches may be required, but the opening must be controlled within a few thousandths of an inch.

Regardless of the speed or distance involved, the screw-down motor and control must be capable of positioning the rolls within close limits. The inertia of the armature and brake wheel makes this a difficult problem, even with a compound-wound motor, and has resulted in the adoption of the Ward-Leonard system of control. This system has an added advantage in that it limits the amount of torque that can be applied to the motor, thus preventing damage to the screws by jamming.

On primary mills, modern screw-down drives usually consist of two 230-volt, DC motors of 100 to 200-horsepower rating. If operating under variable-voltage control, they are shunt motors with shunt brakes. If magnetic control is used, they are compound wound with reversing, plugging, dynamic braking, series-parallel control. The parallel connection is used to obtain higher speeds for long screw-down travel.

Another feature that can be incorporated in the control is the automatic preset device. In mills rolling fair-sized orders of a given product, this device positions the rolls automatically after each pass to give the correct reduction for each succeeding pass. Thus, the operators can concentrate their attention on the control of the main drive, table and manipulator motors, the screw-

Table 24—IX. Motor Sizes on Screws, Manipulators, Shears, Etc., on Modern Blooming and Slabbing Mills*

Company and Size of Mill	Great Lakes 46-inch	U. S. Steel Morris- ville, Pa. 40-inch	U. S. Steel Braddock, Pa. 45-inch	Inland Steel 46-inch	Jones & Laughlin 46-inch	Youngs- town Sheet and Tube 45-inch
Screw Motors (No. and hp.)...	2-150	2-150	2-200	2-150	2-200	2-200
Screw Motor Control.....	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage
Manipulator Motors (No. and hp.).....	4-150	4-100	4-150	4-150	4-200	4-150
Lift Fingers (No. and hp.)....	4-50	2-100	4-150	4-150	2-150	1-150**
Manipulator Control	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage **
Front Table Drive (No. and hp.).....	2-150	2-100	2-150	2-150	2-200	2-150
Front Table Control.....	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage
Back Table Drive (No. and hp.).....	2-150	2-100	2-150	2-150	2-200	2-150
Back Table Control.....	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage
Shear (maximum size cut)....	64 in. x 6 in.	16 in. x 22 in.	60 in. x 9 in.	40 in. x 8 in.	800 sq. in.	600 sq. in.
Shear Motors (No. and hp.)...	2-275	2-250	2-250	2-275	Hydraulic	2-500
Shear Motor Control.....	Variable Voltage	Variable Voltage	Variable Voltage	Variable Voltage	Pressure Pumps	Variable Voltage

*Courtesy, "Iron and Steel Engineer."

**All auxiliaries under variable-voltage control.

down settings for each pass being made automatically.

Manipulators and Side-Guards—Mill tests have shown that steel being rolled on primary mills is in the rolls only from 25 to 70 per cent of the total rolling time, the remainder of the time being required for screw-down, manipulator, and table operation. In a modern slabbing mill, the maximum opening of the side-guards is 115 inches, with an overtravel of 40 inches to permit changing of the table rollers. Each of the side-guards is driven by a 150- or 250-horsepower motor. Due to the heavy plugging service required for these devices, and the heavy moving mass that has to be controlled, modern installations are using shunt-wound, variable-speed motors with Ward-Leonard control for this service.

The lift fingers on the manipulators are required to make four-foot lifts and complete the cycle in two

seconds. They always must stop below the top of the rollers, and represent another application for variable-voltage control. Where magnetic control is used, compound-wound mill-type motors with reversing, plugging, dynamic-braking features, plus a jamming resistor and arranged to operate two motors in series with series brakes, should be specified.

Blooming-Mill Shears—Blooming-mill shears can be driven either electrically or hydraulically. Many of the older installations were driven by an induction motor with high secondary resistance to allow the flywheel to supply the peak power for the cut. The motor ran continually and the shear was actuated through a clutch. Some of the more recent shears are of a start-stop type, driven by two shunt-wound motors of 200 to 300 horsepower, each equipped with a shunt brake and variable-voltage control.

SECTION 13

FUTURE DRIVES

This résumé on drives for steel mills is indicative of the ever-changing design in electrical apparatus. It is no longer a problem to add loads of many thousands of horsepower to existing systems. Motors have been improved in design and increased in rating so that they can be obtained for all types of service. Improved con-

trols, too, have kept pace with motor improvements, so that it is possible to maintain close relationship between individual large motors on successive continuous-mill stands rolling product at rates up to 6000 feet per minute.

The present trend is toward still greater flexibility in

drives. Recent continuous mills have been designed with separate generators for each drive motor; this allows the greatest range in speed.

Some few mills are introducing the mercury-arc rectifier as a source of DC power to substitute for the motor-generator set. With electronic control, this device can maintain voltage within very close limits and has a higher over-all efficiency than a motor-generator set.

The story of motor drives for mills is one of continual improvement in design to obtain higher speeds and closer regulation, to provide for more and better products from the mills.

Chapter 25

ROLLING OF STEEL INGOTS TO BLOOMS, SLABS AND BILLETS

SECTION 1

PRODUCTION OF BLOOMS AND SLABS

Introductory—Prior chapters have described the several processes for making steel and the production and treatment of steel ingots. This chapter relates to the first stages in forming useful products from ingots by hot rolling. Hot working by other means, such as forging by hammering or pressing, was discussed briefly in Chapter 19.

These early stages consist of a series of operations whereby the ingot cross-section is reduced to a square, oblong, round, or other simple shape, all having rounded corners, and of dimensions approximate to nominal specified size. The length of the ingot is increased, corresponding to the decrease in cross-section. The concluding operations cut a relatively short length called **crop** from each end of the rolled piece as scrap, and cut the remaining long piece, if necessary, into multiples to suit the required lengths or weights for subsequent operations.

These resulting pieces are known by the general names: **blooms**, **slabs**, and **bILLETS**. There is no widely accepted precise definition for any of these terms, and local applications of the terms are used somewhat on a traditional basis. Distinctions are made according to general appearance, influenced by overall size and the proportions of the three linear dimensions, and also by

intended use. The distinction between blooms and billets is principally a distinction of size, billets being smaller than blooms in cross-sectional area, and both having a length several times greater than their maximum cross-sectional dimension. The distinction between blooms and slabs is principally one of cross-sectional dimension proportion, blooms tending to be round, square, or nearly square, and slabs being always oblong and tending to be relatively wide, thin, and (until recently) of short length. There are many exceptions, and there are special names for pieces intended for special uses.

For example, any piece to be rolled into a plate almost invariably is called a slab, regardless of size or of dimension proportion. Likewise, any piece produced on a billet mill is termed a billet, regardless of shape and size, with the exception of round billets. **Round billets** intended for seamless-tube piercing, especially if in short lengths, are usually known as **tube rounds**. **Blooms** in short lengths are sometimes called **blanks**, or **blocks**, and special-shape blooms for structural sections frequently are called blanks regardless of length.

A rude guide using only the cross-sectional characteristics as the distinguishing features, which may serve in place of definitions, is shown schematically in Figure 25—1.

GENERAL FEATURES OF BLOOMING AND SLABBING MILLS

It is possible, and often quite economical, to roll ingots directly through the bloom, slab, or billet stage into more refined and even finished steel products in one mill, of varying numbers of stands from one to about twenty, in a continuous operation, frequently without any reheating. Large tonnages of standard rails, wide-flange beams, and plates, and a lesser proportion of wide hot-rolled coiled strip are produced regularly by this practice from ingots of medium to large size. A few very small plants follow a similar practice in rolling small ingots, such as 4-inch to 6-inch squares, directly into wire rods, concrete-reinforcing bars and other bar products.

However, most of the ingot tonnage is rolled into blooms, slabs, or billets in one mill, following which they are cooled, stored, and eventually rolled in other mills or forged. The reasons for this more common practice are primarily economic; sometimes size or shape of product and certain quality requirements peculiar to the final article to be manufactured determine the steel-rolling method.

A variety of names for mills rolling ingots has come

into common use to differentiate between them in reference to the particular kind of product intended or generally produced, the basic mechanical design features, or the general layout of the mill. For a long time, the term **blooming mill** was used rather commonly for all such mills, but, with the increasing variety, it has come to be restricted by many to a mill producing blooms, and the term **primary mill** is gaining acceptance as a generic term to cover both blooms and slabs. In addition to a name, these mills are designated as to size, expressed in inches, with the result that almost every mill is known by a combination of size and descriptive name which, taken together, with the size always expressed first, indicate a rough mental picture of the mill; for example, 54-inch blooming mill, or 43-inch three-high mill, or 45-inch slabbing mill.

The composite name applies specifically to the roll stand, or to the first roll stand if the mill is a multiple-stand one. However, the name is used as an abbreviation for the entire group of facilities needed to produce blooms, slabs, or billets and for the organization operating them, known more properly as a mill department.

TYPICAL CROSS-SECTION
AND
DIMENSIONAL CHARACTERISTICS*



SLAB

ALWAYS OBLONG
MOSTLY 2 TO 6 INCHES THICK
MOSTLY 24 TO 60 INCHES WIDE



BLOOM

SQUARE OR SLIGHTLY OBLONG
MOSTLY IN THE RANGE 6" X 6" TO 10" X 12"



BILLET

MOSTLY SQUARE
MOSTLY IN THE RANGE 2" X 2" TO 5" X 5"

* DIMENSIONS USUALLY GIVEN TO NEAREST ROUND NUMBER.
ALL CORNERS ARE ROUNDED, AS SHOWN.

Fig. 25-1. Comparison of the relative shapes and sizes of rolled steel governing nomenclature of products of primary and billet mills.

Primary-Mill Activities—The activities of all primary mills are the same, and it follows that the facilities for carrying out these activities are fundamentally the same in all mills, but they differ in detail according to the particular requirements of each mill. The primary operation is the conversion of a steel casting (the ingot) into a rolled steel product; the secondary function of the rolling operation is to produce this rolled steel product in pieces of the desired cross-sectional dimensions and weights. Auxiliary operations, outlined below, include cropping, conditioning, cutting to length, and the collection and assorting of crops, roll scale and other by-products that are subsequently delivered to the steel-producing and blast-furnace departments.

The primary operation is carried out in a sequence of thermal and mechanical operations:

- (a) Heating the ingots (discussed in Chapters 20 and 21).
- (b) Breaking up the coarse crystalline structure of the ingot into a refined structure by heavy rolling pressure and recrystallization during hot working.
- (c) Closing solidification voids by the same means as in (b).
- (d) Cutting off such portions of steel as are metallurgically (both physically and chemically) unsuited for the intended final purpose.
- (e) Cutting off test specimens.
- (f) Cooling to atmospheric temperature those products not destined for immediate further hot working.

The secondary function is performed in conjunction with operations (b), (c), and (d), above, since they can

be carried out with the same facilities. The essential steps in performing the secondary function are:

- (a) Positioning the rolls, which have been shaped to produce the desired cross-section.
- (b) Cutting the rolled piece to measured length or to specified weight per piece.
- (c) Weighing individual pieces, or weighing and measuring test specimens.

The auxiliary operations pertain to the rolled product and the by-products:

- (a) Conditioning the steel products by removing injurious surface imperfections, removing portions containing injurious internal defects, and by correcting physical conditions by straightening bent pieces, cutting overlength pieces to proper length, and so on.
- (b) Collecting identified crops, other ferrous scrap, roll scale, and cinder and delivering these by-products to steel-producing and blast-furnace departments.

In order to carry out these activities, all primary mills have the same general kind of facilities and all are operated in the same general manner, but no two are exactly alike, although there are a few instances of close similarity. The differences among mills are due to the sizes of ingots and product, the particular kind of product and the quantity required to be rolled in a given time, as well as to the date when the mill was built or subsequently altered, and to the most economical kind of fuel and power available at that time. The differences are in size, in details of design, in the auxiliary equipment provided, and in the arrangement with relation to other facilities, especially those producing the ingots and those rerolling the products of the primary mills.

In a large number of present-day mill arrangements, the primary mill supplies hot steel products either directly or through reheating furnaces to other mills producing semi-finished products such as billets and coiled strip, or finished products such as rails or structural shapes, and the number of such mills supplied by one primary mill is from one to three in various tandem and parallel combinations. When the product coming from the primary mill is rolled, while it is still hot and without reheating, directly in a succeeding mill, rolling capacities must be considered. In the ideal case, the primary mill will just keep the succeeding mill or mills busy, without loss of time in either the primary mill or the succeeding mills. The arrangement of the mills determines the material-handling facilities and the size and location of the mill buildings.

The basic operation in a primary mill is the gradual compression of the steel ingot between the surfaces of two rotating rolls, and the progression of the ingot through the space between the rolls. The physical properties of the ingot prohibit making the total required deformation of the steel in one pass through the rolls, so that a number of passes in sequence are always necessary. There are several ways in which a sequence of passes can be made, and some particular way is selected by choosing one which is suited to the quality of material to be produced and also likely to be the least expensive. This choice determines the general type of mill. The elements of pressure, motion, weight, and time, together with the sizes of ingots and sizes of products, plus the quantity of product desired in a given time period, determine the size and design of the mill.

Each type of mill has been developed to meet a definite need, and, in performing this one task, it is superior to other types. However, mills are expensive to install and are capable of long life, while changing conditions

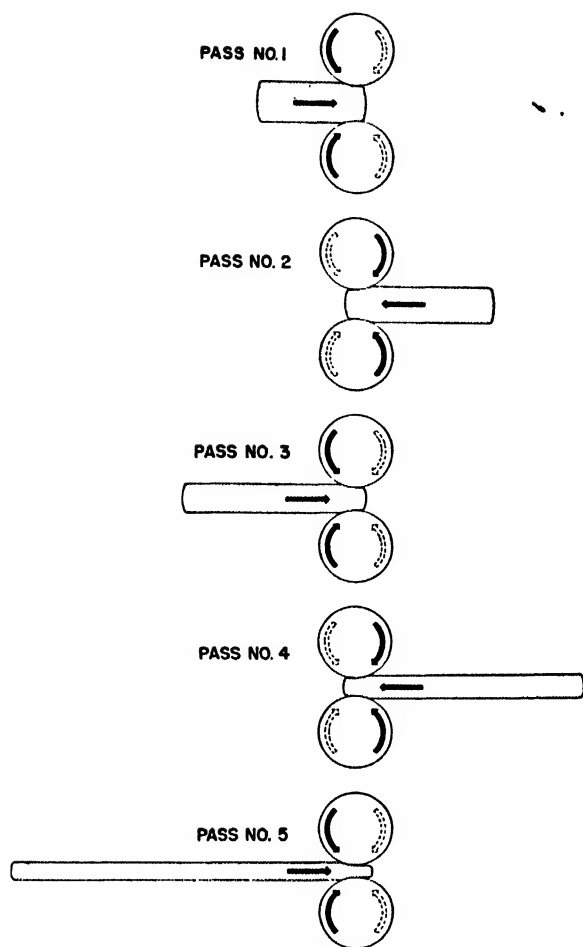


FIG. 25-2. Diagrammatic representation of the sequence of rolling operations involved in reducing an ingot to a slab on a reversing two-high mill.

in the steel industry sometimes result in a number of changes in the needs to be met by a primary mill during the period of its useful life. This results in adapting existing mills to new needs, frequently with alterations to some of the facilities, and many of the present-day primary mills are rolling ingots and producing products that were not contemplated when the mills were built.

The general characteristics of each of the types of primary mills is outlined below.

Two-High Reversing Mill—In the phrase “two-high reversing mill,” the term “two-high” refers to the fact that the mill consists of two rolls, one over the other, as in Figure 25-2. “Reversing” means that after the piece has gone through the rolls in the direction of the first pass, the rolls are brought to a standstill and then caused to rotate in the reverse direction shown in Pass No. 2, so as to impose the next reduction on the piece (the next pass), and so on until the piece has been reduced the desired amount. This type of mill has maximum flexibility in size range of ingot and product, as well as wide range of adjustment in the amount and rate of steel deformation in any pass. The mill is manually operated in order to permit instant variations in practice, the crew on electrically-driven mills consisting of two men and three on steam-driven mills. A very high degree of skill is essential in each man in his own duties and, in addition, there is the necessity for com-

plete coordination among crew members of all motions of the equipment if the highest production rate is to be attained, since each individual is controlling a part of what is, in effect, one machine working on one piece of steel.

For the same size ingot rolled to the same size product, this type of mill has a lower production rate, in terms of tons per unit of time, than any of the others, and it was due to this relatively low output rate that the other types were devised. However, the two-high reversing mill remains the principal type in use, exceeding in numbers all other types combined, by a wide margin. Its chief characteristic, flexibility, is so valuable that this type of mill is combined frequently with other types to compensate for low production rate, the extreme being two two-high mills of different sizes arranged in tandem. Many such combinations are in operation at this time.

While there is no limit to the size or weight of ingot for which this type of mill is used, there is an economic limit to the length of piece which should be rolled from one ingot on one mill. It is determined by two factors: inertia of moving parts of equipment, and size of building to house the mill. The ingot is a relatively short casting, mostly from five to nine feet long. Mill parlance refers to length of ingots as their height, because they are poured vertically and this dimension is height at first, but the mill operates on the ingot when the latter is in a horizontal position. In the early passes, the length of the piece is so short that one to three revolutions of the rolls is sufficient to complete a pass, and the rolling speed is preferably very slow. Reversals can be rapid, power loss in reversing is relatively slight, and impact among moving parts is light. As the piece becomes longer in successive passes, the rolling speeds must be increased if the steel is not to lose too much heat. At high speed, the motion in one direction is only a few seconds in duration, but reversals at high speed consume excessive power and subject all the moving parts in the mill tables as well as in the mill and the mill drive to severe shock, even with the best electrical controls. The mill table should be as long on each side of the mill as the rolled piece, plus adequate clearance for manipulation. Any appreciable increase in length of the rolled piece requires the combined length of the tables on both sides of the mill stand to be increased by twice that amount and requires a corresponding increase in length of building for proper accommodation. This economic limit of length of an uncropped piece rolled from an ingot seems to be about 90 feet.

The two-high reversing mills in use today are of three forms. The predominating form, known simply as a blooming mill, is provided with one pair of horizontal rolls in which several grooved passes provide the means of controlling the shape of the piece during rolling, and particularly of working the corners of the piece. This form of mill exists in a wide range of sizes and is designed to roll ingots of square or nearly square cross-section with a maximum thickness of about 34 inches, and can edge vertically, in a grooved pass, a piece of about 40 inches maximum width. A typical modern example is the 46-inch blooming mill in the Lorain Works of National Tube Division of United States Steel Corporation, shown in Figure 25-3.

A variation of this form, designed for edging pieces up to about 78 inches wide resting on their narrow edges, is known as a high-lift blooming mill and occasionally as a blooming and slabbing mill. It differs from the more conventional form in being provided with higher mill housings to permit greater elevation of the top roll for the edging passes on wide pieces. The roll body is usually longer and has fewer grooved passes, and the motors

FIG. 25-3. The 46-inch two-high reversing blooming mill in the Lorain Works of National Tube Division of United States Steel Corporation. Relationship of this mill to other mills in the No. 4 Blooming, Bar and Billet Mill is shown in Chapter 41.

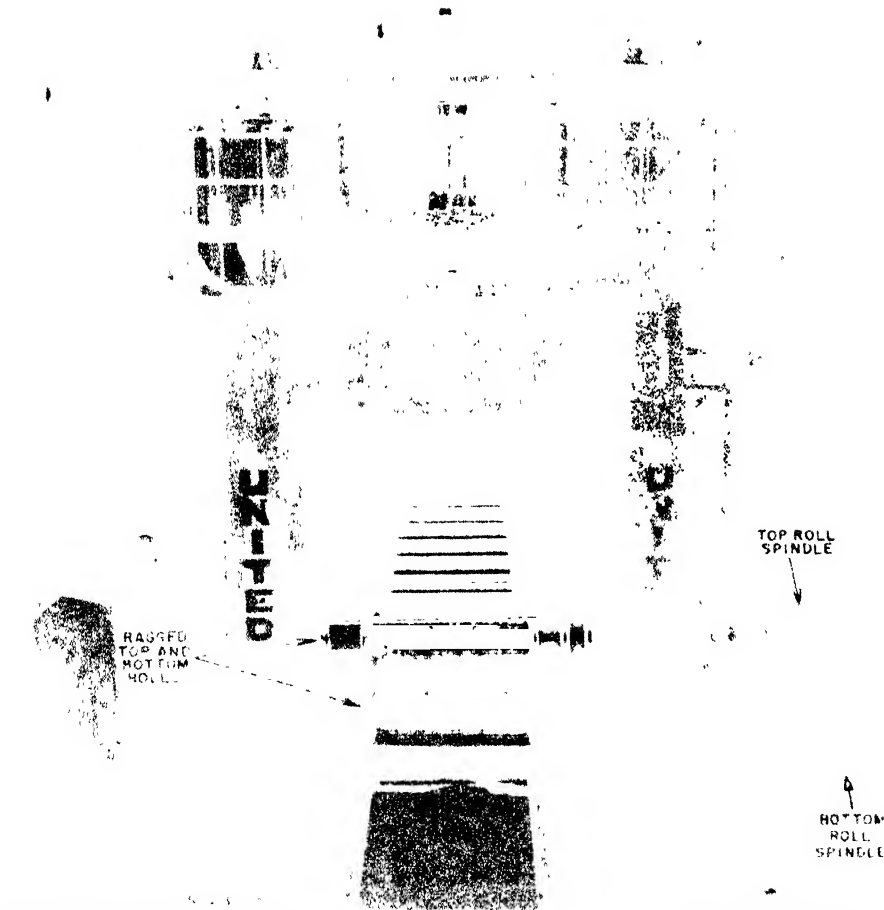
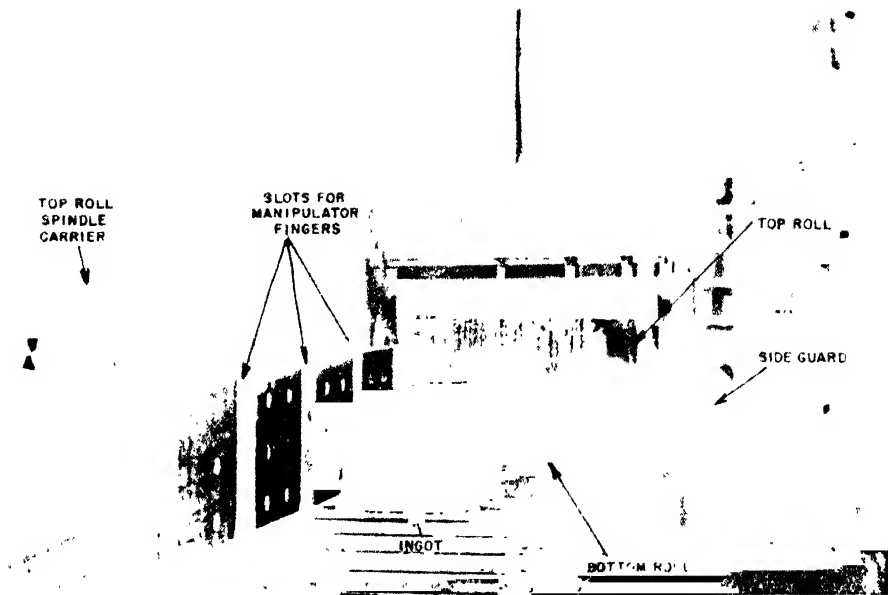


FIG. 25-4. The 45-inch, high-lift, two-high, reversing blooming and slabbing mill at the Geneva Works of Columbia-Geneva Steel Division of United States Steel Corporation.

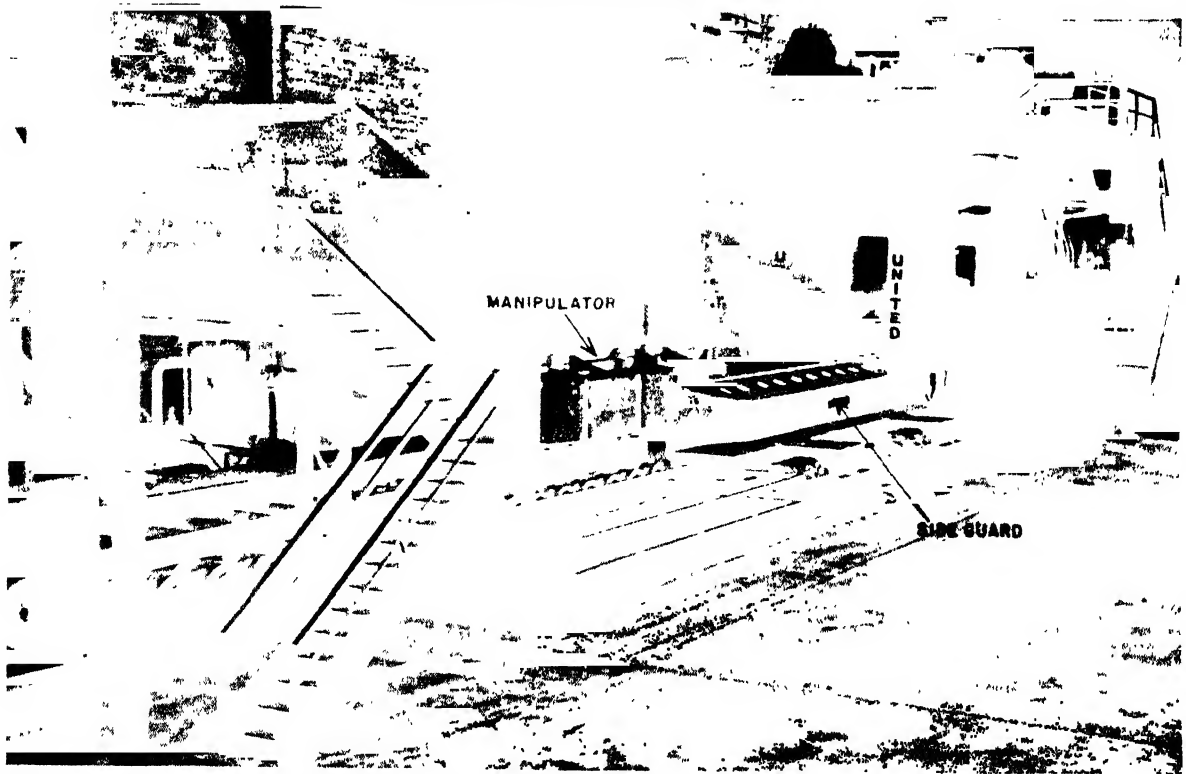


FIG. 25—5. Overall view of the 45-inch universal slabbing mill in the Homestead District Works, designed and operated solely for the production of slabs. Auxiliaries of this mill are shown in Figures 25—9 to 25—12, inclusive.

which operate the screws for controlling top-roll elevation are usually larger in order to provide faster raising and lowering of the top roll. This latter feature is provided since edging passes are few in number but require excessive time unless the top-roll travel is accomplished in no longer time than that required for the manipulator to turn the ingot 90 degrees. This form of mill can be provided with interchangeable sets of rolls so that it can produce not only wide slabs but also blooms and, for this reason, it has been installed rather widely since around 1940. It has the greatest range in sizes of ingots and products of all three forms. The 45-inch high-lift blooming and slabbing mill at Geneva Works of Columbia-Geneva Steel Division of United States Steel Corporation, shown in Figure 25—4, is a modern example of this type.

A third form, actually a special-purpose mill limited to the production of wide slab sections, is the universal slabbing mill, often referred to simply as a slab mill. It is designed to increase the production rate for wide slabs by avoiding expenditure of the time required for vertical edging passes in a blooming mill, and is provided with a pair of vertical rolls in addition to the pair of horizontal rolls, the vertical rolls performing the edging. In this arrangement, no grooves are used in either pair of rolls. This lack of grooves prevents support of the ingot corners during rolling, and also limits the mill to production of slab sections, since it is incapable of avoiding consistently the inadvertent production of a diamond-shaped product in square or nearly square sections. The slab product rolled on universal mills has relatively sharp corners compared to that of the other mills. Figure 25—5 illustrates the 45-inch universal slabbing mill at Homestead District Works of United States Steel Corporation.

Rolling a certain size ingot to a certain size bloom is done at the same rate in the conventional and the high-lift blooming mills. The installation cost of the latter is slightly higher. Rolling the same size ingot to the same size slab is done at a somewhat faster rate in the universal slab mill than in the high-lift blooming mill, but the installation cost of the universal slab mill is substantially higher due to the cost of the vertical roll assembly and the additional prime mover to drive it. Before the high-lift mill was developed, the universal mill was the means for producing wide, heavy slabs for plate mills, and it has been built in sizes to produce widths to about 78 inches, maximum.

Two-High Tandem Mill—This mill consists of several single stands, each containing one pair of rolls, spaced one following another at such distances as to permit the rolled piece to be free between stands. Normally, the piece is rolled one pass in each stand, but at Edgar Thomson Works a unique arrangement of tables (Figure 25—6) transfers the piece from the second stand back to enter the first stand and second stand again for the third and fourth passes.

Because each stand rolls only one pass, there is no time lost in reversing the travel of the piece, with the result that this type of mill has the maximum output rate of all types. In addition, each stand can be designed for the almost ideal draft and rotating speed for each of the successive rolling passes. This is a great advantage in the early passes where the weakening effect of the ingot crystal structure exists. Opposed to these three very desirable features is the narrow limit of ingot and product cross-sections which the mill can roll without sacrificing the advantages by excessive idle time to change rolls and by compromise on roll-pass designs. For these reasons, the tandem mill is used as a roughing

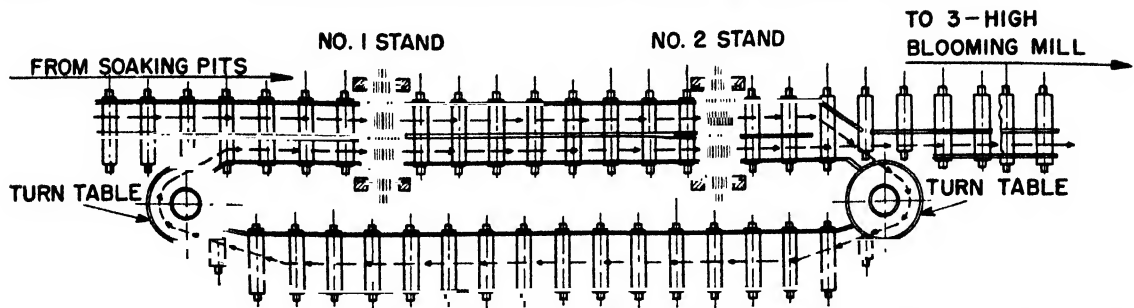


FIG. 25-6. Unique arrangement of transfer tables at Edgar Thomson Works making it possible to return a rolled bloom from the second stand back to the first stand for another pass in each of the two roll stands. Four passes may be given in the two stands combined, without the necessity for reversing either mill. The arrows indicate the path of travel of the steel. While not indicated in the sketch, the piece is turned end for end before entering No. 1 stand the second time.

mill to supply hot steel to succeeding mills which permit flexibility in product sizes, but is confined to a narrow limit in ingot cross-section dimensions.

Two installations have been built in which a tandem mill was followed by continuous billet mills, and the entire mill in each case sometimes was called a continuous mill. This designation is slightly misleading with respect to the tandem roughing mill.

Due to the large amount of equipment in a tandem mill, it is the most expensive mill to build for a given ingot-to-product reduction but, when scheduled to full capacity, its operating cost per ton of product is lower than that of any other type mill rolling the same-sized ingot to the same product. A tandem mill can roll any size of bloom or slab for which it is designed, but its application has been to the medium and small sizes, at least up until 1955, since these were the only sizes in which sufficient orders could be obtained to provide the quantity of production to match the capacity of a tandem mill designed to roll them.

Three-High Mill—This type of mill consists of one stand containing three horizontal rolls, one above the other, each of which has grooved passes, and an elevating table on each side of the mill so that the piece being rolled can be passed alternately between the bottom and middle rolls and between the top and middle rolls. In order to keep the roll dimensions and mill and table dimensions from being prohibitive in size, a compromise pass design is used wherein the grooves in the middle roll are used for rolling both with the top and with the bottom roll. This method limits the reduction possible

in the second pass through each middle-roll groove to less than that possible in single passes. The rotating speed of the mill is also a compromise, since each pass should be progressively faster than the one preceding, but, with all passes in one set of rolls, that is impossible. The speed selected is usually too fast for the early passes and too slow for the last ones. There are normally five or seven passes, so that it is often necessary to employ heavy ragging in at least the first two passes to prevent slippage of the steel both in entering the pass and during rolling.

Because the rolls rotate in one direction only and at constant speed, a relatively simple and inexpensive constant-speed prime mover, aided by a flywheel, is used as the main drive. No power is lost overcoming inertia as there is no reversing. Raising and lowering the tables to guide the piece into successive passes can be accomplished in less time than the corresponding manipulations in a reversing mill. For these reasons, a three-high mill is less expensive to build and has a higher output rate than a two-high reversing mill rolling the same ingot to the same product.

Opposed to these favorable features are certain disadvantages. Like the tandem mill, the three-high mill with its fixed passes is rather inflexible with regard to size of ingot cross-section and product, except at the expense of serious operating delays due to roll changing. Then, too, its usual rather fast rolling speed in the first few passes makes it less desirable generally for the rolling of ingots in these passes.

The three-high mill is best adapted to a place as an

FIG. 25-7. A 40-inch, three-high blooming mill, forming an intermediate stand that rolls roughed down ingots (from another mill stand) to blooms that are supplied to the 28-inch billet mill in the far left background. (Courtesy, United Engineering and Foundry Company.)

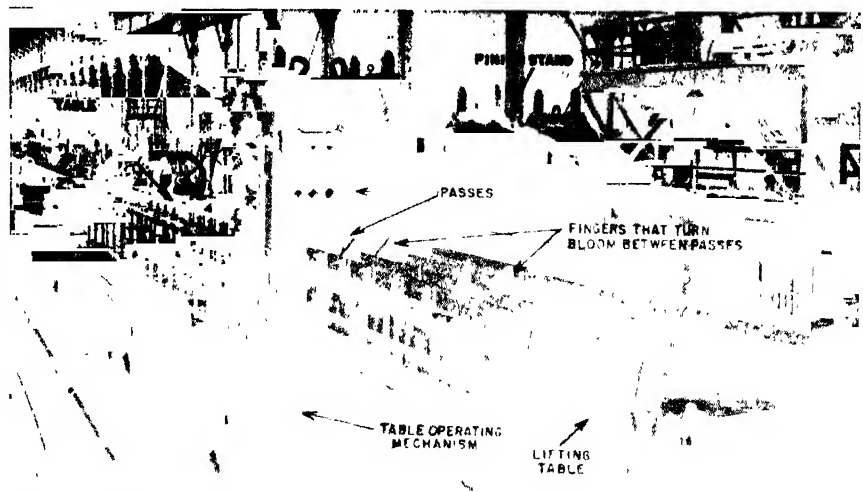


FIG. 25-8. Diagrammatic sketch of a modern high-lift, two-high, reversing blooming mill, with the main constructional details indicated.

LEGEND

- | | |
|-----------------------------------|---------------------------|
| A. Counterweight latches. | L. Screwdown drives. |
| B. Roll-balancing counterweights. | L-1. Crossover shafts. |
| C. Steelyard rods. | M. Housing separator. |
| D. Scale pit. | N. Wobblers. |
| E. Counterweight pits. | O. Universal couplings. |
| F. Housing shoe. | P. Main-drive spindles. |
| G. Roll-change rig rail. | R. Spindle counterweight. |
| H. Housings. | S. Top-spindle carrier. |
| K. Rolls. | T. Motor couplings. |
| K-1. Bullhead pass. | U. Main drive motors. |
| K-2. Edging passes and collars. | |

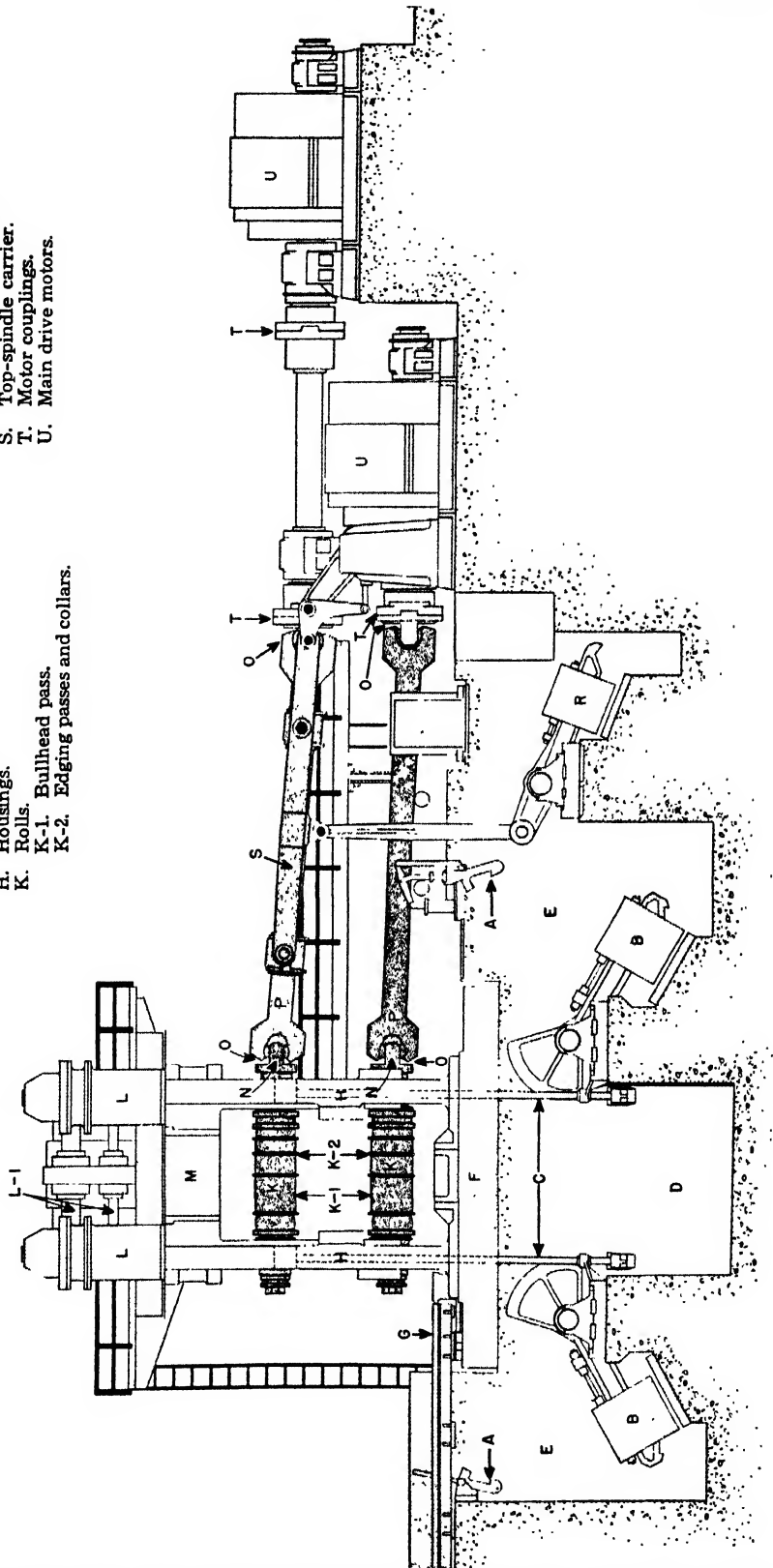




FIG. 25-9. Ingot buggy (pot car) transferring an ingot, heated in the soaking pits in the background, to the ingot receiving table of the 45-inch universal slabbing mill in Homestead District Works.

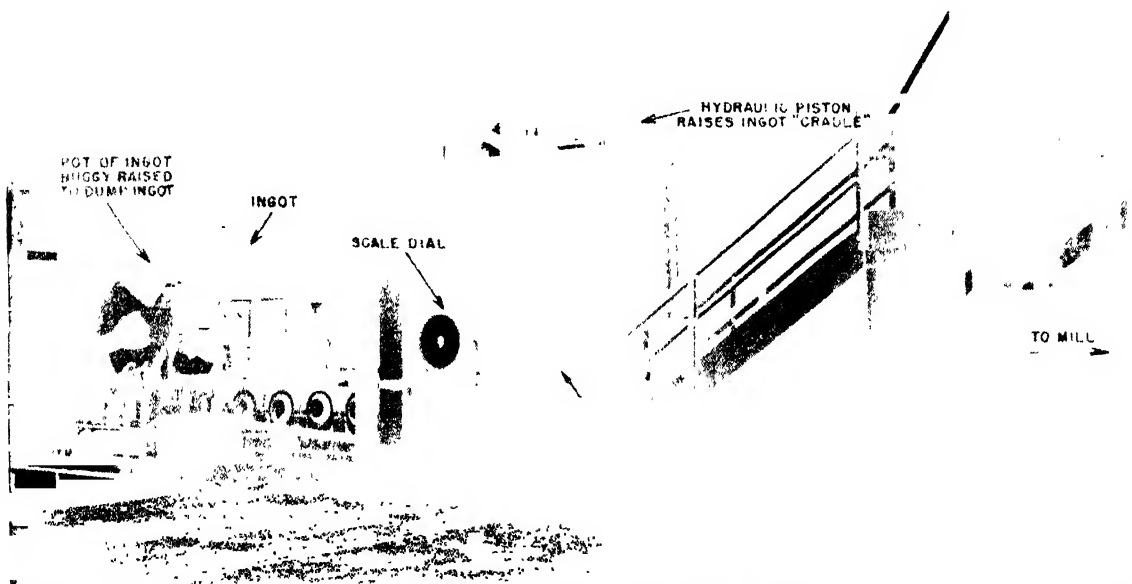


FIG. 25-10. Ingot buggy discharges hot ingot onto ingot receiving table (left) of the 45-inch universal slabbing mill in Homestead District Works. The receiving table of this mill is equipped with both an ingot turner and scale, as shown.

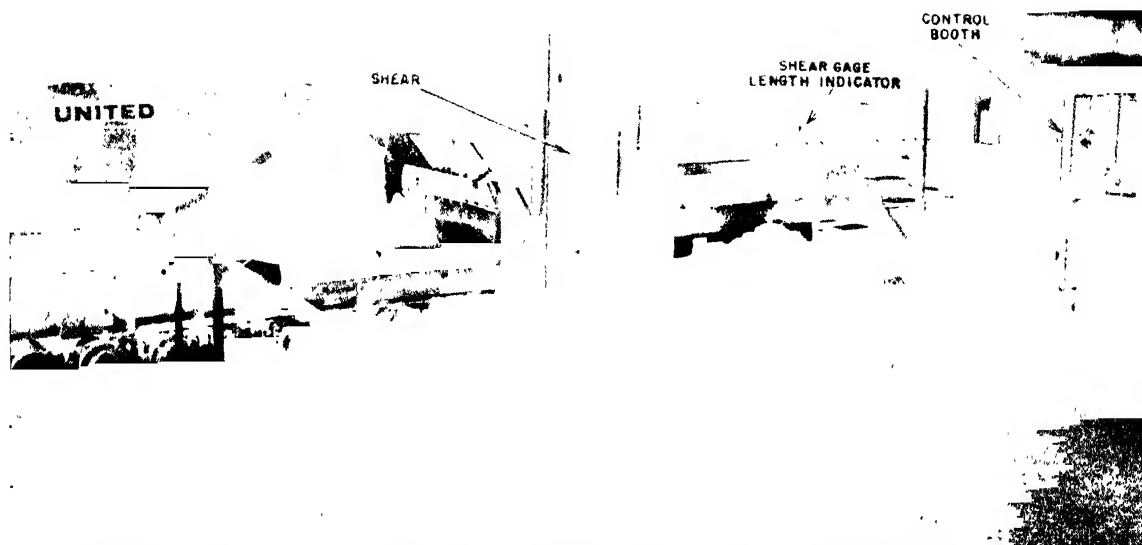


FIG. 25-11. The down-and-up-cut shear of the 45-inch universal slabbing mill in the Homestead District Works.

intermediate stand (Figure 25-7) where it can be supplied with bloomed-down ingots (ingots which have received a few heavy reduction passes), or large blooms, and where it can supply smaller hot blooms of one or two sizes directly to a following mill. Its rather simple design permits a rolling schedule among three mills in sequence which can be balanced within reasonable limits by alterations in the pass design to adjust the

entering bloom and the product bloom sizes to control the production rate of the preceding and following mills.

It finds a use as an ingot-rolling mill in those plants whose standard ingot sizes are very small, on the order of 14-inch square or less in cross-section, and whose total ingot-producing capacity is also quite modest. In these applications, the three-high mill functions more as a billet mill or as a roughing stand for a directly

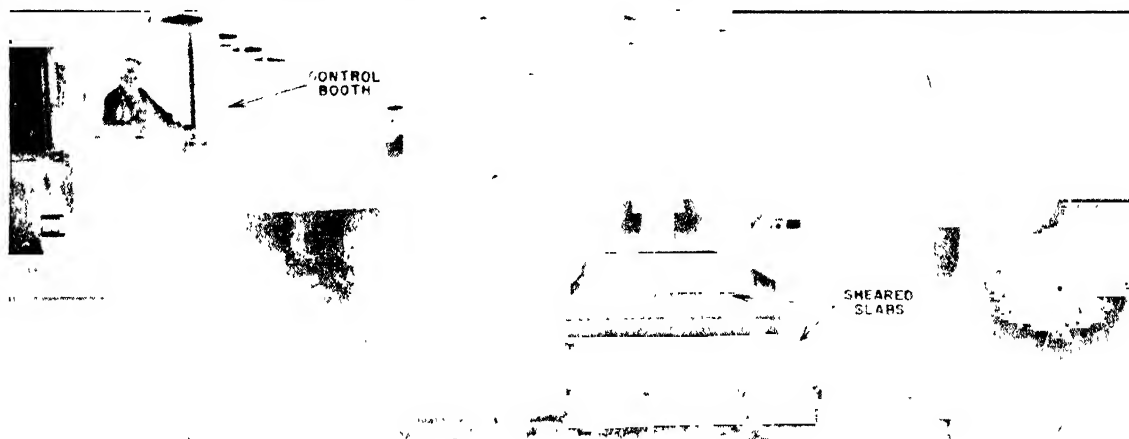


FIG. 25-12. Slab piler of the 45-inch universal slabbing mill in the Homestead District Works, receiving slabs by roller table from the shear. A section of roller table is lowered in increments as each slab increases the height of the pile. When the pile height reaches the maximum limit for lowering the table section, the rollers are operated to move the pile onto a transfer car, shown in the foreground.

connected finishing mill. The 43-inch three-high blooming mill at Ohio Works of United States Steel, which uses an ingot of medium size in a large tonnage operation, is an exceptional use of the three-high type.

Operating Units Comprising a Blooming Mill—To avoid frequent digression, the physical layout will be described here for rolling ingots on a two-high reversing mill. Essentially the same auxiliary equipment is used in rolling on other types of primary mills. The main parts of the rolling mill itself are shown diagrammatically in Figure 25-8. One modern mill is the 46-inch blooming mill at the Lorain Works of National Tube Division of United States Steel Corporation in Lorain, Ohio, already illustrated in Figure 25-3, whose layout is shown diagrammatically in Figure 25-30 in Section 2 of this chapter.

Two men are required to operate the controls of an electric-motor-driven two-high reversing mill, the roller and the manipulator, whose functions are defined by their titles; their pulpit, or control station, is placed on a bridge over one of the mill tables. (The steam-engine-driven mill requires a third man in the pulpit, the engineer, to operate the engine controls.)

Rolling—A pot car or ingot buggy (Figure 25-9) transfers a heated ingot from the soaking pits to the ingot receiving table, which delivers it to the mill approach table. The receiving table, in a few mills, is equipped with an ingot turner so the ingot may be rolled butt-end first (Figure 25-10). This practice aids the shearman in cutting the proper discard from the butt end of the rolled piece.

The mill approach table transports the ingot to the front mill table or roller table in preparation for rolling. It can move pieces at speeds of 200 to 400 feet per minute on modern mills. A few mills have a scale in the ingot receiving table or mill approach table for weighing the ingots before rolling (Figure 25-10).

During the complete rolling cycle, the hot steel is transported by reversible, live rollers in the mill tables. These rollers are subjected to high temperatures, heavy loads and impact. Depending upon the size of the mill they serve, rollers range in size from 12 to 21 inches in diameter, of whatever length is required for the particular service, and are spaced 2 to 3 feet apart. Forged steel generally is used in making the rollers which, in new mills, are mounted in roller bearings having a circulating lubricating system. Rollers on modern tables usually are driven electrically through a line shaft by miter gears; in some cases, they may be driven separately. As the pieces move along the tables, or are turned over by the manipulating equipment between passes, any loose scale falls between the rolls into a trough or depression beneath the tables and is removed to a settling pit with the aid of a stream of water.

Front and back roller tables alternately feed and receive the piece during each pass through the rolling mill, and mechanical units called manipulators rotate the piece through 90 degrees as required and move it from pass to pass. Some of the precautions to be observed in rolling are discussed later in this chapter.

Shear approach tables carry the rolled product from the back roller table of the mill to the shear. These tables have side guards that line the piece at right angles to the shear so that square cuts can be made. Some mills have a hot-scarfing machine in the mill delivery table or in the shear approach table for "skinning" the surface of the hot piece to remove, in part, some of the surface defects.

Shearing—The shear, generally called the crop shear, usually is located 100 to 200 feet from the rolls, and in line with the mill. Live rollers transport the rolled pieces

between the mill and the shear. Primary function of the crop shear is to remove from the rolled piece sufficient of the back and front ends (corresponding to the top and bottom of the ingot, or vice versa), so that the sheared piece remaining will meet chemical and metallurgical specifications. Secondary functions are to cut the remaining piece into desired semi-finished lengths and to shear test pieces.

Shears may be operated with electric or hydraulic power. A typical modern slab shear has an 80-inch wide knife, and a cutting-pressure capacity of 1200 tons; it can make straight cuts up to 8 inches thick and 60 inches wide, at a rate of 10 cuts per minute. It is driven by two 350-horsepower air-cooled motors. The top knife comes down to act as a gag and hold the piece while the bottom knife moves upward to make the cut. This principle of operation has caused this type of shear to be designated as a down-and-up-cut shear. The down-and-up-cut shear of a slab mill is shown in Figure 25-11.

The after shear tables which receive the cut product are designed according to the type of shear. Modern down-cut shears have a table that can move vertically to compensate for the action of the material during shearing. Either kind of shear (down-and-up-cut or down-cut) may have a shuttle motion for the after-shear table, which permits it to move back so the scrap may drop onto a conveyor beneath the shear. A crop pusher on the front side of the shear facilitates pushing the last crop end from the shear. Most after-shear tables have adjustable guards for lining up the last piece to insure a straight cut.

The shear gage measures the length of pieces to be cut. It is operated by a motor-driven screw. The gage head stops the bloom or slab at the desired point between the shear knives.

Sheared-off discards are moved by the crop conveyor to transportation facilities for return to the steelmaking departments where they are charged into the furnaces as scrap.

Located adjacent to and operating with the after shear tables are mechanical stamping devices or platforms for hand stampers to mark pieces for exact identification. A scale also usually is provided for weighing sheared pieces.

Transfers move sheared product from point to point, and may be of the continuous chain type with pusher dogs that engage and move the pieces over skid bars, or of the reciprocating-beam type which moves pieces progressively from one set of supports to another. Transfer buggies or cars also are used to move finished products from the runout table onto the pilers or transfer. Mechanical pilers separate the product into desired classifications as to weight, size of piece and type of cut, or other characteristics. Where the type of product permits, the sheared pieces may be moved by roller table from the shear to a piler of the type shown in Figure 25-12, from which the piles may be removed by a "C" hook on an overhead crane, or moved by rollers onto a transfer car.

COMBINATIONS OF CONVENTIONAL-TYPE MILLS FOR SPECIAL PURPOSES

Two Two-High Reversing Mills in Tandem—The most recent installation for the production of a wide range of sizes of square and round billets and narrow slabs at a high production rate is the No. 4 Blooming, Bar and Billet Mill at Lorain Works of National Tube Division of United States Steel Corporation. The 46-inch and 38-inch reversing mills forming part of this installation are described here and illustrated in Figures 25-3 and 25-13, respectively; the continuous billet mills com-

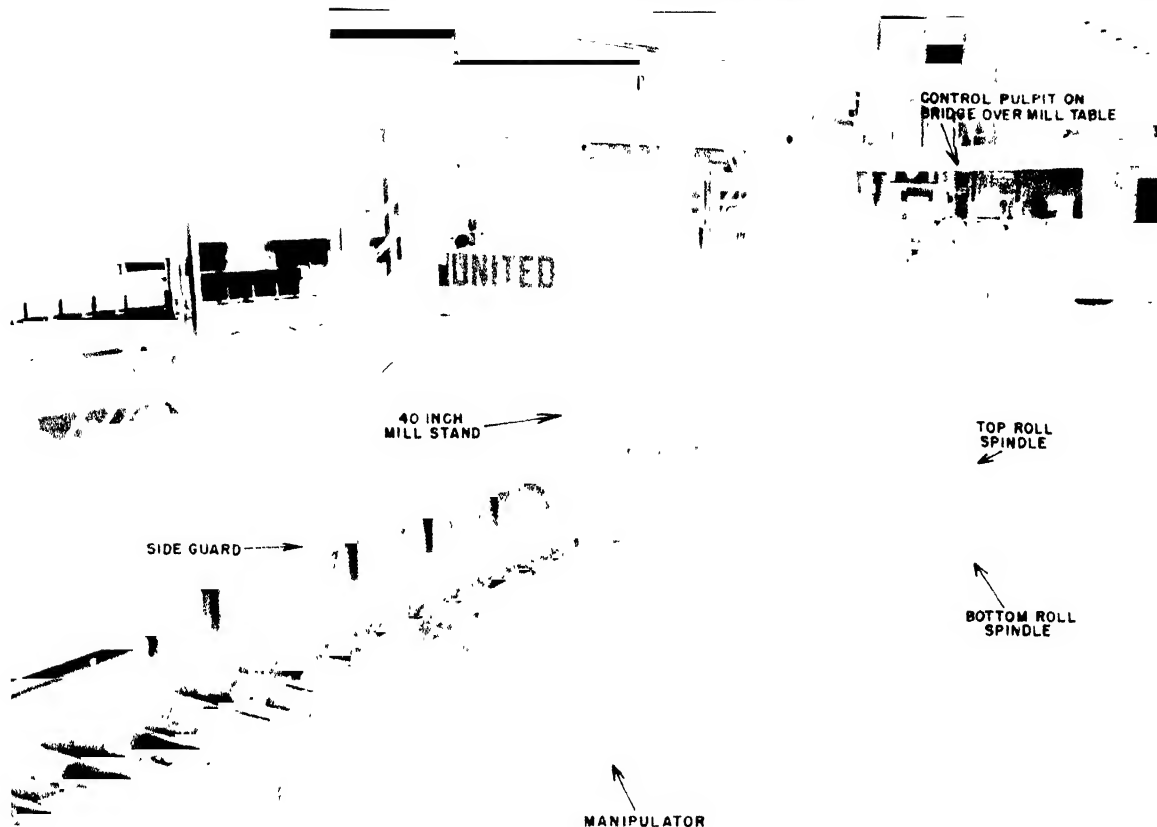


FIG. 25—13. The 38-inch two-high reversing blooming mill forming part of the rolling facilities in the No. 4 Blooming, Bar and Billet Mill in the Lorain Works. Operating in conjunction with the 46-inch mill shown in Figure 25—3, this mill has the position in the production line shown diagrammatically in Figure 25—30.

prising the remainder of the rolling equipment are described in the second section of this chapter. Layout of the combined facilities is shown diagrammatically in Figure 25—30, also in Section 2.

The layout is designed for extreme flexibility, inasmuch as the mill must roll many sizes of blooms, billets, rounds, and skelp slabs with a minimum number of roll changes, and adequate transfer equipment provides several outlets for these products. A battery of 24 modern soaking pits, each of which can accommodate, at one time, ten $32\frac{1}{2}$ by $32\frac{1}{2}$ by 94-inch ingots weighing 22,890 lb. each, constitutes the heating facilities. Ingots from the soaking pits are conveyed to the blooming-mill receiving table by either one of two ingot buggies. A scale for weighing ingots and a turntable for turning them 180 degrees have been provided at the receiving table.

The 46-inch by 100-inch two-high reversing blooming mill is equipped with front and back tables, individually driven feed rollers, double-rack-type manipulators with tilting fingers on both sides, and hydraulic roll changing rig. Mill rolls are equipped with oil-film bearings which are believed to have been the first application of this type of bearing on any blooming mill. The top roll is hydraulically counterbalanced and is positioned by a two-motor screw-down equipped with roller bearings and both mechanical and Selsyn-type roll-opening indicators. Two 15-inch diameter, 2-inch pitch screws provide a maximum roll lift of 45 inches at the rate of 230

to 460 inches per minute. Mill rolls are driven directly through universal spindles and couplings by two 4000-horsepower, 40/80 r.p.m., reversing, direct-current motors. Collar diameter of the rolls is 46 inches, with a 100-inch body length. The roll body contains 34-inch, 20-inch, 15-inch, and 8-inch passes, $3\frac{7}{8}$ inches deep. This roll design avoids the common practice of working an ingot too much in an open bullhead pass, which has a tendency to sharpen corners. By confining the majority of reductions in grooves with large radii, blooms are comparatively free from corner cracks and seams.

This mill rolls the $32\frac{1}{2}$ by $32\frac{1}{2}$ -inch ingot to a 15-inch by 15-inch bloom in 17 passes. This size of bloom is the primary section delivered by the mill runout table to the shear, which is situated in line with the mill and 145 feet distant from it.

Shearing equipment consists of an approach table, a down-and-up-cut motor-driven shear of the start-and-stop type, shear-crop chute, gage, crop-end pusher and crop car, delivery table, bloom pusher, and transfer car and track.

After cropping, the 15-inch bloom is delivered to the 38-inch by 96-inch reversing mill (called locally a bar mill) situated in line and 206 feet from the shear. This is utilized as a secondary mill to further reduce the 15-inch bloom to 9-inch by 8-inch rectangular blooms or to various large tube rounds.

While normal operating practice calls for the 15-inch

bloom from the 46-inch mill to pass to the 38-inch mill for further reduction, the 46-inch mill is required at various intervals to roll smaller sizes down to 9-inch by 8-inch. These may be rolled, cropped, sheared to required length, and removed by transfer car to the bloom yard. These operations may continue while a 15-inch bloom is being rolled on the 38-inch mill to a 9-inch by 8-inch rectangular bloom or a large round, the 15-inch bloom having been received from the 46-inch mill. Or, in the event of a breakdown or roll change on the 38-inch mill (which normally supplies the six-stand continuous billet mill to be described in Section 2 of this chapter), the 9-inch by 8-inch bloom may be rolled on the 46-inch mill, cropped and sheared to required length, then transferred to an auxiliary roller table in line with the continuous mills, thus by-passing the 38-inch mill entirely (see Figure 25—30 of Section 2).

The 38-inch by 96-inch two-high reversing mill is similar in construction to the 46-inch mill, containing all its features with the following exceptions:

The mill is driven by one 5000-horsepower, 65/130 r.p.m., reversing, direct-current motor, through a 38-inch pinion stand.

Feed rollers were eliminated, these not being necessary due to the longer bloom lengths received from the 46-inch mill.

A double-carriage roll-changing rig, hydraulically operated, facilitates rapid roll changes.

Integral guide units, equipped with hydraulically operated releasing clamps and end-screw adjustment through the roll housings, are on both sides of the mill for each set of rolls, facilitating guide changing.

To obtain maximum rolling efficiency, the 38-inch mill rolls are at all times provided with a 9-inch by 8-inch pass to feed the six-stand continuous mill, along with necessary grooves for producing a specified large tube round. The mill thus alternates from bloom to round, or vice versa, according to production requirements. A typical provision of rolls for the 38-inch mill comprises four sets of 40-inch by 96-inch rolls, each to produce one of four order sizes of $8\frac{13}{32}$ -inch, $10\frac{11}{64}$ -inch, $10\frac{1}{4}$ -inch, and $12\frac{29}{64}$ -inch rounds, each set having the necessary grooves to roll the 15-inch square bloom from the 46-inch mill to either the before-mentioned rounds or 9-inch by 8-inch rectangular sections.

The foregoing example of operating requirements, and consideration of possible future sizes this mill may be called upon to roll, shows why it was necessary to provide rapid devices to minimize roll-changing time. All reductions take place in a specific groove, using the screwdown with Selsyn roll-opening indicator for obtaining the proper drafts in each groove. A predeter-

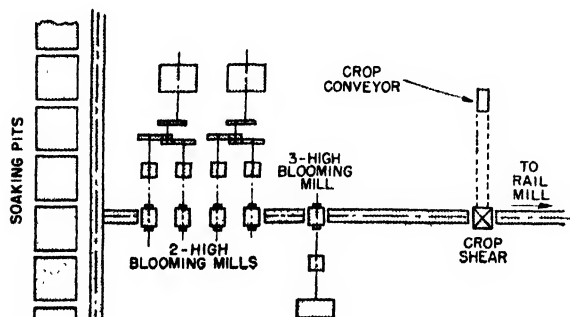


FIG. 25—14. Schematic arrangement of blooming mills in tandem, supplying blooms to the Gary Rail Mill.

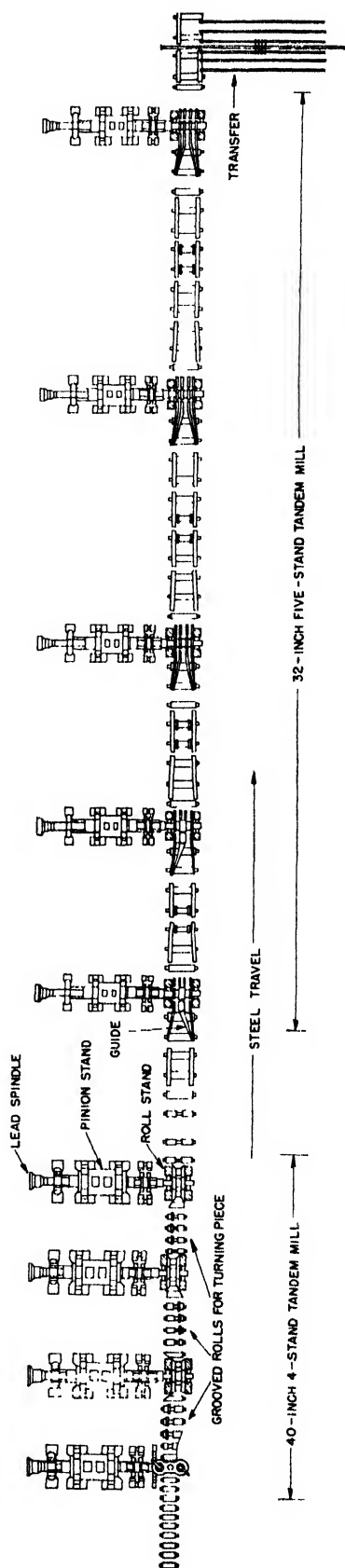


FIG. 25—15. Blooming-mill section of the Gary Billet Mill, the world's largest producer of billets, narrow slabs, and small blooms. This illustration is schematic only.

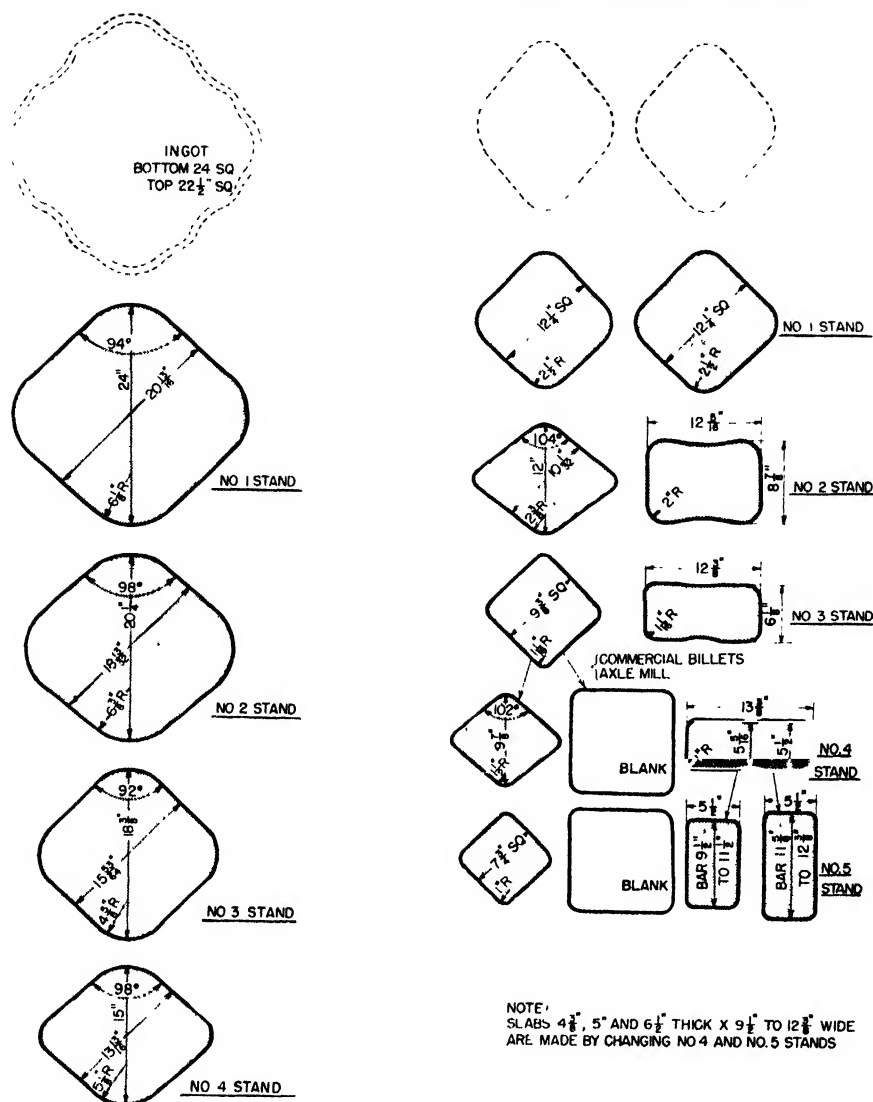


FIG. 25—16. (Left) Arrangement of roll passes in the four 40-inch stands and (right) arrangement of passes in the five 32-inch stands of the Gary Billet Mill, in a typical set-up for producing a variety of shapes from a single ingot size.

mined setting was established for the single pass through the finishing round groove.

Each set of rolls contains 15-inch, 12-inch, and 9-inch passes, plus an octagon leader pass and a round finishing pass, with sufficient collar widths to obtain guide-setting efficiency.

All large rounds are rolled direct in a straight flow line, thus: from ingot through the 46-inch mill, through the 38-inch mill, to saws for cropping and cutting to required lengths, and continuing on to shifting tables.

The shifting tables are equipped with kickout devices for delivering each individual cut to chain-conveyor cooling beds. Upon terminating their travel over the beds, the rounds are removed by an especially designed escapement device to counterweighted chains, which slowly lower them in groups to the cradles from which they are taken by overhead crane to storage bays for conditioning.

As previously mentioned, the 38-inch mill must also deliver 9-inch by 8-inch blooms to the six-stand mill for direct rolling to billets, smaller rounds and skelp slabs. These blooms are rolled from the 15-inch by 15-inch size in 9 passes and conveyed by roller line directly from the 38-inch mill, passing through a continuous

oxy-acetylene hot-scarfing machine situated 150 feet away, then continuing on to the transfer table. Blooms are transferred to the roller line directly preceding the six-stand mill, where they are cropped and divided by a saw.

If the 9-inch by 8-inch bloom is delivered from the 46-inch mill, it must by-pass the 38-inch mill. Therefore, the hot-scarfing machine is mounted on rails for convenience in shifting it to a position suitable for either type of operation. Following scarfing, cropping and dividing operations, the 9-inch by 8-inch bloom enters the six-stand mill.

Tandem and Three-High Mill in Tandem—Figure 25—14 shows the arrangement of blooming mills in tandem supplying blooms to the Gary rail mill. The installation comprises four 40-inch, non-reversing, two-high roll stands, in each of which the ingot receives a single pass, and a three-high mill in which five passes are given, the ingot being turned 90 degrees between passes on the latter mill. Stands Nos. 1 and 2 are driven at 5 r.p.m. by a 2000-horsepower motor, and Nos. 3 and 4 by another 2000-horsepower motor. A 6000-horsepower motor drives the three-high mill. This arrangement provides for reduction of a 23 1/4-inch by 23 1/4-inch ingot

to an approximately 8-inch by 8-inch bloom which, after cropping and cutting to lengths, is rolled directly, without reheating, on the rail mill.

Four-Stand and Five-Stand Tandem Mills in Tandem—The Gary billet mill is the largest tonnage producer of billets, narrow slabs, and small blooms. The blooming-mill section of this mill is described here to illustrate the extreme case of utilization of tandem mills.

The blooming mill, shown in Figure 25—15, consists of four 40-inch, two-high, non-reversing stands, followed in line by five similar 32-inch stands. The first two 40-inch stands are driven by one motor, with the third and fourth stands being driven by a single second motor. All five of the 32-inch stands are driven by a single motor through a line shaft and miter gears. The increasing distance between successive 32-inch stands allows for increase in length of the piece being rolled as it is reduced in each pass. No screw-down mechanism is required on these stands, because all of the passes are fixed. The mill is restricted to practically one ingot size, but can be set up to produce a variety of shapes, as indicated in Figure 25—16. Manipulation is effected by the proper design and application of table rollers, collars, and side guards.

DESIGN OF BLOOMING-SLABBING MILL ROLL STANDS

The design of primary rolling mills is based on the work to be done, the size of material to be handled, and the length, weight, and shape of the sections to be produced. It is necessary in all cases to have all parts of the roll stand designed to survive the severe service conditions imposed by the inherent nature of the operations performed.

Foundations for primary mill stands are proportioned to the size and weight of the mill, and are built to withstand any tendency to settle or distort. On modern units, foundations are anchored to bedrock by piling to increase their ability to withstand the shocks and blows common to primary rolling operations. In counterweight-balanced mills, such as that shown in the diagram represented by Figure 25—8, foundation design is complicated by the space needed for this mechanism. Design of foundations for two-high continuous mills is simplified by the type of housing and roll arrangement used. Three-high mill foundations must provide room under and adjacent to the mill for the mechanisms of lifting and manipulating devices that lift and turn pieces between passes.

Foundation design also must provide for the draining of water, oil, and greases to low spots, or sumps, from which they can be removed by pump or siphon if natural drainage cannot be provided. Passageways giving access to the equipment beneath the mill, sluice ways for handling scale that drops off pieces during rolling, and electric-cable tunnels also must be provided in the foundations of a modern mill.

Stand Design—The essential parts of rolling mills were discussed in Chapter 22, and need not be discussed again here for the particular case of the primary mills. The point should be made, however, that the component parts and auxiliary equipment for primary mills are extremely rugged and of very large size as compared to most other mills. Mention might be made of the fact that, in primary mills, two-high tandem and three-high mills generally are constructed with open-top housings, while two-high reversing mills, high-lift reversing mills, and universal mills are built with closed-top housings.

Front and back feed or housing rollers are required to move the ingot or piece between the mill tables and the horizontal rolls of the mill, because housings of two-

high reversing, high-lift two-high reversing, and universal mills are so bulky in design and the tables for these mills are laid out in such a way that there is an open space between the last roller of the mill tables and the horizontal rolls of the mills. (Post and table construction of two-high continuous and three-high mills make it possible in these cases only for these rollers to be eliminated from their design.) These feed rollers have separate electric motor drives in modern mills. They must be designed with heavy bodies and necks to withstand pressure and impact, and should have carefully designed, well lubricated bearings that are protected from scale and water.

In universal-type primary mills, one of the feed rollers must be omitted between the vertical rolls, since these rolls must move in and out with respect to the center of the table. This one roller is replaced by a short dummy roller. If a piece "stalls" on this dead roller, the vertical rolls must be run in to contact the piece to move it onto the live table rollers on either side of the vertical rolls.

ROLL DESIGN AND ROLLING PROCEDURES

Roll Design—Roll sizes for blooming-slabbing operations are determined by the type and size of mill and the product being rolled. In most two-high reversing mills, roll diameters are usually 2 to 5 inches less than the size of the mill (based on the pinion drive). Rolls for a two-high tandem mill usually are short and heavy, since usually only one pass is made in each roll stand and diamond passes predominate.

In most three-high mills, the train-line diameters of

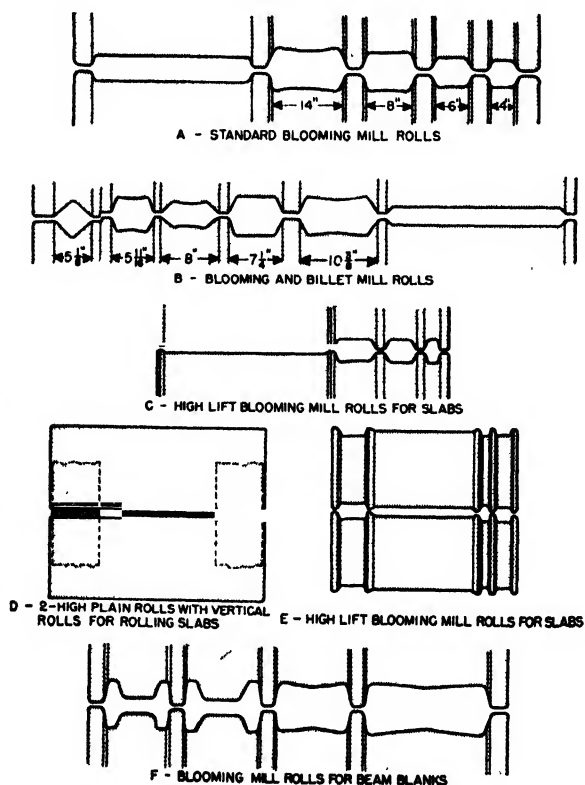


FIG. 25—17. Schematic representations of rolls for various types of blooming mills. In Sketches A, B, C and F, only part of each roll is shown to indicate the shapes of the pass openings. Sketches D and E show elevations of entire roll bodies.

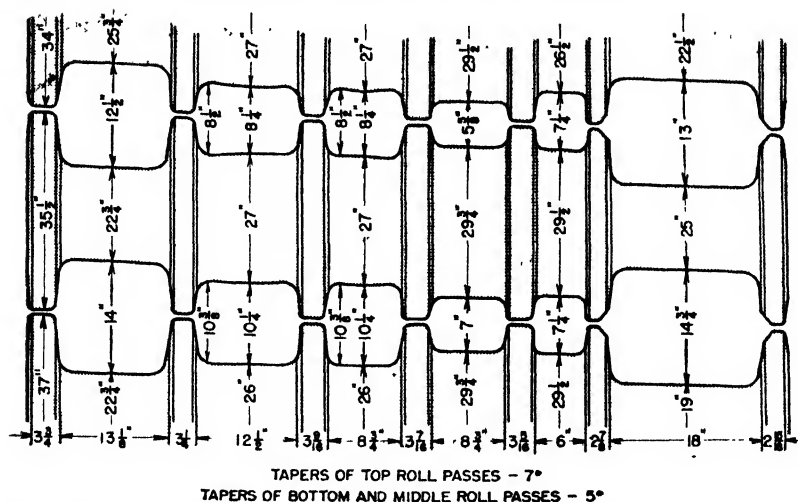


Fig. 25-18. Schematic representation of rolls for a three-high, single-purpose mill. Only part of each of the top and bottom rolls are shown, sufficient to indicate the shape and size of the pass openings. The diameters of the rolls in the various passes, however, are given, to fix the size of the top and bottom rolls.

the rolls are all different, the bottom roll being the largest, with the middle roll next in size and the top roll the smallest in diameter. These rolls are all of the same length body and neck.

Effect of Pass Design on Rolling Procedures—Drawings of rolls for various types of blooming mills are shown in Figures 25-17 and 25-18. Figure 25-17A shows rolls designed to produce a great number of sizes; this is typical of general practice. Since a large number of sizes must be produced on one set of rolls, it might be supposed that it would be well to roll on flat rolls without grooves. However, such procedure would produce blooms that were far from square. Grooves are provided, therefore, to produce the most popular sizes.

It is best practice to get the ingot into a box (grooved) pass as soon as possible in rolling, since any great amount of reduction of a piece without protection on the sides would cause the steel to crack. The spreading of the metal in these grooves causes wear of the rolls. Therefore, the sides of the grooves are tapered so that, with the dressing of the rolls, the pass may be made to approach its original width. The taper is made as great as possible for this purpose. However, if too much taper is allowed, the piece being rolled will have a tendency to turn down as it emerges from between the rolls. Tapers up to 15 degrees per side have been used effectively.

Figure 25-17B illustrates rolls designed for a more specific purpose, while allowing a certain degree of flexibility. The largest proportion of product made on this set of rolls consists of 4-inch by 4-inch square billets. To use these rolls, it is necessary to provide grooved table rollers for the diamond passes. This also restricts manipulation and, consequently, full use of the roll body.

Rolls for a three-high, single-purpose mill are shown in Figure 25-18. In this case, the ingot size, bloom size, and number of passes is fixed. On this type of mill, much heavier reductions are given in the early passes than would be used in general practice on the two-high mill using the rolls of Figure 25-17A. With the rolls of Figure 25-18, edging occurs after each two passes. Since the ingot is protected on all corners and spreading is somewhat suppressed, and edging is done so frequently, it is possible to roll with the heavy reductions required

to produce a bloom from an ingot in so few passes.

Figure 25-17F illustrates the pass arrangement for the production of shaped blooms called beam blanks. This type of roll, like the blooming-mill rolls of Figure 25-17A, usually lacks the necessary space for the most desirable arrangement of passes. Considerations involved in the design of rolls for beam blanks include: (1) reduction of the ingot to a smaller rectangle or square; (2) provision for an edging pass to aid in the reduction of an ingot; and (3) web and flange proportioning in the beam-blank pass. Available roll length usually limits the design to one pass for each of these three steps in the rolling operation. For the smaller sizes of blanks, it is possible to include a second shape pass in the rolls. Shape blanks to be rolled on universal mills producing beams are made in this type of rolls. For the larger beam sizes, a beam-shaped ingot is used. This permits a more uniform reduction that results in a better flange build-up in rolling.

Figures 25-17C, D, and E show three sets of high-lift blooming-mill rolls. While the method of employing vertical and horizontal rolls is fastest for producing slabs, use of the horizontal rolls for edging, as in Figures 25-17C and D, is advantageous from the standpoint of surface of the semi-finished product of the mill. The edging pass, using horizontal rolls with the piece resting on its narrow edge, cracks the scale from the wide faces of the slab (which are in the vertical position) permitting the scale to fall off the piece and between the table rollers into the sluice-way beneath the mill. With the flat-rolling method using both vertical and horizontal rolls, the piece is not turned and some means must be employed to blow off the cracked scale; various ways are used, which will be discussed later.

Convexity of Passes—The convex shape (belly) of some passes will be noted in the illustrations; this convexity serves a double purpose. One is to prevent the formation of fins. If a straight-sided piece is edged in a succeeding pass, spreading causes the metal to squeeze out between the collars of the roll, as shown by the dotted lines in Figure 25-19A. Convexity of the passes produces a shape with concave sides, allowing considerably more spread in succeeding passes before a fin can occur. Fins, when a piece is turned 90 degrees then rolled in a succeeding pass, are rolled over and produce

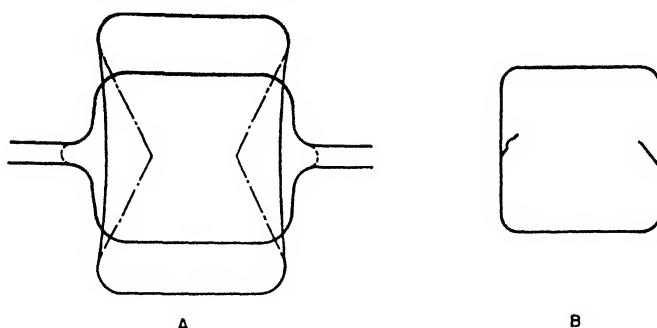


FIG. 25-19. (Left) Schematic representation of the effect of concavity of ingot sides in preventing the formation of fins. The large round-cornered shape is that of the piece entering the pass; the smaller shape represents the cross-section of the piece in the pass, with dotted lines indicating fins formed in the openings between the rolls if the ingot is straight-sided or has insufficient concavity. The dot-dash lines indicate extreme concavity which, as indicated at right, also will cause laps during rolling.

what is known as a lap, and no amount of further rolling can eliminate this defect. It is impossible for fins to roll back into the parent metal because an oxide forms on them which prevents clean metal-to-metal contact between the fin and the parent metal that would be essential to welding the two integrally together. Laps (or seams) also may result if the sides of a piece are excessively concave (Figure 25-19). Laps must be chipped or scarfed out before the bloom or billet can be rolled on finishing mills for further conversion.

The permissible amount of convexity is dependent upon the draft. It might seem that the convexity should be made very great to eliminate all possibility of fin formation. However, if the convexity is made too great, as illustrated by the dot-dash lines of Figure 25-19A, the reduction, in penetrating to the center, will not be able to push out the metal in the center and folding will occur that ultimately will form laps, as indicated in Figure 25-19B.

It will be seen from Figure 25-17F, showing the beam-blank roll set, that the box passes have a deep convexity; this is desirable in this case because of the shaping effect which is helpful in producing the shaped bloom.

The second purpose of convexity is to provide a convenient place for the ragging which is put on rolls for primary mills to increase their "bite."

Depth of Passes—The depth of any pass other than the bullhead pass in blooming-mill rolls is governed by two factors: (1) the maximum draft to be taken on any piece rolled in that pass, and (2) the minimum thickness to be produced in that pass. The relationship between the depth of any pass and the bullhead or barrel-pass diameter must be such that the bottom of the bullhead pass is not elevated too high above the table rollers. If the bottom of the bullhead or barrel pass is too high above the table rollers, there would be excessive impact from the piece each time it passed through the mill and was discharged onto the rollers. Passes should be made as deep as possible, consistent with these considerations, however, because deep coverage of the bloom helps to guide the piece and also aids in preventing the formation of fins.

High-lift reversing mills require rolls of extraordinary length; most mills of this type have rolls 100 to 120 inches in body length. This is necessary to provide sufficient width for the grooved edging passes, along with a barrel or bullhead pass at least 6 inches in excess of the maximum width of slab to be rolled. Mills of this

type may operate in conjunction with a separate vertical roll stand having relatively short, heavy rolls, located at the end of the mill runout table, to give a square edge to the finished product.

Universal-mill roll sizes are dependent directly upon the width of material to be rolled, along with the desired product size. In a typical mill, the bodies of the horizontal rolls are 45 inches in diameter and 80 inches in length.

Ragging, or grooves cut into blooming-mill rolls to prevent slippage when using heavy drafts, should be shallow and parallel to the axis of the rolls. Otherwise, it may prove injurious to the surface of the semi-finished steel by causing slivers and laps. Sometimes, ragging will be designed to protrude from the roll, instead of consisting of grooves cut into it. In general, the less ragging the better.

Bearings—Roll-neck bearings of primary mills are, for the most part, babbitted with brass-grid inserts, although many mills are equipped with composition bearings of the laminated-fabric phenolic types. At least one mill uses roller bearings, which represent the latest development. Automatic lubrication of the babbitted bearings adds considerably to their service life, as compared to bearings hand-packed with lubricant. Water is used to lubricate the composition bearings, and care must be exercised to prevent grit from being carried into the bearing with the water.

Roll-Opening Indicators—On mills where the distance between the rolls is varied at the will of the operator to suit the particular rolling operation, some means must be provided for indicating visually the position of (i.e., the distance or opening between) the rolls. The conventional type of indicator is a large, calibrated dial with clock-like hands. The shaft that drives the hands is connected mechanically by rods, gears and cables to some rotating part or shaft on the screw-down mechanism, so that the position of the hands on the dial changes in unison with the movement of the rolls. The dial usually is mounted at the top of the mill, where it can be seen from the pulpit.

More recently, an electrically-operated indicator has been used which utilizes two small, specially-wound (Selsyn) motors. One is connected mechanically to the high-speed shaft of the screw-down and acts as a generator. The current generated by this motor (called the transmitter) is conveyed by wires to the second motor (the receiver) and causes it to rotate in unison with the transmitter. The receiver is connected mechanically to

the shaft driving the hands of an indicator dial which is located in the operators' pulpit.

Roll-Changing Devices—Rolls in open-top housings are changed by removing the housing cap and picking out the rolls separately or collectively with crane hooks.

With closed-top housings, roll changing is effected through a window of the housing by a counterweighted porter bar, by a sleeve using the ingoing roll as a counterweight, or by a "C" hook. Any of these methods requires the use of an overhead crane to remove the rolls one at a time. The rolls are pulled out of the housing by attaching a socket fitted over the protruding roll necks.

In modern mills, a roll-changing rig removes worn rolls and installs new ones. It is set in a permanent mounting, level with the housing sill, and consists of a motor-operated crosshead mounted on a rail frame and driven by a rack and pinion. It withdraws both top and bottom rolls, with their respective bearings, one at a time. The old set-up then is removed from the rig, on which the new set-up is placed and positioned in the mill housings.

Cooling Water—During operation of a blooming or slabbing mill, a liberal supply of cooling water should be distributed carefully and uniformly over the rolls. If possible, the rolls always should be warm, never chilled, since the sudden contact of the surface of a hot ingot with a cold roll will cause too rapid a differential expansion of the roll surface, and cracks called fire cracks may develop. Water should be turned off when the mill is not rolling. If water is kept flowing when the mill is not rolling, the rolls should be kept turning to avoid uneven cooling which is one of the most common sources of cracks in rolls. Rolling without water at first and then putting water on the heated rolls is also a cause of cracking.

In some mills, great care is taken to warm up rolls

before they are used in primary rolling operations. In one plant, the rolls are warmed by steam coils in a special pit before use. This practice minimizes fire cracking and roll breakage.

Manipulators—The rate of rolling steel in the blooming or slabbing mill depends upon the auxiliary equipment to a marked degree, as well as on the speed of the rolls. The importance of the auxiliaries is demonstrated by consideration of the fact that, in normal good rolling practice, a piece being rolled is in actual contact with the rolls only 25 to 40 per cent of the total rolling time. The remaining time is consumed in handling or manipulating the piece, emphasizing the importance of rugged, efficient equipment for performing this work. Transfer of the work from one point to another and away from the mill stand and between the different work stations is performed by the various live-roller arrangements discussed earlier.

Turning of the piece between passes, moving it sideways on the roller tables from one pass to another and, when necessary, straightening it, is the function of the manipulators. Usually, they are built to have both horizontal and vertical motion and, in most mills, are operated by electric or hydraulic power.

Location of blooming-mill and slabbing-mill manipulators depends upon the type of mill. In most three-high mills, the manipulators are located under the table, on the entering side of the rolls, the fingers coming up between the rollers to engage the piece as the table is lowered. The principle of operation is shown in Figure 25-22, as applied to a three-high billet mill. In other mills, they are located on side guards, stroking laterally over the tables on one or both sides of the mill. These side guards are equipped with retractable manipulator fingers or arms, which have a vertical or near vertical stroke and serve to lift the piece by a corner in the process of turning it 90 degrees.

SECTION 2

PRODUCTION OF BILLETS

Development of the Billet Mill—It already has been described how, as the technique of pouring larger and larger ingots progressed, it became necessary to adopt the use of blooming mills to roll blooms of sizes which could be handled by separate finishing mills, either directly, or after reheating. A further step in the trend became necessary as ingot sizes increased still further, in that many reversing blooming mills are capable of producing satisfactory billets in sizes down to 4 inches by 4 inches, but this size is still too large for rolling a great part of the products rolled in finishing mills. About half the steel now produced is rolled into material of small section, and to finish these products with one heating, mills rolling these sections start with small billets. Hence, an intermediate mill between the blooming and finishing mills is required for rolling billets.

Blooms from 6 inches by 6 inches to 10 inches by 10 inches usually are taken to the billet mill from the blooming mill directly without reheating. The ends of the bloom are cropped, and it is often necessary to cut the remainder of the bloom into two or more pieces, as will be explained later. Hot scarfing also may be performed at this time. These operations must be performed rapidly to retain as much heat as possible in the bloom to keep its temperature high enough for good rolling in the billet mill. In this connection, billet mills capable of handling large blooms not only save time on

the blooming mill, but also receive hotter steel, because the smaller surface area per unit of weight exposed by pieces of heavy cross-section lessens heat losses.

Although most billets are rolled directly from blooms without reheating, there are some plants that provide reheating furnaces between the blooming and billet mills (Figure 25-24). The older blooming and billet mills generally did not provide for reheating since the grades of steel then produced could be rolled through a greater temperature range without harmful effects. Since the more critical grades of steel, with their restricted range of rolling temperatures, have become more widely used, some of the newer primary-mill installations provide reheating facilities between the blooming mill and the billet mill. These mills are arranged sometimes so that the ordinary grades of steel can be sent direct to the billet mill without reheating.

Since the billet section is a simple one and the requirements in the way of accuracy as to finish and dimensions of section are not exacting, the first requirement of the billet mill is that it be heavy enough to handle fairly large blooms, and speedy enough to reduce the piece to the desired size before it becomes too cold. Most billets, however, must be straight, square, and free from surface defects. Billets that are twisted, bent, or not square, will not charge into or push through reheating furnaces properly, causing pile-ups in the furnace; this is due to the fact that billets lie on the hearth

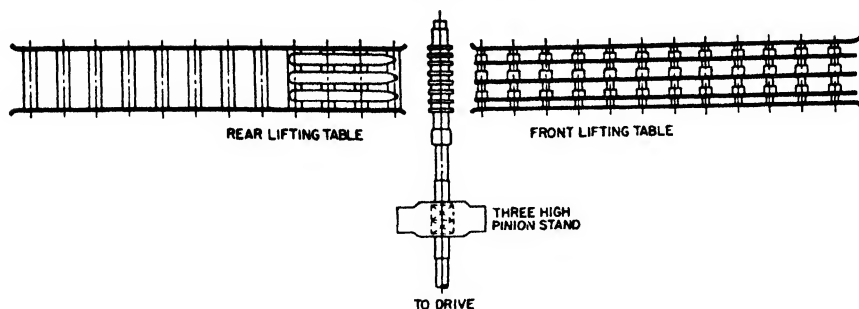


Fig. 25-20. Schematic layout for a three-high billet mill, using the rolls shown mounted in their housings in Figure 25-21.

side by side, and depend, for their movement through the furnace, on pushing against the exposed side of the last billet charged to move all the billets forward through the heating chamber. Poor surface causes excessive conditioning or even losses due to rejections.

In this section of the chapter, only the rolling of billets will be discussed. The rolling of sheet bar and skelp will not be considered, for, although these are both considered as semi-finished products, their production on billet mills has been abandoned almost completely in favor of using products of the continuous hot and cold strip mills as raw materials for the production of sheets, welded pipe, and tubes.

TYPES OF BILLET MILLS

Three-High Billet Mills—Billet mills may be of several types. One is the three-high mill with lifting or tilting tables. This type of mill consists of three rolls mounted one above the other in a single roll housing. Billets are rolled in one direction between the bottom and middle rolls, and through the middle and top rolls in the op-

posite direction. The lifting or tilting tables move the billet to the two different pass levels. Mills of this type have fixed drafts, and only a few limited sizes of billets can be rolled with a given combination of rolls. To produce various sizes of billets on this type of mill would necessitate numerous roll changes, which are costly.

Figures 25-20 and 25-21 show the layout for a three-high billet mill and the rolls used in the mill. It can be seen that alternate passes are used in the top and bottom rolls while every pass is used in the middle roll. On this mill, four rolls comprise a set, consisting of a top, bottom, and two identical middle rolls although only three rolls are used at one time. When the middle roll becomes worn to the extent that it must be replaced, the mill is changed; that is, the rolls are removed from the housing and the worn top roll is placed in the bottom position, the fourth roll or second middle is placed in the middle position and the worn bottom roll is placed in the top position. This presents a whole series of new passes and permits the same tonnage to be produced with the set of four rolls that otherwise would require two sets of three, or six rolls. Since the piece is rolled in one direction between the bottom and middle rolls

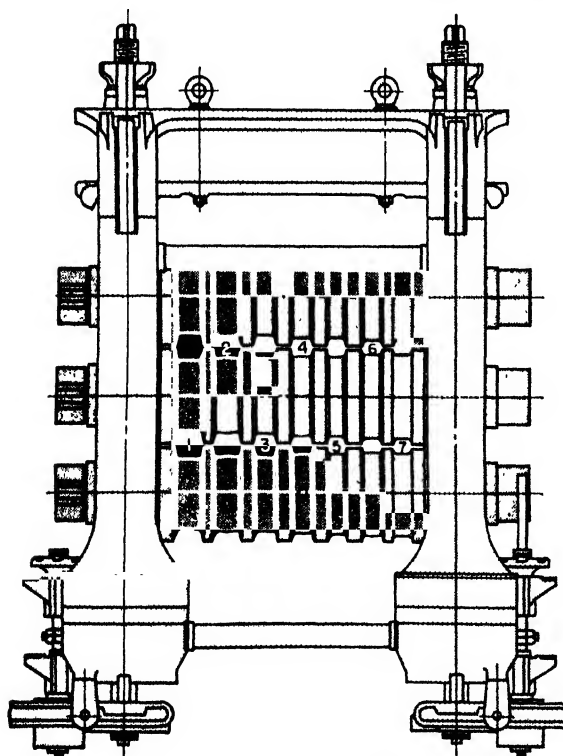


Fig. 25-21. The rolls (mounted in the housings) of a three-high billet mill. Numerals in the pass openings indicate the sequence of passes used in rolling (see also Figure 25-23).

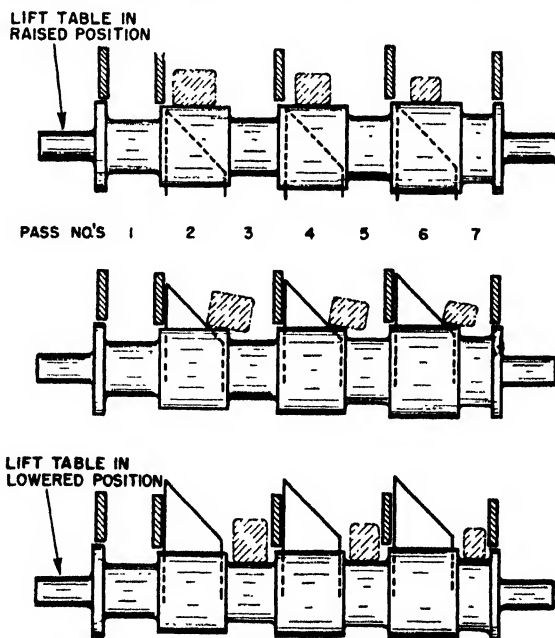


Fig. 25-22. Diagram showing the action of the fingers of a stationary-type manipulator for advancing the piece from pass to pass, while simultaneously turning it 90 degrees. The fingers remain stationary, and their action is performed by the motion imparted to the piece as the table is lowered from the raised position. The middle diagram represents the lift table in intermediate position.

and in the opposite direction between the middle and top rolls, the use of lifting or tilting tables is necessary to transfer the piece between the bottom and top pass lines. The table on the side of the mill from which the bloom enters (the side nearer the blooming mill) is called the front table, while that on the opposite side of the stand is called the rear or back table. Manipulators must be used in conjunction with these tables to turn the piece between passes and move it into line with the next pass. Both tables on a three-high mill are raised and lowered as a unit. The lift type table is more common and will be described along with only one of the many possible arrangements of manipulators.

Figure 25-22 shows diagrammatically a three-high mill lift table in raised, intermediate and lowered position. The front table contains twelve cast-steel rollers, each of which has five collars for turning the billets. These collars create four grooves extending from end to end of the table. The rollers are driven by an electric motor. There are side guards on the edges of the table and at the front end are side guards for putting the bloom into the proper pass. The front table is equipped with a stationary manipulator for advancing the billets from pass to pass which consists of fingers bolted to a pedestal on the foundation of the mill. These fingers are flat, cast-steel plates mounted vertically and with their tops shaped at an angle giving a 45-degree slope toward the outside of the mill. The fingers do not reach above the level of the roll passes when the table is elevated and the billets run out on the collars of the rolls. When the table is dropped, the billets encounter the stationary fingers and slide down into the grooves in position for the next pass. The rear table is operated through the same shaft as the front table but owing to the fact that it not only must raise the billets from the bottom pass line to the top pass line, but also must advance them one pass toward the outside, it has to travel through an arc in rising to bring the billet in line with the next pass. This is done by causing the table to slide toward the next pass as it is raised, by the use of pull-over rods attached to pedestals on the proper side of the scale pit. When lowered, the table slides back into place again. This table consists of twelve cast-steel rollers driven by a motor similar to the one used on the front table. Three heavy cast-steel side guards between the four passes that are used on the bottom roll divide the table into four grooves. There is a manipulator in the first groove that consists of five forged-steel fingers. The upward motion of the table draws the fingers with it and causes them to turn the piece and, as stated before, the sideward motion of the table advances the billet to the next pass. This manipulator lies below the table when the billet is delivered from the bottom roll and acts only

to turn the billets 90 degrees from the first to the second pass.

The mechanical operation of these tables leads to high maintenance for, when the piece drops into a groove, it causes severe shock to the bearings and other parts of the equipment, and when the tables are lowered, they may strike the stops with rather high speed. These severe service conditions make necessary considerable maintenance and repair work to maintain the tables in good operating condition.

Since the piece being rolled must be entirely out of the rolls for the lift table to transfer it between the top and bottom passes, its length must be limited to that which the tables will accommodate. This requires cutting the blooms from the blooming mill into the correct lengths so that when they are elongated, due to draft in the three-high mill, they can be handled properly on the tables. Cutting the blooms into pieces reduces the yield considerably as the ends of each piece usually are cropped after rolling.

The roll housings generally are cast steel, although some mills use cast-iron housings. The middle roll is not adjusted, but the top and bottom rolls are adjusted toward the middle. Cast-steel guides and side guards are held in guide cages or on rest bars which are bolted to the receiving and delivery sides of the housings. The pinions preferably are made with double helical or herringbone teeth. The three-high mill ordinarily is driven at a constant speed by a steam engine or electric motor. Methods of calculating power requirements for mills were discussed in Chapter 24.

The three-high billet mill shown in Figures 25-20 and 25-21 rolls four sizes of billets, which are reduced further in size on a smaller billet mill. As noted previously, the blooms are turned only once on the rear table and from one to three times on the front table, depending on the number of passes taken in the mill.

Figure 25-23 shows the shape of the piece out of each of these roll passes. The rolls used are cast alloy steel, 30 $\frac{3}{8}$ inches in diameter with a 76 $\frac{1}{2}$ -inch roll body. Box passes, as shown in Figure 25-21, are used generally on three-high mills for rolling billets. Slightly greater reduction can be taken using a diamond and square series but, since in passes of this shape the piece is rolled on the diagonal, the overall height of the piece is greater, requiring a deeper groove in the roll than for box passes. Such deeper grooves weaken the roll and increase the rolling or contact angle. Generally speaking, billets produced in box passes do not have as uniform diagonals as those rolled in diamond passes. If the box pass is narrow enough to restrict spreading, the action of the steel attempting to spread wears the sides of the pass and it rapidly becomes too wide. When the

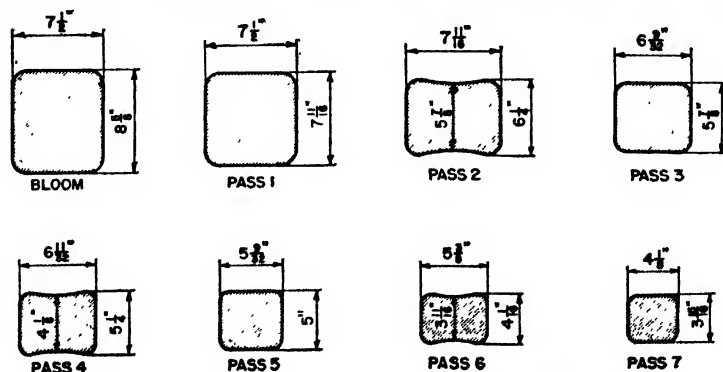


FIG. 25-23. Cross-sections of the pieces out of each pass when billets are rolled on the mill shown diagrammatically in Figures 25-20 and 25-21.

pass wears wide, the stock entering the pass will "float" to opposite sides in the top and bottom of the pass and, hence, produce a diamond billet. The stock cannot be adjusted very easily to control this condition since the three-high mill is a fixed-draft mill, and any adjustment for one pass affects all passes. Another reason for not using too heavy reductions on the three-high billet mill is that the rolling or contact angle must be small enough to assure the entry of the piece in the roll pass. When several pieces are rolled at once, the failure of one to enter the pass causes the others to be held up until it enters. This delay results in loss of temperature by the steel, with its resultant harmful effects; it also reduces output, since it can affect the production of the blooming mill as well as any mills following the three-high mill.

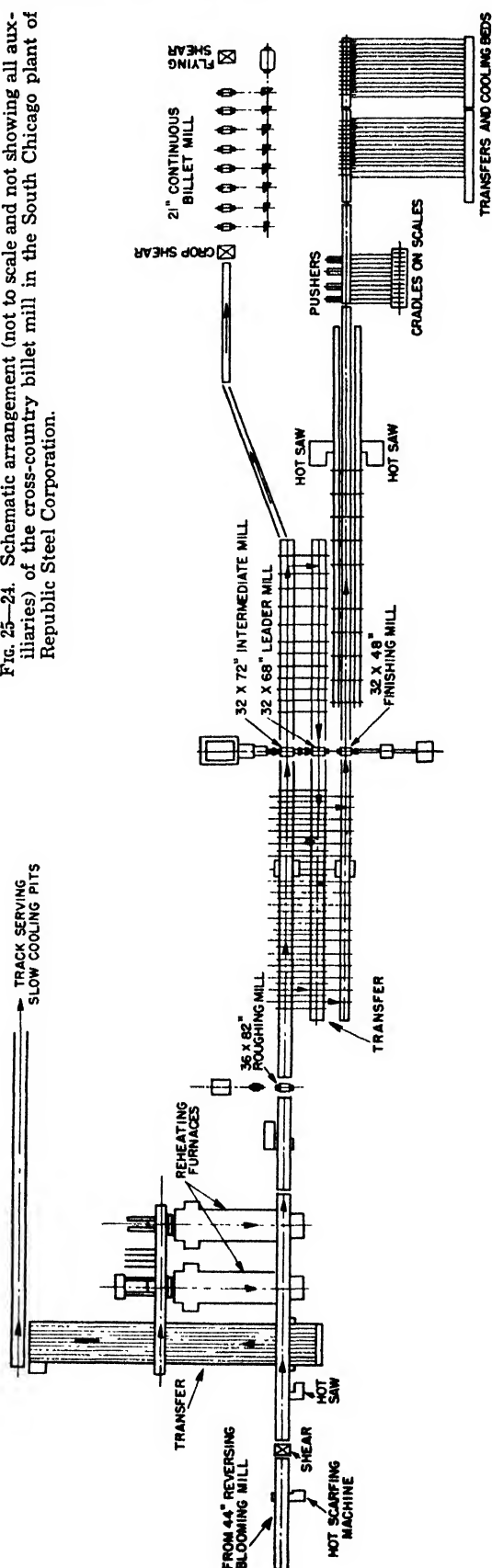
Billets from a three-high mill usually are cut by stationary shears or, in the larger sizes and for special purposes, by hot saws. This type of mill usually is used where only a few sizes of billets are produced, or in conjunction with smaller billet mills that further reduce the billets from the three-high mill.

Cross-Country Billet Mills—Another design of mill is the cross-country mill. This type of mill is composed of several stands of rolls, so arranged that the piece to be rolled is never in more than one stand at the same time. The roll stands may be placed side by side and the piece transferred to the various roll tables, the direction of rolling being reversed in each stand; they may be arranged with two or more stands rolling in one direction, with the piece transferred to roll tables and then rolled through several other stands in the opposite direction, and so on. This type of mill is faster than the three-high, but as in the three-high, the piece from the blooming mill must be cut into several lengths before entering the mill. The cross-country mill is much more flexible than the three-high billet mill in that, in order to roll a complete range of billet sizes, it not only can take the product of the blooming mill during a greater percentage of the operating time, but also can roll various sizes of billets with only one complement of rolls throughout the mill, with only a few minutes delay necessitated by changing guides for the various passes. Quite frequently, production on the blooming mill is not affected by roll changes on the cross-country mill, except when they occur in the first two stands. Some cross-country mills are so arranged that a single pass is taken in each stand while others are composed of a combination of single-pass and multiple-pass stands.

In contrast to the three-high mill, the finishing pass in the cross-country mill can be faster than the first passes, provided separate drives or gear ratios are used. Since the piece being rolled is only in one stand of rolls at a time on a cross-country mill, tables must be provided to carry the piece from stand to stand. Some mills are provided with diagonal tables to direct the piece from one train to another, while other mills provide transfers at the end of each line of stands for moving the piece over to the next train line. The roll passes of a cross-country mill can be any shape required for good rolling practice and quality of product. The piece can be turned as desired between stands by manipulators, guides, transfers, turn-up rolls, and other devices.

Advantages of Cross-Country Mills—The production of any mill is governed by the speed of the finishing pass and the percentage of the operating time during which it can be kept full. In the cross-country mill, the speed of the finishing pass can be greater than in the three-high mill, but pieces have to be spaced far enough apart in going through the mill so that each table is clear of one piece before receiving the following one. It should

FIG. 25-24. Schematic arrangement (not to scale and not showing all auxiliaries) of the cross-country billet mill in the South Chicago plant of Republic Steel Corporation.



be possible to keep the finishing pass of this type of mill full for more than half the time (as opposed to the three-high mill) and this, together with the higher speed possible, enables such a mill to reach higher production rates than the three-high mill.

Cross-country mills, where the piece can be turned at will after every pass, and particularly those mills where the direction of rolling is reversed after every pass, are admirably suited for the rolling of quality steels.

Figure 25—24 shows the layout of the cross-country billet mill at Republic Steel Corporation's plant at South Chicago, which is part of an integrated installation designed for the production of electric-furnace alloy steels. A 44-inch blooming mill supplies blooms to a 36-inch two-high reversing roughing mill, which in turn feeds three 32-inch two-high stands. The major part of production when this chapter was written was being rolled from hot-topped ingots, 25 inches square, weighing 12,000 pounds, and open-type steels from 23-inch by 25-inch ingots weighing 11,500 pounds. The 44-inch reversing mill produces blooms, for subsequent rolling into billets, bars, etc., in the following sizes: 7 by 7-inch, 8 by 8-inch, 9 by 9-inch, 10½ by 10½-inch, 12 by 12-inch, and 13 by 15-inch, depending on the finished size of the re-rolled product. It is driven by a 7000-horsepower motor at 40 to 100 r.p.m.

Delivery tables, transfers and reheating furnaces of the 44-inch blooming mill are so arranged that the blooms may be disposed of in any one of three ways. They may be sent without reheating directly to the 36-inch roughing mill and then on to the 32-inch and 21-inch mills; they may be kicked off on a transfer and then directed through one of two continuous reheating furnaces from which they are discharged onto the same table that leads directly from the 44-inch mill to the 36-inch mill, or they may be removed from the transfer table and placed in slow-cooling pits after which they are conditioned and shipped or are charged cold into the reheating furnaces and re-rolled on the billet mills.

The 36-inch two-high reversing mill is driven by a 50 to 120 r.p.m., 5000-horsepower motor. It is equipped with 36-inch by 82-inch rolls, laid out to use either 5 or 7 passes in roughing blooms down in preparation for rolling in the 32-inch mill. It is possible, however, to vary this practice by coordinating the work of this mill with that of the 44-inch mill, which is the practice usually followed. Manipulators and side guards on both sides of the 36-inch mill permit turning the blooms after each pass. Delivery tables and transfers are arranged so that, by reversing the 32-inch finishing stand (see Figure 25—24), squares from 5 to 9 inches and some narrow slabs may be finished on the 36-inch mill and sent to the saws, while the 32-inch mill is undergoing a complete roll change, thereby permitting uninterrupted production. The three 32-inch two-high stands are located 225 feet beyond the 36-inch two-high reversing roughing mill. The three stands are placed side by side. The first, or so-called intermediate stand, contains 32-inch by 72-inch rolls; the second, or leader, stand, has 32-inch by 68-inch rolls; and the third, or finishing, stand has 32-inch by 48-inch rolls. The three stands are spaced 14 feet, 6 inches, and 21 feet, 3 inches apart, respectively. All stands are electrically driven, are provided with electric screw-downs, and the roll necks rotate in composition bearings. The first two stands are driven by a single 5000-horsepower reversing motor, while the third stand is driven by a 2500-horsepower motor. The intermediate stand is equipped on either side with an electrically operated manipulator carrying tilting fingers. The leader and finishing stands each have one manipulator on the entry side, incorporat-

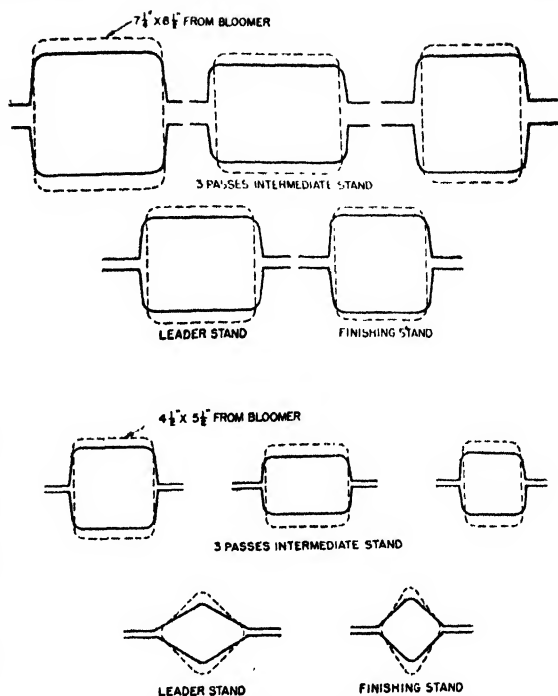


FIG. 25—25. Two typical sets of passes used in the mill shown diagrammatically in Figure 25—24. The five passes shown in the top set produce 5½-inch square billets. The lower set of five passes produce 3-inch square billets. In each pass, the dotted line shows the rolled shape which is received from the preceding pass. The solid line shows the new shape produced, which consequently appears dotted in the next pass.

ing a manually operated turning device. Blooms from the 36-inch mill are given one or three passes in the first or intermediate stand of the 32-inch mill, and one each in the leader and finishing stands, depending upon the finished size of the billet (Figure 25—25). The intermediate and leader stands, being coupled together with a common drive, are reversed together from pass to pass. The finishing stand, being separately driven, operates in only one direction. This mill rolls billets 3 inches to 7 inches square, and rounds 3 inches to 9½ inches in diameter, from blooms and billets varying in size from 4½ inches by 5½ inches to 10¼ by 11½ inches. Two tables, each containing a 60-inch hot saw with a 5-foot, 6-inch stroke, parallel the run-out table from the finishing stand of the 32-inch mill. The double saw arrangement enables cutting the billets to standard mill lengths without slowing up the mill.

It will be noted that a 21-inch continuous mill composed of eight horizontal stands is located beyond the first stand of the 32-inch cross-country mill. A 6¼-inch bloom from the first stand of the 32-inch mill can be rolled into square billets ranging in size from 4 by 4 inches to 2 by 2 inches on the 21-inch mill. The bloom is cropped before entering this latter mill, and the finished billets are cut to length by a steam-operated flying shear. The eight stands of the 21-inch mill are driven by a single electric motor through bevel gears from a main drive shaft, and all roll necks rotate in composition bearings. The billets are twisted between stands by twist guides and twist rolls. These devices are discussed in the ensuing section dealing with continuous billet mills.

It has been mentioned previously that some plants are

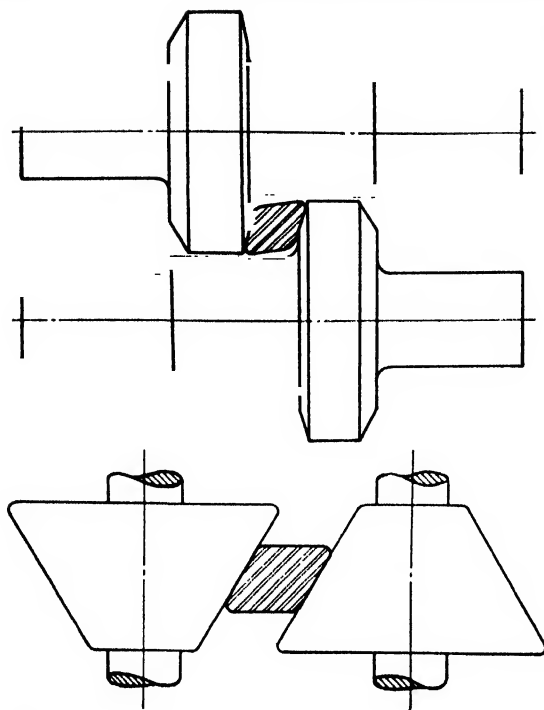


FIG. 25—26. Sketches illustrating two types of friction driven twist rolls, as used on a continuous billet mill to turn pieces 90 degrees between passes. Twist rolls such as those shown in the bottom diagram are called "cone-type" twist rolls.

equipped to reheat blooms between the blooming and billet mills. The above-described mill is so arranged, as shown in Figure 25—24.

Continuous Billet Mill—The continuous mill consists of a series of roll stands, arranged one after the other so that the piece to be rolled enters the first stand and travels through the mill, taking but one pass in each stand of rolls and emerging from the last set as a finished product. These stands may be all horizontal, and may include one or more vertical edgers, or the stands may be alternately horizontal and vertical. In the continuous mill, where the piece is being rolled in several different stands simultaneously, the peripheral roll speeds must be such that the elongation which occurs as a result of reduction in cross-sectional areas of the piece is taken care of by increasing the speed of each successive roll pass. On mills where all of the various stands are driven through gears by one motor or engine, the

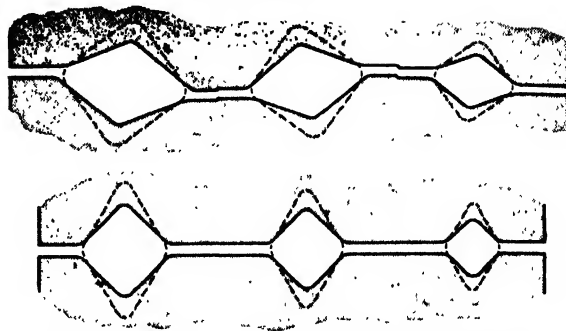


FIG. 25—27. Diagram of oblique roll passes designed to provide a twisting action on billets between stands when being rolled on a continuous mill. In each pass, the dotted line shows the rolled shape which is received from the preceding pass. The solid line shows the new shape produced, which consequently appears dotted in the next pass.

elongation is compensated for in the original design of the mill by choosing gear ratios that drive each set of rolls at a higher speed than the preceding set. The diagram of such a mill appears as part of Figure 25—24, where it is designated a 21-inch continuous billet mill. Any deviation from the originally designed elongation must be compensated for by varying roll diameters and thus changing peripheral speed. The rolls usually are installed in sets and are all dressed together. However, excessive wear on one set of rolls requires excessive dressing on the remaining rolls in the train. Various means have been adopted to gain longer roll life. Some plants will carry extra rolls for those stands that receive the most wear, and thus save excessive dressing on the remainder of the train, while others change the rolls around from stand to stand. As an example of the latter methods, a $2\frac{1}{2}$ -inch square in No. 4 stand can be changed to a 3-inch square without reducing roll diameter, and used on No. 2 stand.

The use of individual drives for each stand in a continuous mill is a great improvement over the single-drive type. With this drive, roll diameters need not be matched and the speeds of the individual motors can be regulated to give the correct peripheral speed for each set of rolls. The speed ratio on continuous mills must be maintained closely to prevent the piece from pushing or pulling between stands.

Since the piece being rolled in a continuous mill is in several stands simultaneously, the piece cannot be turned between passes in successive horizontal stands but, rather, it must be twisted. (The necessity for twisting, however, is eliminated in mills composed of alter-

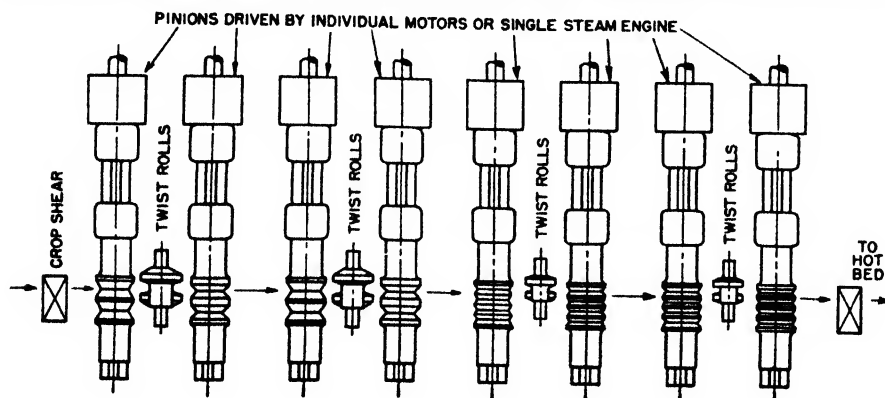


FIG. 25—28. Schematic arrangement of the roll stands and twist rolls for an eight-stand continuous billet mill.

nate horizontal and vertical stands.) Various methods have been devised for obtaining this twist, one of which incorporates the use of twist delivery guides. These guides are iron or steel castings, and are designed so that the piece is twisted gradually by the action of the guide which is contacted by two surfaces of the piece being rolled; the other surfaces have clearance in the guide. The twist in the guide is designed so that the piece being rolled is rotated to the correct degree for entry into the succeeding roll pass.

Another method uses friction-driven twist rolls, mounted in housings, between the various stands of work rolls. Some plants use twist delivery guides that are designed with a slight over-twist, to start the twisting action, in conjunction with twist rolls. The twist rolls relieve the twisting slightly, thus reducing the pressure on the twist guide and minimizing guide scratches on the product. Two different types of twist rolls are shown in Figure 25—26. A third method accomplishes the twist as shown in Figure 25—27. With this latter method, the passes are cut in the rolls obliquely, and the difference in diameters in the pass causes the twisting action. This method, however, requires the use of twist guides to insure the proper twist, since temperatures and the chemical compositions of steels being rolled are not always constant and can cause a variation in the degree of twist. This method of twisting causes excessive roll-pass wear.

High output is one of the chief advantages of the continuous mill. Scrap losses are low, due to the fact that blooms of any length can be rolled, making it unnecessary to cut the bloom after it leaves the blooming mill, except to discard pipe or any other flaws that might be present. Flying shears are placed after the finishing stand on a continuous billet mill, and are synchronized with the speed of the stand on which the billet is finished.

It should be stated, however, that sometimes breaks occur in the steel due to the twisting of the billet, and that when the billet is deflected from one pass line to another, it is more subject to scratching by the guides. These defects increase the amount of conditioning required by the steel. The continuous mill composed of alternate horizontal and vertical stands, which eliminate twisting and arranged so that it is never necessary to deflect the bar from one pass to another, produces a product with much better surface quality.

Figure 25—28 shows a schematic diagram for a continuous billet mill; Figure 25—29, the passes used on a 14-inch mill for rolling two different sizes of billets.

Six-Stand Continuous Mill at Lorain Works—This modern mill represents one of the major developments in rolling operations and embodies many features uncommon to billet mills. Its relationship to the other facilities of the mill of which it is a part is shown diagrammatically in Figure 25—30.

Designed to produce a wide variety of semi-finished products from 9-inch by 8-inch blooms, the mill is equipped to roll 4-inch square billets, 5½-inch by 4-inch, 7-inch by 4-inch, 8-inch by 4-inch, and 9-inch by 4-inch skelp slabs, 4¾-inch and 5½-inch tube rounds, on the same roll and guide set-up, with the ability to alternate quickly on any of the above sizes by push-button-operated controls. Additional skelp slabs up to 12 inches wide also may be rolled on the same set-up from 12-inch or 14-inch slabs received from either the 46-inch or 38-inch mills described in Section 1 of this chapter.

The mill consists of three vertical roll stands numbered Nos. 1, 3 and 5, and three horizontal roll stands numbered Nos. 2, 4 and 6, set in tandem on 10-foot centers (Figure 25—31). Each stand is powered individually by a 1750-horsepower, direct-current motor (300 to 600 r.p.m.), making a total of 10,500 horsepower for the six

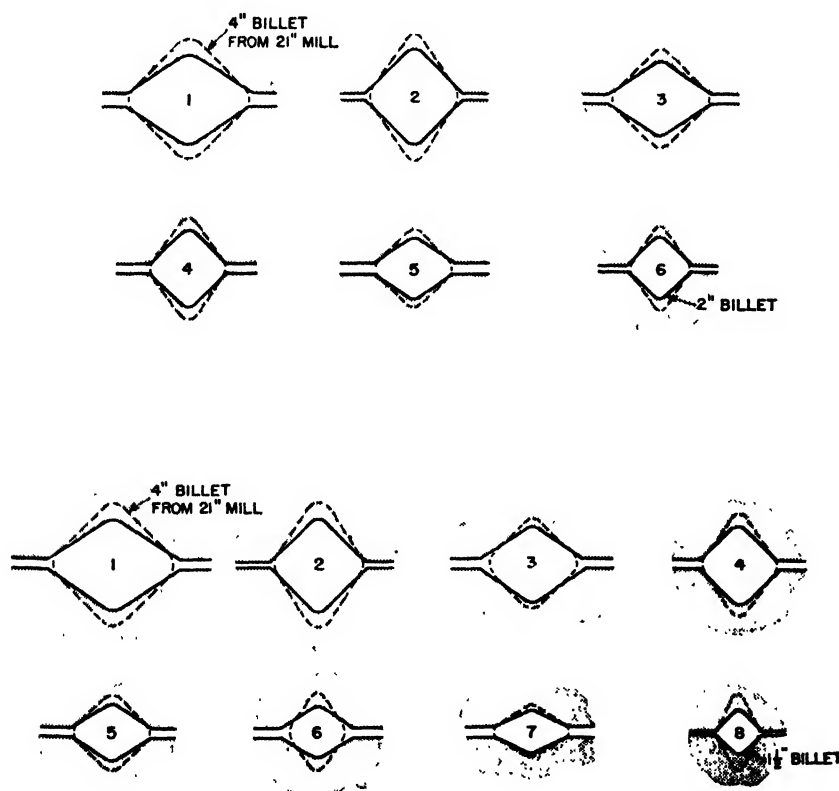


FIG. 25—29. (Above) Six successive roll passes employed in the production of 2-inch square billets from 4-inch square starting material on a 14-inch billet mill. (Below) Method of rolling 1½-inch square billets on the same mill, using the same starting material, in eight passes. In each numbered pass, the dotted line shows the rolled shape which is received from the preceding pass. The solid line shows the new shape produced, which consequently appears dotted in the next numbered pass.

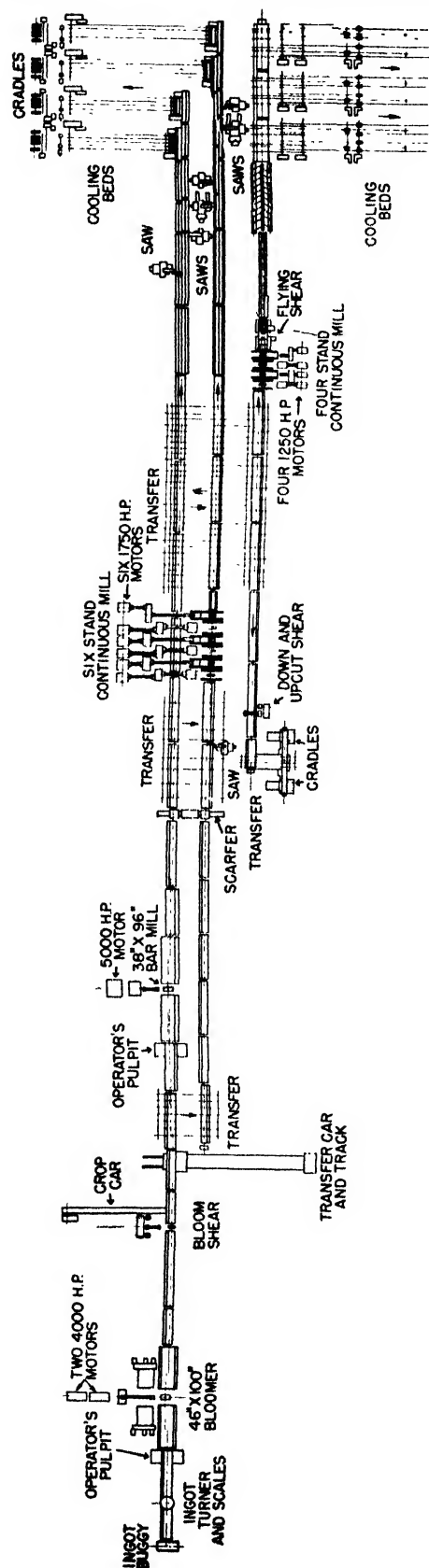


FIG. 25-30. Diagrammatic layout (not to scale) of the No. 4 Blooming, Bar and Billet Mill in the Lorain Works of National Tube Division. Flow of material through this mill is illustrated schematically in Figure 25-34 which also indicates vertical sizes mentioned.

stands combined. Suitable gear reduction sets provide for a finishing speed of 346 to 692 feet per minute at No. 6 stand. All vertical mills are of sufficiently rugged construction to enable drafting equal to the horizontal stands. The main vertical housing containing two 30-inch by 48-inch rolls is a self-contained, integral unit and resembles a horizontal stand laid over on its side. A fixed outer roll held securely by hydraulic pressure against the housing seat, and a movable inner roll hydraulically balanced against motor-driven screws, plus anti-friction roll-neck bearings and thrust units, comprises the roll-adjusting mechanism. Screw speed is 6 inches per minute, and screw travel is sufficient to permit operating the rolls with the distance between their centers set at any point between $35\frac{3}{4}$ and $27\frac{1}{4}$ inches.

Power for the vertical mill rolls is transmitted from the main gear drive (Figure 25-32) through a horizontal shaft to an intermediate bevel-gear drive, through a shaft inclined at 40 degrees to a combination bevel-gear drive and fully-enclosed 30-inch pinion stand. Universal couplings and spindles equipped with anti-friction thrust bearings, suspended and attached beneath the main mill housing, complete the drive. The main housing is enclosed within a secondary housing supported on the shoe plates. The secondary housing carries the screw-up mechanism and guiding slides for controlling the vertical movement of the main housing with respect to the horizontal-mill pass line. The motor-driven screw-up mechanism operates by push-button control at $10\frac{1}{2}$ inches per minute, and provides for a maximum vertical movement of the main housing of 36 inches. Thus, a set of vertical rolls may contain several passes within this range, each of which may be aligned properly with the horizontal mills. The male coupling end, transmitting motive power from 30-inch pinions through universal couplings to the rolls, is forged in one piece with the spline shaft that telescopes with the pinions. This arrangement permits maintaining the drive connection during upward movement of the rolls. The three vertical mills are of similar design, having all parts interchangeable, including intermediate bevel-gear drive. Rolls with complete bearings are changed in pairs, using double roll-changing hooks.

Vertical-mill roll and guide equipment is of such design that changes are unnecessary when rolling any of the previously mentioned sizes. The useful proportion of rolling time on the mill is thus increased, since it is necessary to change rolls only when worn or in the event of possible breakage. The entry guide unit is attached to the secondary housing in a fixed, pre-leveled position, to accommodate the upper pass in the rolls. The remaining passes may be utilized by adjusting the screw-up mechanism upwards, thus positioning the desired pass in front of the guide.

Delivery guides are set up within a swinging integral unit, including guides for all passes in the vertical-roll set. Once set in position, they seldom need adjustment, fixed-roll guides remaining stationary while the movable-roll guides are carried along with the roll during adjustment. Upon changing rolls, the entry guide retracts clear of the rolls by a hydraulically operated mechanism. Delivery guides are manually swung clear of the rolls. The rolling operations of the three vertical mills in conjunction with the three horizontal stands will be described subsequently.

The three two-high horizontal roll stands (Nos. 2, 4 and 6) resemble strip-mill stands in their construction features, having been designed for stability, smoothness of operation, accurate roll settings, and ease in roll changing. Housings are of cast-steel, closed-top construction. The top roll is hydraulically counter-balanced



FIG. 25—31. General view of the six-stand continuous billet mill in the Lorain Works of National Tube Division of United States Steel.

against motor-driven screws. The screw-down permits the screws to function independently or in unison. The bottom roll is fixed, with provision on the outward chuck for endwise adjustment. The top-roll outward chuck is fixed, while both drive-side chucks are free to "float" in the housing. The 28-inch diameter by 48-inch body length rolls are mounted on anti-friction bearings of the radial thrust type, all preassembled with fully-enclosed chucks. Rolls thus can be quickly changed by use of a "C" hook similar to those used in changing strip-mill rolls. Horizontal roll stands Nos. 2 and 4 each are driven independently through a double reduction-gear drive and a 27-inch pinion stand. No. 6 horizontal roll stand is driven through a single reduction-gear drive and 27-inch pinion stand. Universal couplings and spindles mounted on carriers connect the pinion stands with the rolls.

A feature essential to the flexibility of operation of this mill is the endwise movement provided for the horizontal roll and pinion stands. To utilize all the grooves provided in the horizontal rolls for various product sizes, provision was made for rapid alignment of horizontal-roll grooves with respect to vertical-roll grooves to maintain a straight line of product travel throughout the desired grooves of all six stands. This was accomplished by mounting the roll stand, spindle-carrier base, and pinion stand as a unit on the bed plate. By using the bed as sliding ways, the entire unit can be moved endwise horizontally 42 inches through power supplied by a 14-inch diameter hydraulic cylinder operating at 1250 pounds per square inch. To maintain the drive connection during such movement, the main drive shaft telescopes a spline-type flexible coupling at the pinion stand. Hydraulically operated quick-release clamps anchor the assembly to the bed plate, obviating the use of foot bolts.

As in the case of the vertical mills, the roll and guide equipment of the horizontal stands was designed to roll all of the previously mentioned sizes without roll or guide changes. Entry and delivery guides are mounted

as integral units on sliding rest bars which, in turn, are supported by the main rest bar attached securely to the roll housings. All main rest bars are adjustable vertically by jack screws mounted on the housings. Rolls must be changed after excessive wear occurs. To minimize roll-changing time and increase efficiency, all guide equipment is mounted on a retractable mechanism operated hydraulically, which makes it possible to move the guides clear of the housing window to permit rapid and simple removal or insertion of the roll-and-chuck assembly.

To thoroughly understand the functions of all the foregoing features on the six-stand mill, a discussion of a typical roll-pass arrangement and the sequence of operations is advisable. These are as follows:

Vertical roll set V-1 (Stand No. 1) consists of rolls 30 inches in diameter with a 48-inch body. It is provided with three duplicate 8 $\frac{1}{4}$ by 7-inch box passes, all of which are set up complete with the necessary guides. Either pass may be used at any time, the 7-inch dimension varying with finished bar size and adjusted by mechanical screws on the movable roll. In the event of a worn-out pass, the roller may, by push-button control, quickly raise or lower the entire main housing unit to the desired position so that an unworn pass can be utilized. A gage attached to the secondary housing post, and a pointer on the movable housing, indicate within easy view of the operator the proper level of the pass.

Horizontal roll set H-2 (Stand No. 2) has rolls 28 inches in diameter and 48 inches in body length. It has two duplicate passes, 7 $\frac{1}{4}$ by 6 $\frac{1}{4}$ inches, plus a 13-inch wide bullhead pass used when rolling skelp slabs. The normal setting as used for 4-inch by 4-inch, as well as 4 $\frac{3}{4}$ -inch and 5 $\frac{1}{2}$ -inch rounds, is 6 $\frac{1}{4}$ inches and is also subject to variation by use of the top roll mechanical screw-down. All three grooves are completely equipped with the necessary guides. The roller also may select any groove rapidly by operating the hydraulic valve controlling the mechanism which moves the entire roll-housing unit endwise. A gage on the housing shoes

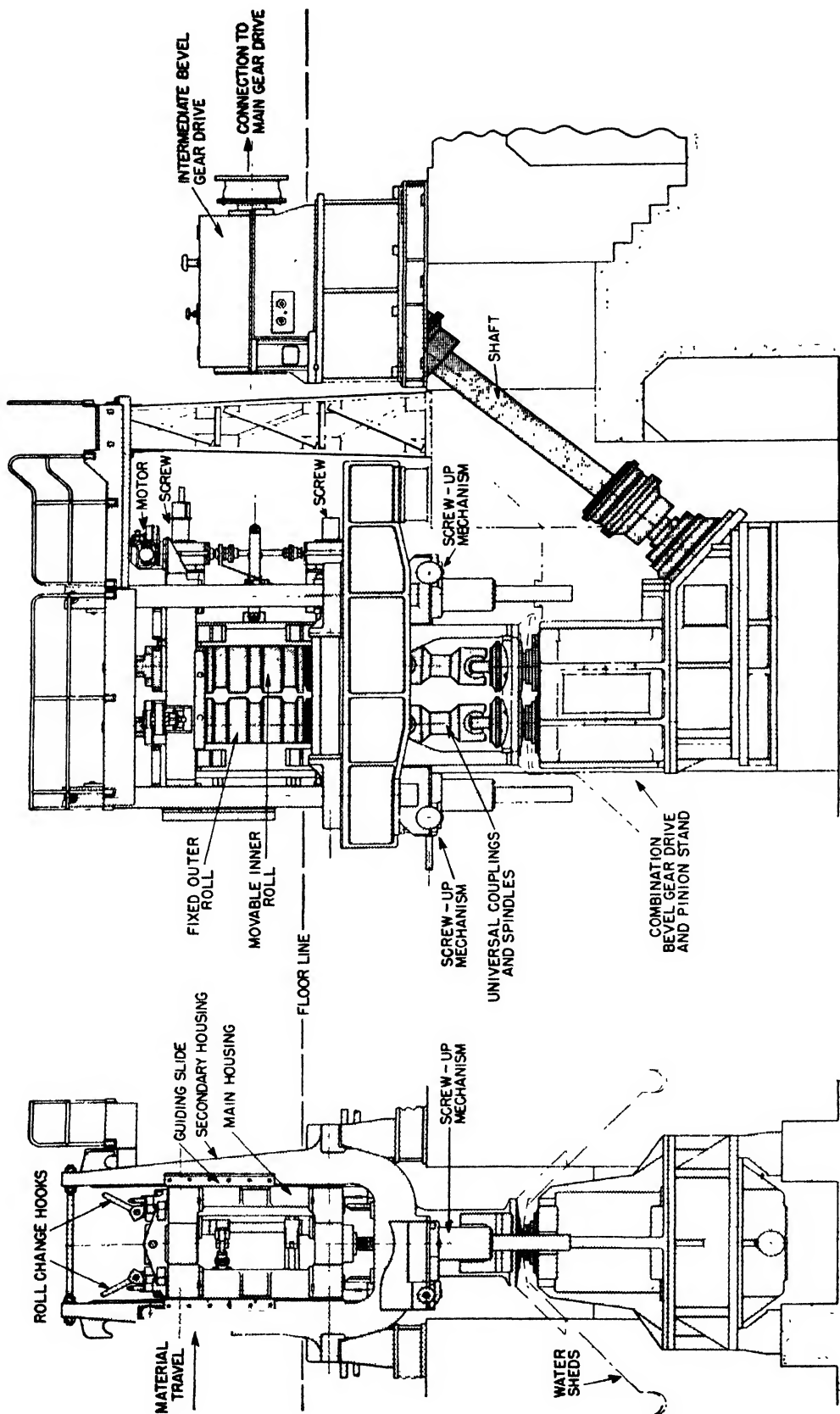


FIG. 25-32. Schematic arrangement of the principal parts of one of the vertical stands of the six-stand continuous billet mill in the Lorain Works of National Tube Division of United States Steel Corporation.



FIG. 25—33. General view of the four-stand continuous billet mill in the Lorain Works of National Tube Division of United States Steel.

indicates the proper position for the horizontal groove with respect to the vertical to assure straight line travel of the piece being rolled through the mill.

Vertical set V-3 (Stand No. 3), with rolls 30 inches in diameter and 48 inches in body length, also has three 6½ by 5-inch duplicate grooves, and functions in a manner similar to the V-1 set. The 5-inch setting is variable for finished product size.

Horizontal sets H-4 and H-6 have rolls 28 inches in diameter with 48-inch bodies. Each has its designated passes for 4-inch by 4-inch squares, and 4¾-inch and 5½-inch rounds, plus a skelp-slab pass, and have operating controls similar to H-2. Vertical set V-5, with rolls 30 inches in diameter and 48-inch body length, which is the leader pass, also has its designated passes. Two leader passes for the 4-inch by 4-inch square, and one each for the two sizes of rounds, have been provided in sets V-5 and H-6. Vertical set V-5 utilizes the 4-inch by 4-inch passes for edging skelp slabs, with operating controls similar to V-1 and V-3.

The run-out table from the six-stand mill delivers all product to transfer equipment designed to handle bar lengths up to 160 feet. Rounds may be delivered direct to either one of two saw lines, each equipped with stationary and movable saws for cropping and cutting to required lengths. Each of the saw lines converges to its own independent chain-conveyor cooling bed. Or, product may be transferred to the double saw line which normally handles product (rounds) from the

38-inch mill (described in Section 1 of this chapter), each line having a single saw with individual cooling bed. Thus, the finishing equipment for tube-round product from the 38-inch mill and the six-stand mill comprises altogether six saws, four saw lines, and four cooling beds. All products 4 inches by 4 inches to 12 by 4 inches delivered from the six-stand mill for skelp conversion are transferred to a roller line that carries them to shearing facilities and inspection and storage. For further reduction to smaller billets and tube rounds, 4-inch by 4-inch or 4½ by 4½-inch billets are delivered from the six-stand mill and transferred to a roller line directly preceding the four-stand continuous mill.

The Four-Stand Continuous Mill at Lorain—This mill (Figures 25—30 and 25—33) is similar in design and employs all the advanced features of the six-stand mill. It comprises four sets in tandem; V-1 vertical, H-2 horizontal, V-3 vertical, and H-4 horizontal. Each set is powered individually by 1250-horsepower, 400 to 800 r.p.m., direct-current motors, with reduction gear drives designed for a finishing speed of 430 to 860 feet per minute. All mill rolls are 20 inches in diameter, with body length of 36 inches, and to meet original requirements, were set up to produce 3¼ to 3¾-inch rounds and 2½-inch square billets, with sufficient space available for rolling other sizes. Billets delivered from the six-stand mill lie flat on the roller feed table and, prior to entering the No. 1 vertical (V-1) pass of the four-stand mill, are turned 45 degrees on the fly, by an air-

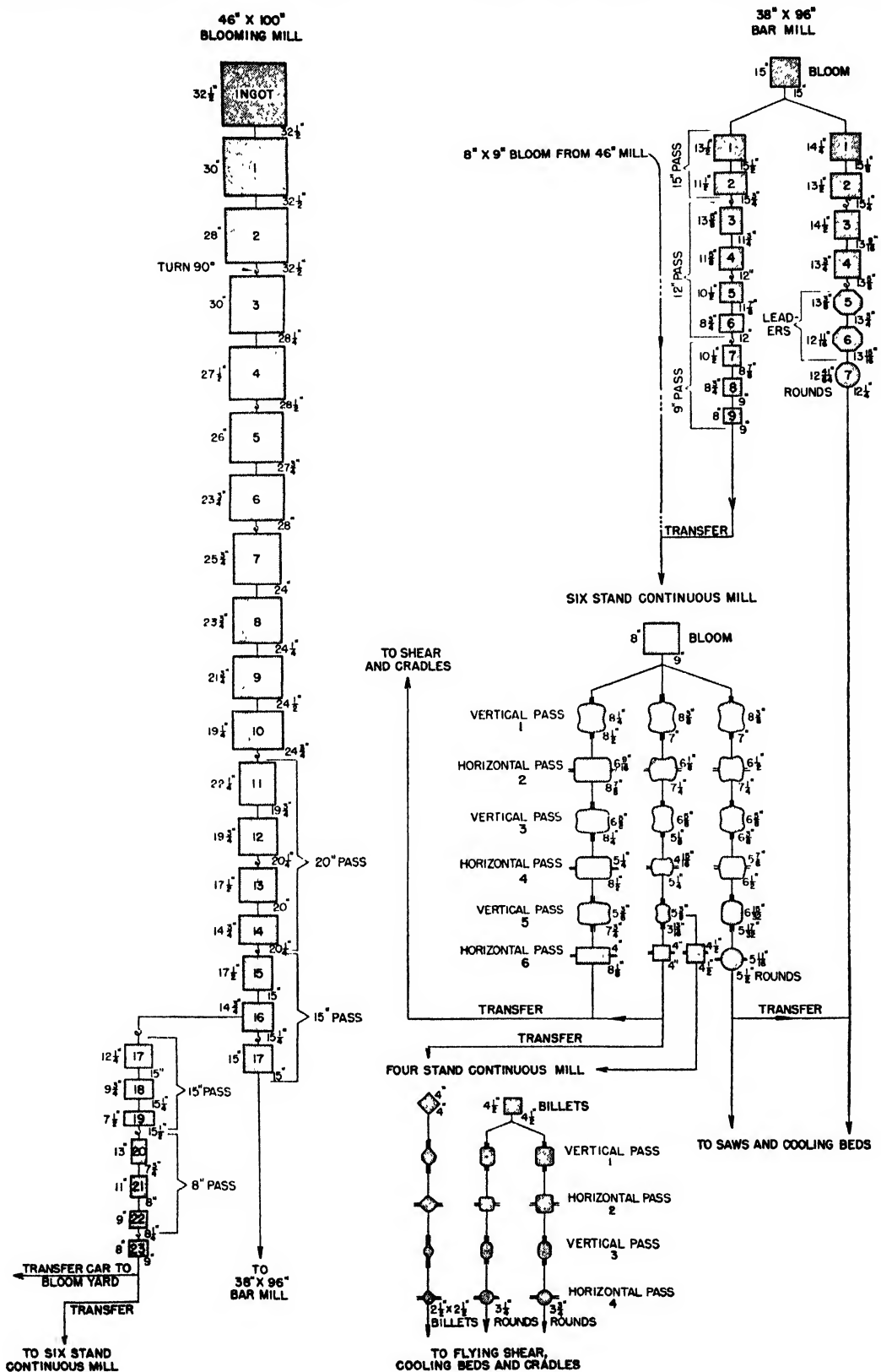


FIG. 25-34. Flow chart summarizing typical series of operations possible due to the extremely flexible arrangement of the modern No. 4 Blooming, Bar and Billet Mill in the Lorain Works of National Tube Division.

operated turning device, to permit finishing the 2½-inch square billets in diamond passes on their diagonals.

Billets of 4½-inch square size enter the V-1 pass on the flat for alternate flat and edge reduction to lead-oval and finished rounds. No. 4 horizontal set delivers billets to the flying shear for cropping and cutting to length. A specially designed double skew table with disappearing stops receives the product from the flying shear. Here the product is gaged, then conveyed in groups to the roller line opposite the cooling beds. The roller line has disappearing stops that position these groups opposite either one of the three adjacent cooling beds, over which the product travels to the cradles and storage bays. This completes the cycle of operations beginning with the 32½-inch ingot on the 46-inch blooming mill, as described in Section 1 of this chapter. Figure 25-34 is a flow chart summarizing a typical series of operations possible with this modern mill.

Hot Scarfing Machines—The increased demand for conditioning of semi-finished products (see Chapter 26) has led to the installation of hot-scarfing machines in many primary mills. Such machines may be placed immediately after the bloom shears or, when multiple units are fed from the same blooming mill, they can be installed after the roughing or the first billet mill.

The scarfing head of the machine contains oxy-acetylene burner nozzles and is designed to be adjusted rapidly to accommodate several different sizes of billets. The head also can be moved laterally to permit the scarfing of blooms or billets from different passes in a set of rolls, or to remove it entirely from the line to permit the passage of steels not to be scarfed. The machine desurfacing process can remove defects within ⅓ inch of the surface, which is usually sufficient to remove such defects as rolled seams, light scabs, checks, etc. It is not always economically advisable to cut at depths necessary to remove the deepest defects. General practice in those mills using the process is to desurface each grade of steel to the depth which strikes the right economic balance between loss in yield due to removal of metal and processing costs on one hand, and savings in lessened rejection loss on the other hand. This depth will vary with different materials, but the metal loss is generally 2 to 3 per cent of the product.

Roll Adjustment—There are several different methods for adjusting rolls on two-high mills (in addition to the one described earlier for three-high mills). One is a simple method of using a rider bearing on the bottom roll neck. The top-roll carrier bearing rests on liners on the bottom rider bearing. A wedge is placed between these liners to vary the distance between the rolls. Another method of adjustment consists of supporting the top roll by springs from the housing cap. Turning of the housing screw compresses the springs, and thus the space between the rolls is varied. Still another method of adjustment in which the top roll remains stationary and the bottom roll is moved up or down is used in a good many mills. This is accomplished in the following manner: a cast-steel screw box in which the threads have been babbitted to prevent excessive wear is placed on the sill of each housing. A screw bolt is placed in each of these, the outside one containing a left-hand thread while the inside one has a right-hand thread. The two screws are joined together by a cast coupling. Above the screw bolts, and resting upon them and the screw boxes, cast-steel wedges are mounted containing the same number of babbitted threads as the screw box. The bottom chocks are wedge-shaped and rest on the screw wedge. Since the two screws are coupled together, turning of the screw on the outside of the mill causes both screw wedges to move in or out and, due

to the wedge shape of the bottom chock, the bottom roll is raised or lowered. This method of adjustment is used on many continuous mills. One of the newer methods of adjusting rolls on billet mills makes use of an electric screw-down. With this type of adjustment, the top roll is supported by two hydraulic cylinders through steel-yard rods in each housing, reaching up to the top-roll carrier bearings. The top roll is held down by screws reaching through the cap to the breaker blocks on the rider bearings. An electric motor mounted on the housing cap actuates the two housing screws.

Shears—The product from the blooming mill must be cropped to discard pipe before entering the billet mill and the finished product from the billet mill must be cut to length. The bloom-crop shears, however, usually are considered part of the blooming mill rather than the billet mill. When the primary mill set-up consists of several billet mills fed one from the other, and all or part by the same blooming mill, crop shears usually are placed before each unit. Crop shears assure a clean end entering the mill and also enable cutting of the stock to the right length to accommodate the various mill tables. These crop shears usually are stationary, and may be steam, hydraulically or electrically driven.

Billets of large cross-section generally are cut by stationary shears. Such billets are usually short enough to be handled on tables and transfers, and there are comparatively few cuts to the finished piece. These shears cut one or more pieces at the same time. The length of cut is determined by an adjustable stop on the shear table and the shears usually can cut accurately to within ¼ inch of the desired length.

Another form of shear is the gang shear in which several shears are placed in line in such a manner that any length of billet may be sheared by moving the position of individual shears within the limits of the first and last shear in line. They may be made to cut in unison, or separately, as desired.

The flying shear, as opposed to the stationary shear, cuts the piece as it travels in a continuous operation. These shears are used for the smaller sizes that would require excessively long tables and transfers and would, of course, consume too much time in handling, if cut on stationary shears. This is obvious when it is realized that an 8,000-pound ingot, rolled into a 1½-inch by 1½-inch billet, would be over 1000 feet long. By the use of flying shears, this piece can be cut effectively into shorter lengths as it leaves the finishing stand.

Billets also may be cut to length by saws or flame cutters to eliminate distorted ends produced by shears. Sawing is usually slower and more expensive than shearing, and saws are more costly to maintain.

Flame cutting, while entailing a comparatively small initial investment, is costly in that it is slower than shearing and also requires the use of relatively expensive gases.

Identification—It is of primary importance that every billet be identified properly. Every billet should be stamped with its heat number, and some orders require that top and bottom cuts from the ingot be identified. The steel must be followed closely through the mill, especially in those mills composed of several units, to make certain that the correct steel is coming for which the stampers already have received instructions. This tracing of the steel can become quite complicated at times, and must be adhered to rigidly in order to prevent costly errors in identification. Some mills identify all products by the use of special hand or pneumatic hammers having suitable heads for containing the stamps. Heat numbers can be machine stamped on billets after leaving the finishing stand by the use of a

wheel, suitably mounted and containing the proper marking devices or stamps, contacting the steel. These wheels can be held in a stationary housing, so arranged that the height above the table can be varied for different sized billets, or they may be moved by a manually controlled hydraulic cylinder to contact each piece as it passes over the table. Usually, heat numbers only are applied in this manner.

After leaving the shears, billets are delivered usually to cooling beds, or hot beds as they are sometimes called, where they are cooled before shipping. In many mills, the above mentioned hand stamping is done on billets on the hot bed. The hot beds are at right angles to the direction of delivery from the mill and are wide enough to handle the longest billet length regularly produced on the mill. These beds are composed of skids placed close enough together to support the billet, but not too close to permit air to circulate around them. The skids may be composed of rails or of a series of specially designed castings. The castings give much better results than rails

since they do not bend so readily with prolonged use. After cooling, the billets are pushed off the hot beds into cradles, from where they can be loaded easily into cars for shipment, or transferred to the conditioning area where they are handled as described in the next chapter.

Chapter 26

PREPARATION OF SEMI-FINISHED STEEL FOR FINISHING

SECTION 1

SURFACE DEFECTS ON SEMI-FINISHED STEEL

Semi-finished steel is previously hot-worked cold steel intended exclusively to be subjected subsequently to further hot deformation. When steel is rolled from ingot to semi-finished forms, such as blooms, billets and slabs, a variety of ingot defects and some defects arising during heating and rolling may be carried through to appear on the surface of the semi-finished product. It often is necessary to remove these defects before the steel is converted further into finished products by forging, rolling or other hot forming operations. The removal of these defects, which operation is known as **conditioning**, will be the subject of the first three sections of this chapter; a discussion of the principles and practice of controlled cooling of certain grades and sizes of semi-finished products from rolling temperatures will be presented in Section 4.

Surface defects on semi-finished products may be caused by defects on or in the ingot or by improper handling after pouring or by variations in heating and rolling practice.

The major surface defects which generally are attributed to defects on the ingot are:

1. **Ingot Cracks**—These defects occur as both transverse and longitudinal ruptures in the ingot wall, and normally are observed first while the ingot is being rolled on the primary mill, although some are apparent on the surface of the ingot itself, especially if it becomes cold. Ingot cracking has been the subject of numerous investigations, and some of the causes have been brought to light although many still remain obscure. Excessively high pouring temperature (a temperature considerably above the solidification temperature) has been established as one definite cause of ingot cracking. During solidification a dendritic crystalline structure is developed in the ingot and interdendritic zones of weakness are formed which extend from the ingot surface toward the center. The larger the dendrites, the more pronounced are these zones. Lower pouring temperatures will help eliminate cracking from this origin by limiting the size while increasing the total number of individual dendrites. Folds due to surging of the molten metal in the mold form discontinuities in the ingot wall that lead to transverse ingot cracking. This type of defect has been minimized by use of mold coatings and improved mold design. Steels of the 0.15 to 0.25 per cent carbon grades, especially the fine-grained killed types, have the greatest tendency toward transverse cracking. Generally, the higher carbon steels have the least tendency toward this transverse rupturing. **Hanging** of hot-topped ingots by fins forming over the edge of the mold wall usually produces a transverse crack approximately 6 inches below

the hot-top junction, easily recognizable and termed a **hanger crack**. Hanger cracks from this source can be prevented by use of properly designed hot-tops. Plain-sided molds are more prone to produce or at least accentuate transverse cracking than the fluted type. Longitudinal cracks generally are related to the flute or corner design of the mold.

Other and more obscure causes and corrective measures for ingot cracking are a constant subject for study by practically all steel-mill personnel.

2. **Scabs**—This type of defect is caused by splash of metal against the mold wall when an ingot is being teemed. Rapid solidification and oxidation of the metal on the mold wall causes it to stick to the ingot surface and finally appear as a scab on the surface of the rolled product. A picture of a scabby surface on a bloom is shown in Figure 26—1. This defect can be minimized by bottom pouring, use of mold coatings, and proper mold design, as well as by proper attention to pouring speed and technique, as discussed in Chapter 20. A “cold” heat usually causes difficulties in pouring, such as inability to obtain a tight closure of the ladle nozzle; the resulting leakage of molten metal when the ladle is positioned over the mold generally results in an ingot surface covered with scabs. Properly prepared mold coatings not only will minimize scabs but also will improve the surface condition generally. Many types of coatings have been employed with varying success, such as graphite, tar, aluminum paint, oil, and others. However, it might be noted that a poor coating is often worse than no coating at all, becoming a better adherent for splashes than the original mold wall itself.

3. **Seams**—This term is used to denote the straight-line longitudinal cracks or openings that may appear on the surface of semi-finished steel. Generally, they may be divided into two classes:

(a) A long deep seam usually is the result of ingot cracks, either longitudinal or transverse. A transverse crack in the ingot is elongated during rolling, and the seam so produced often has a “steeple” at the head, drawing out into a long straight line or a pair of parallel lines on the product. A deep seam on the semi-finished product surface such as originates in an ingot crack, is shown in Figure 26—2. The basic causes of these defects in the rolled steel are obviously the same as those outlined above for ingot cracks.

(b) Short seams, usually light and in clusters, may be due to a number of causes. Surface pits on the ingot, or subsurface blowholes which become exposed and oxidized during heating, will produce this defect. Some other causes include poor heating practice, and rolling



FIG. 26—1. Scabby surface on a bloom, showing the effects of roll collars on the pre-existing scabby surface on the ingot from which it was rolled.

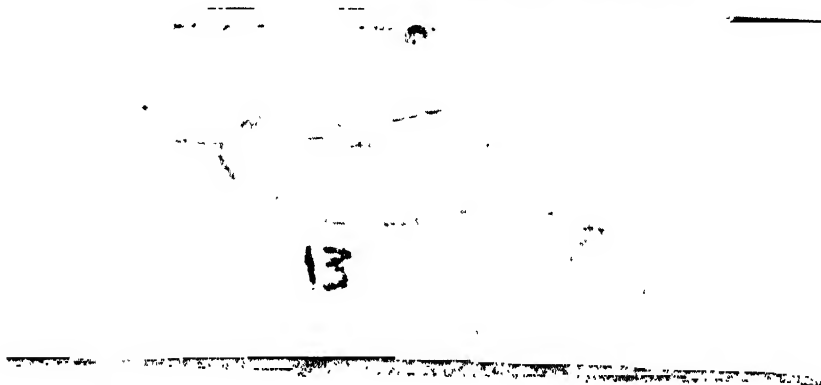


FIG. 26—2. Deep seam on the surface of a semi-finished rolled product, originating with an ingot crack.

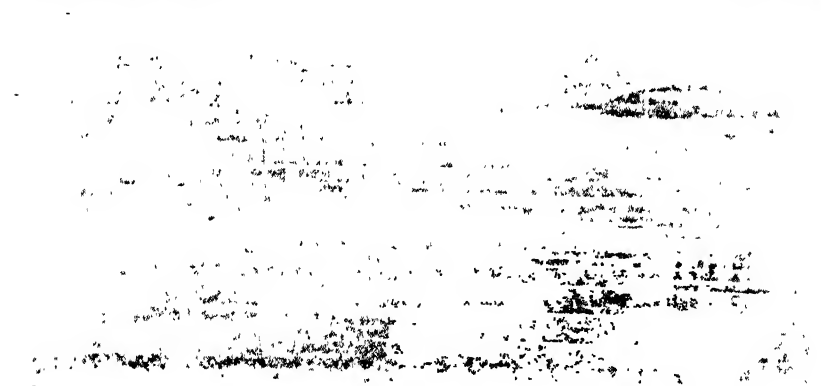


FIG. 26—3. Clustered seams on the surface of a semi-finished rolled product, pickled after rolling to facilitate detection of defects.

outside the proper temperature range. Clustered seams are illustrated in Figure 26—3.

Those surface defects which originate at the soaking pits and on the primary mill and other mills rolling semi-finished products are:

1. **Cinder Patch**—This defect is the result of pick-up of material from soaking-pit bottoms, and generally has the appearance of a very scabby bottom. Cinder patch can be overcome to a great extent by proper attention to bottom making. Some plants charge hot-top ingots of the big-end-up type upside down in the pits, thus

causing the cinder patch to be confined to the hot-top or discard portion of the ingot.

2. **Burned Steel**—This defect is very often the result of flame impingement on the surface of ingots, usually on their corners, as they are heated in the soaking pits, causing penetrating oxidation at the grain boundaries with a resultant tearing or rupturing of the ingot on the primary mill. A burned bloom is shown in Figure 26—4. Burned steel seldom can be salvaged and normally must be scrapped. Some steels are more sensitive to burning than others.

CONDITIONING

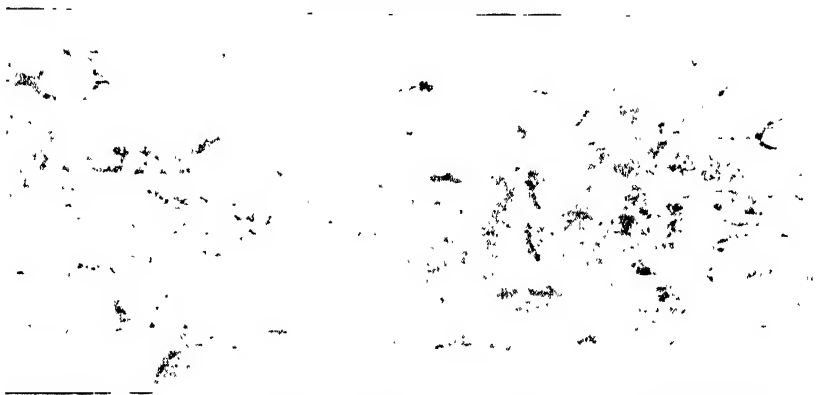


FIG. 26-4. Surface of a "burned" steel bloom.

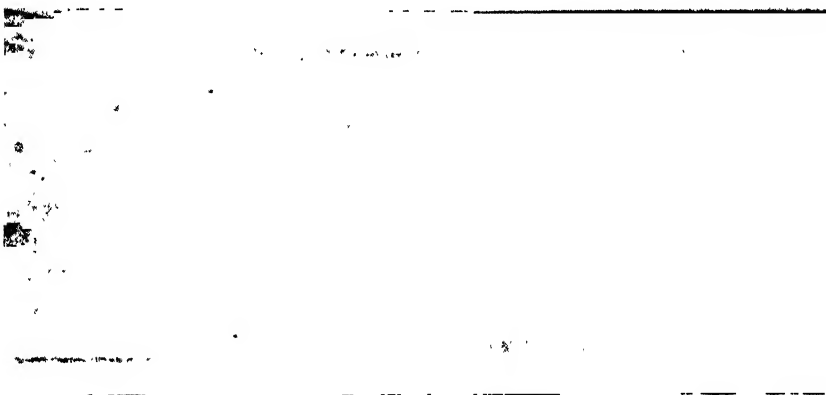


FIG. 26-5. A lap on the surface of a rolled steel billet.

3. **Laps**—Laps are the result of overfilling in the mill passes that causes fins or projections which turn down as the material rolls through succeeding stands in the mill train.

A lapped billet is shown in Figure 26-5. Laps usually are deep and the product often cannot be salvaged economically.

The foregoing list of defects is not all-inclusive, but presents merely the more important causes of surface defects that require removal in conditioning. There are numerous other defects, such as: twist, guide marks, collar cut, diamond, bent, crop ends, and so on, the names of which are self-explanatory. Most of these will receive attention at various points in the text.

SECTION 2

INSPECTION

Conditioning is that function of steel manufacture which, by removal of defects, renders steel more suitable for subsequent hot- and/or cold-forming operations. Of primary importance is the detection and proper removal of the defects where required. It is neither economical nor always necessary (and, strictly speaking, probably not possible) to remove all of the imperfections.

What, if anything, is removed is determined by the prospective subsequent operations and final end use of the product. The following paragraph is an example of the method for determining the conditioning practice in a plant producing semi-finished steel for bar products.

Using an arbitrary classification, product end uses can be grouped to include under one class semi-finished products which require the removal of all surface defects visible after pickling or skinning. As an example of this class of material, if the steel is intended for alloy-steel bars of high hardenability and the bars are further quenched in oil after reheating, it is obvious that any light seams may result in quench cracks, so that all

defects should be removed from the semi-finished steel. In a second example, where the material is to be rolled or forged to some finished form, but not heat-treated, oxidation during the reheating for rolling or forging will eliminate, in many cases, slight surface imperfections, and semi-finished products for this use could be placed in another class for which removal of only those defects visually detectable without pickling would be required. If material is to receive considerable work in further hot rolling or cold machining, even medium-sized seams can be acceptable, offering another classification which might be defined as a class of material requiring the removal of only major defects such as scabs, ingot cracks, laps, and deep seams. Finally, the remainder of the product can be placed in a class from which only those defects need be removed such as large scabs or ingot cracks, deep laps, and burned steel which might be damaging to either or both the final product or the operation (such as subsequent hot rolling where large scabs might cause cobbles on the mill, tear out the

guides or cause other damage). Thus, it can be seen that the basis for inspection is predicated both on the end use of the product as well as upon the operations employed to shape the steel into its final form.

Pickling for Inspection—Pickling, as applied to surface inspection and conditioning, consists of immersing the steel in a chemical solution for the purpose of removing the scale on the outside surface so that defects will be exposed. The usual solution for ordinary steels is composed of 5 to 8 per cent sulphuric acid in water, used at temperatures between 150 and 180° F. A good, clean surface is desired for optimum results in inspection, and to attain this, relatively close control of temperature and acid concentration is necessary. A thorough rinse

after pickling is required, either by immersion in clean, hot water or washing with high pressure nozzles. The time required for pickling varies with the grade of steel; for example, a 10-ton lift of blooms of plain-carbon steel may require pickling for around 45 minutes, while the same quantity of constructional alloy steel may require treatment for from 1 to 2½ hours.

Pickling generally has been associated with hand chipping, as a dual operation. In those plants where hand scarfing has replaced hand chipping, the pickling practice has been abandoned, too. Modern practice for the same specifications is: for rectangular sections, to skin by scarfing and then spot scarf where needed; for round sections, to peel mechanically and then spot scarf.

SECTION 3

REMOVAL OF DEFECTS

Hand Chipping—Until a few years prior to the First World War, the practice of extensive conditioning of semi-finished steel was almost unknown except for seamless tubes and special forgings. For most purposes, the use of a cold chisel and hand hammer was about all the work that was done, and this only to knock off such things as large slivers or scabs that might tear out the guides in rolling on a secondary mill. For the most part, steel was either accepted or scrapped in the semi-finished form. It is not to be assumed from this that a good deal of unusable material was produced, for it must be borne in mind that, in those days, a great deal of stock-removal in machining was accepted as commonplace and the tolerances for finished work were much wider. This was the only method of steel preparation used to any great extent until the early 1930's.

Hand chipping, as it is known today, consists of using a pneumatic hammer with a cold chisel placed in the barrel, so that the reciprocating action of the piston on the chisel causes the chisel to cut into the steel and remove the surface to the desired extent. Figure 26-6 shows a close-up view of a chipper at work.

Machine Chipping—There are available several large machines built specifically for the purpose of removing

imperfections from the surface of steel blooms and billets. These machines fall into two general types: the planer and the milling machine.

The planer type, illustrated in Figure 26-7, is constructed with a large table which can be moved longitudinally and a tool holder which can be moved to any position in a plane vertical to the table motion. The steel is clamped onto the table by mechanical or hydraulic vise-type jaws. The table then is operated to move in a horizontal direction, back and forth in a reversing motion, so that the steel passes under the tool. The tool can be adjusted while the machine is in operation. The distance of travel and the depth of cut are governed by the operator. Speeds are varied in accordance with the hardness of the steel and the depth of cut desired. In this manner, any length of cut or repeated cuts in depth can be made, so that either complete skinning or partial surface removal can be accomplished.

The milling machine, typified in Figure 26-8, is constructed with a multiple set of tools in a revolving head. The steel is clamped in a stationary position, and the tool head can be moved in a horizontal as well as a vertical direction so that surface removal can be restricted to selected spots or the piece being conditioned

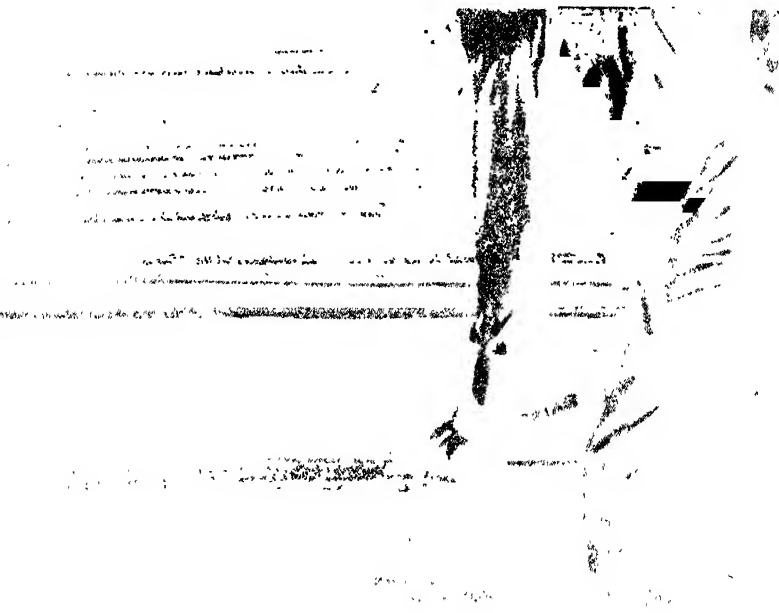


FIG. 26-6. Chipper using a pneumatic hammer equipped with a cold chisel for removing selected portions of steel billet surface.

CONDITIONING

FIG. 26-7. Planer-type machine for removing surface imperfections from surfaces of semi-finished steel.



FIG. 26-8. Milling machine adapted to the removal of surface imperfections from semi-finished steel.

FIG. 26-9. A scarfing torch in use removing surface imperfections from a rolled steel slab.



can be skinned completely. The entire frame of the machine carrying the tool is capable of traveling the entire length of the pieces processed.

Scarfing—Scarfing in the steel mill consists of surface removal by the use of oxygen torches (Figures 26—9, 26—11 and 26—14). The oxygen rapidly oxidizes the steel surface, generating elevated temperatures that cause the oxidized product to become liquid. The process can be carried out on hot steel between stages of rolling, when hot scarfing machines are used, or on cold steel in the conditioning area by the use of the hand torch. In principle, the tip of the scarfing torch is similar to the regular oxygen-gas torches used for the burning of steel. As early as 1919, surface-defect removal on semi-finished steel was attempted by hand scarfing, but because of lack of knowledge of proper torch design and the difficulties encountered with the fuel gas, the results obtained were unsatisfactory. During the 1920's, some attempts at specialized uses of torches met with success, but it was not until 1929 that a determined effort was made to procure fuels and equipment to perform satisfactorily the function of defect removal from the surface of semi-finished steels. Promotion of the chemical reaction that forms the melted iron oxide is not the only function of the oxygen; its kinetic energy must also force the liquid, oxidized metal from the path of the torch.

Hand Scarfing of Cold Steel—As might be expected, the first hand torches, though successful, were cumbersome and slow, but since 1929 many outstanding improvements have been effected so that the bulk of semi-finished steel now is conditioned by the scarfing method. So far, only oxygen has been mentioned, but fuel gas, too, is necessary to the functioning of the scarfing torch. The gas acts as a preheating agent to elevate a spot on the steel surface to such a temperature that the oxygen and the steel will begin to combine chemically. Once this preheating has been accomplished, the heat of reaction between oxygen and iron produces sufficient heat in front of the torch tip to maintain the reaction and fuel gas is no longer a necessary part of the operation. An interesting development with respect to this spot pre-heating is the so-called **starting rod**, which is a small rod of steel that extends into the flame and is heated until a small drop of melted rod falls onto the steel surface, thus instantly kindling the reaction. The melting of this steel rod is a matter of a fraction of a second as opposed to the 6 or 8 seconds required to heat sufficiently a spot on the steel surface with the flame alone. Many different types of torches and many different kinds of fuel gases are employed currently.

Perhaps the largest tonnage of all conditioned steel is processed by hand scarfing. Scarfing can be used for both small and large sizes down to about 2 inches by 2 inches, below which a bowing effect is produced.

A factor present in scarfing cold steel which is not present in other types of conditioning previously de-

scribed might be termed the metallurgy of scarfing. When a relatively small portion of a large piece of steel, such as the area adjacent to a cut made by the scarfing torch, is heated to a high temperature and the source of heat is suddenly removed, a **quenching effect** is produced by the rapid extraction of heat from the hot area by conduction of heat into the surrounding relatively cold areas. This rapid cooling of a heated portion of steel from a temperature above its critical temperature range causes it to harden, to a degree proportionate to its carbon content and to some extent also to the hardenability of the particular steel being scarfed. In plain low-carbon steel, this effect is not noticeable, but as the content of carbon or alloying elements increases, the effect becomes more and more severe, producing, in the extreme case, a hard, martensitic layer on the surface where scarfing has been performed.

This hardened surface will crack upon cooling if no preventive measures are taken. Scarfing cracks on a billet are shown in Figure 26—10. To prevent scarfing cracks, preheating before scarfing to temperatures between 300 and 500° F normally will suffice, although in highly critical grades, such as SAE 52100 (1.0 per cent carbon, 1.3-1.6 per cent chromium), postheating for stress relief after scarfing also should be employed. To eliminate completely the hardened surface, a normalizing or annealing heat treatment is necessary. Ordinary brick pits, gas fired, can be used for routine preheating before scarfing, where a sufficient tonnage of sensitive steels is involved. Conventional car-bottom or continuous heat-treating furnaces are employed for complete softening of the steel by heat treatment. The necessity for proper preheating of some steels cannot be over-emphasized, since scarfing cracks cannot be tolerated in finished products.

Mechanical Scarfing of Hot Steel—The mechanical hot scarfer (Figure 26—11) originated about 1930. It is installed directly in the mill line, and is composed of a number of scarfing torches so designed that they form a pass on the mill. The torches are used in exactly the same manner as the hand torch. A starting rod is not used because the high temperature of the product being rolled is enough to obtain quick starting action. The working speed of the machine must be coordinated with the mill so that no delay in rolling production is encountered. Only a moderate amount of metal can be removed but very satisfactory results can be obtained in conditioning some products. This equipment is not confined to four-side removal, but has been employed for two-side removal (of slab edges and the like). Such machines can be designed to meet the needs of individual mills.

Grinding—Conditioning by grinding also is in general use. For semi-finished steels, the **stationary** or **swing-frame grinder** is employed. This machine consists of a large grinding stone, up to 24 inches in diameter, electrically driven. The machine is suspended so that it can

FIG. 26—10. Surface cracks resulting from scarfing to remove surface imperfections from a rolled steel billet.

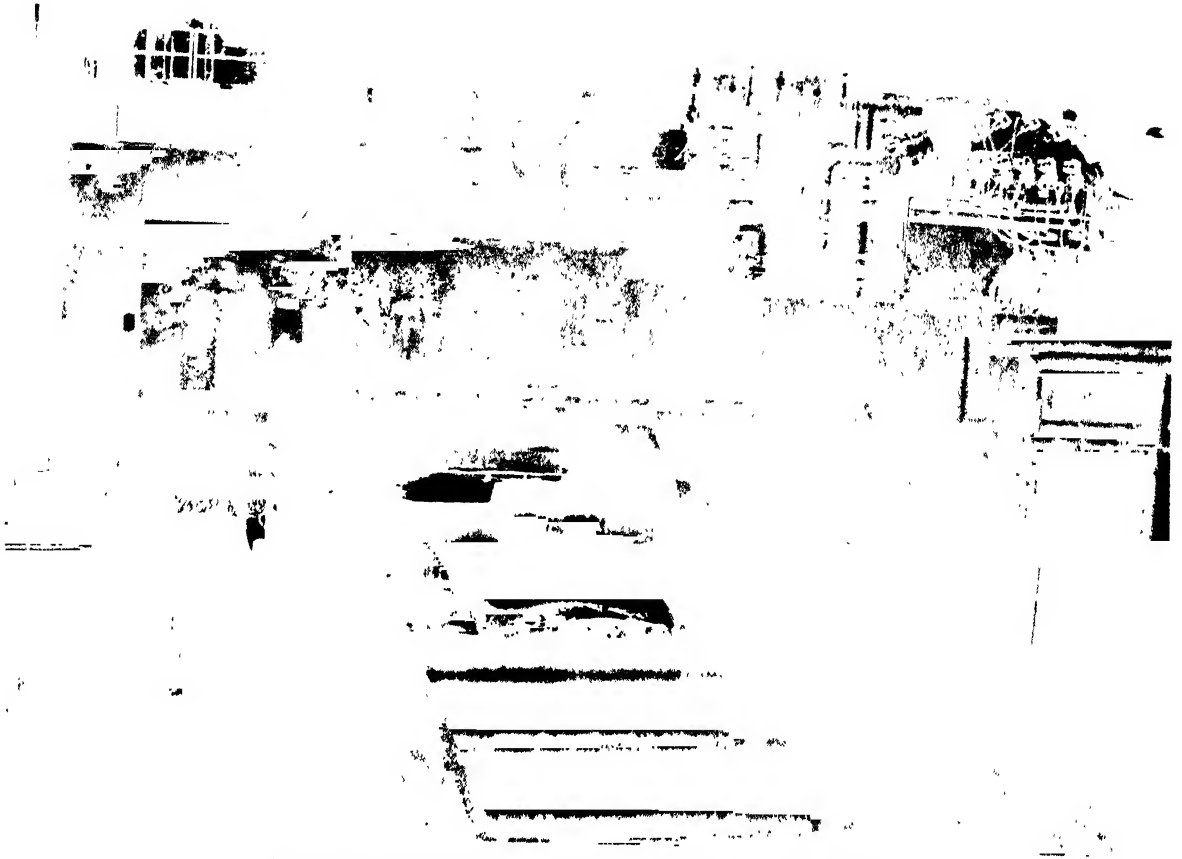


FIG. 26-11. View of a hot scarfer in operation at a slab mill

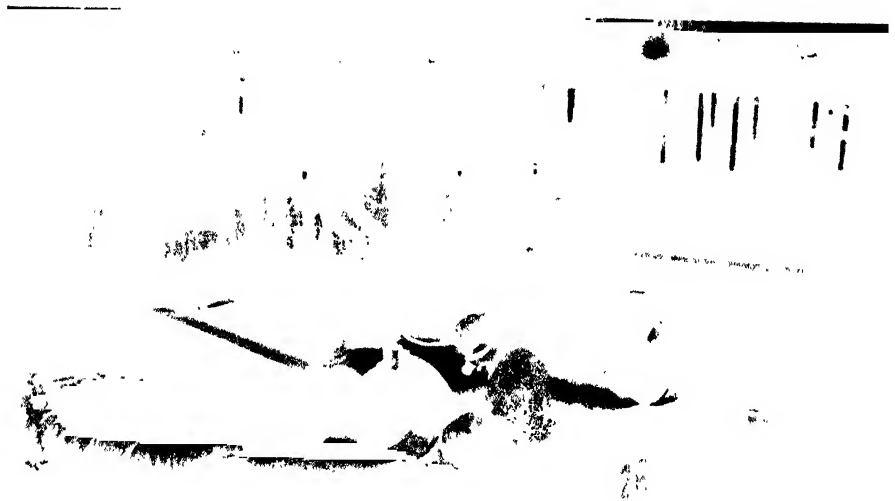


FIG. 26-12. Two swing grinders in operation conditioning the surface of a steel slab.



FIG. 26—13. Straddle carrier in right foreground facilitates the transporting and stocking of semi-finished material.

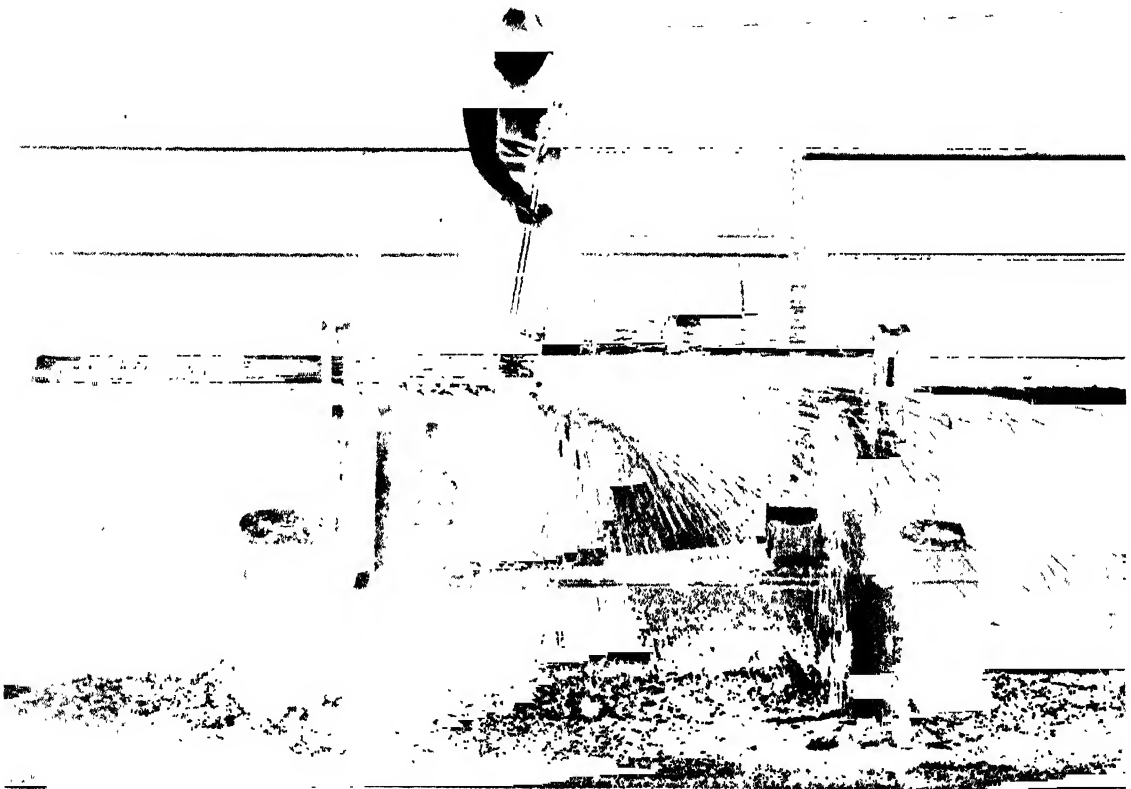


FIG. 26—14. Overall view of a bloom-turning machine showing a bloom on the supporting arms being scarfed.

be raised or lowered readily to suit the product being ground. When the material is positioned properly and held firmly, the operator grasps the handle bars and, by body pressure, forces the revolving stone against that portion of the surface to be removed, moving the stone with a slight back and forth horizontal movement until the defect is gone (Figure 26—12). The grinding wheel can be considered as a revolving cylinder carrying many sharp tools, which are the abrasive grains. When revolving at a proper speed, the grains cut very small chips from the material being ground. It is, therefore, the cumulative effect of the action of a very large number of cutting points which produces the result. Grinding cracks can result from the heat generated by grinding unless the proper wheels are used; this occurrence is related to the quenching effect discussed under "Hand Scarfing of Cold Steel."

Grinding is a more or less specialized process, being particularly adaptable to stainless steel and other products which cannot be scarfed or chipped.

Material Handling—New developments in the material handling field are being employed in increasing numbers to simplify the handling and stocking of material for conditioning. The use of **straddle carriers** (Figure 26—13), in some plants which have large conveniently located vacant areas, has revolutionized the transporting of semi-finished material, permitting flexibility and simplification of stocking areas, improved scheduling and better control of inventory. **Bloom-turning machines** (Figure 26—14) safely and efficiently position material automatically for hand scarfing, permitting increased tonnage to be conditioned. These machines are designed to be loaded and unloaded by fork lift trucks, eliminating the need for cranes.

SECTION 4

CONTROLLED COOLING OF SEMI-FINISHED PRODUCTS

Blooms, billets and slabs are rolled at temperatures well above the critical temperature range of the steel and thus, in cooling to atmospheric temperature after rolling, must pass through its transformation range. Depending upon the chemical composition of the steel and the size of product rolled from it, it may be necessary to retard the rate of cooling by artificial means for two reasons:

- (1) To prevent the formation of small internal ruptures, called flakes.
- (2) To minimize development of internal stresses.

Some steels are relatively very sensitive to rapid changes in temperature, and this thermal sensitivity requires that great care be exercised during both heating and cooling to avoid physical damage to the steel due to high internal stresses.

Nature and Prevention of Flakes—Flakes are small internal ruptures that usually occur some distance away from the end of a piece and often midway from the surface to the center of a section. Although there is not complete accord as to the relative importance of the

various factors contributing to flaking, it generally is considered that hydrogen dissolved in the molten steel makes it more susceptible to flakes and that proper retardation of cooling from forging or rolling temperatures effectively will prevent their formation.

Development of Controlled-Cooling Practices—The earliest method employed for retarding the rate of cooling after forging or rolling was to bury the steel, as it came from the press or mill, in ashes or some other insulating material such as sand or slag. The purpose of this was not to prevent flakes, but to retard the cooling sufficiently to lessen the hardness developed by the steel on cooling and to minimize the development of internal stresses. Rates of cooling varied greatly, depending upon the type of insulating material used and the depth to which the steel was covered. No positive control of cooling rates was possible with such crude methods, and the time required before the steel safely could be removed usually was two to ten days. Another method of slow cooling involved the use of unfired pits, lined with brick to provide insulation. After the steel was placed in these

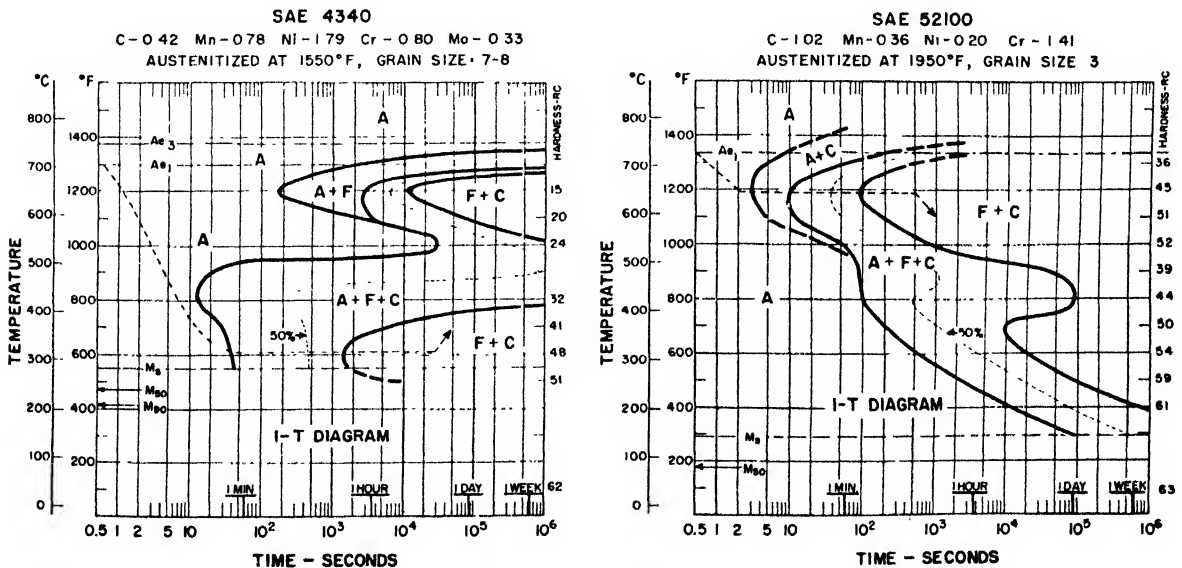


FIG. 26—15. Cooling curves superimposed on isothermal transformation diagrams of two steels, showing cooling cycles employed to effect transformation in a minimum of time while still avoiding flake formation.

pits, they were covered with insulated lids, and the charge cooled at a rate dependent upon the mass of the charge and the rate at which heat was lost by radiation. In another method, steel boxes were inverted over piles or bundles of hot steel as it was delivered from the mill. Because of the space and long time required, and the difficulties involved in handling, the burying processes proved entirely inadequate as increased ton-nages of alloy steel came to be produced. Consequently, it became imperative to develop more suitable means for slow cooling.

In recent times, use has been made of automatically controlled furnaces, insulated and heated railway gondola cars with covers, and heavy pits equipped to control temperatures during slow cooling. With such equipment, it has been possible to take advantage of cooling cycles of considerably shorter duration with better control of results. When using equipment that makes possible close control of rate of cooling, the temperature of the steel can be lowered rapidly and arrested at the desired temperature which improved knowledge of heat-treating procedures has established as effective for each particular type of steel. After a suitable holding time, measured in hours instead of days (as was the case when the steel was simply buried in ashes), the steel can be removed and air cooled in the open, permitting prompt placement of the next charge in the furnace or pit.

Much experimental work has been done to determine the proper holding temperature for the different grades of steel. Figure 26-15 shows the cooling curves for AISI 4340 and 52100 steels superimposed on the isothermal transformation curves for these steels. The latter curves, formerly called "S" curves, show the time required to complete transformation at various temperature levels, and a more detailed description of their derivation and use will be found in a later chapter. The objective in selecting the holding time for each of these grades was to determine what temperature would require the minimum holding time to obtain transformation and still give a product free from flakes. In the case of the 52100 steel, 1200° F was selected as the temperature from which the billets could be air cooled. For the 4340 steel, holding at 1200° F was not feasible, as excessive time would be required for transformation. Since 600° F corresponded to the minimum time of holding, this was the temperature selected. Transformation in this

range would result in an appreciable amount of internal stress which might result in flakes on cooling in air to atmospheric temperature. Hence, instead of air cooling immediately after the end of the holding period, this particular steel should be heated instead to a temperature just below the critical range to relieve these stresses, after which it may be cooled.

These two types of cycles are applicable to a large number of different grades of steel, and the holding temperature for a particular grade can be determined from the proper isothermal transformation diagram. In general, the higher holding temperatures followed by air cooling can be used for the carbon and lower alloy grades of steel; the use of lower holding temperatures, followed by immediate reheating to just below the critical temperature range, is applicable to the more highly alloyed deep-hardening steels.

Chapter 27

ROLLING OF CARBON-STEEL PLATES

SECTION 1

PLATE-MILL OPERATIONS AND PRODUCTS

Plates are classified, by definition, according to certain size limitations, in order to distinguish them from sheet and strip. In accordance with this classification, carbon-steel plates are considered to be those flat, hot-rolled, finished, carbon-steel products that come within the following dimensional and weight limitations:

- 0.180 in. or thicker, over 48 in. wide
- 0.230 in. or thicker, over 6 in. wide
- 7.53 lb. per sq. ft. or heavier, over 48 in. wide
- 9.62 lb. per sq. ft. or heavier, over 6 in. wide

There are a few exceptions to the above classification. Flat hot-rolled semi-finished carbon-steel products such as slabs, sheet bar and skelp are not classed as plates, although their dimensions and weights may be included within the foregoing limits.

Plates are used in plate girders, chord sections, gussets, and other parts of bridges, in freight and tank cars, in ships, line pipe, storage tanks, penstocks, pressure vessels, machine-tool frames, weldments, galvanizing pots and many special applications.

Plates are produced by hot rolling, either from reheated slabs or directly from ingots. The terms applied to plates to differentiate between the several kinds are based upon the types of mills used in their production, e.g., sheared plate, or sheared mill plates when rolled between straight horizontal rolls and later trimmed on all edges. They are called universal plates, or universal mill plates (abbreviated U. M. plates) when rolled simultaneously between both plain horizontal and vertical rolls and trimmed on the ends only. Grooved rolls are sometimes substituted for the plain horizontal rolls used for universal plates.

Carbon-steel rolled floor plates are also flat, hot-rolled, finished steel products which come under the plate classification. Floor plates are hot-finished in the final pass or passes between one or more pairs of rolls. One roll of each pair has a pattern cut into it so that one surface of the plate passing between the rolls is forced into the depressions on the pattern roll to form a raised figure at regular intervals on the surface of the plate. Individual floor-plate patterns are produced exclusively by each manufacturer, the patterns differing in both dimensions and appearance.

In addition to carbon steels, the complete range of steel-plate production includes high-strength, alloy and stainless steels, which are discussed in later chapters relating to these individual subjects.

Carbon steels constitute the major plate-steel production, but a discussion of the rolling mills and auxiliaries is common to all plate-steel production. There are, however, even with carbon plate steel, special manufacturing and quality requirements which have been adopted by the industry. These are not dis-

cussed here, but are given in full in "Steel Products Manual, No. 6," issued by the American Iron and Steel Institute.

The several types of plate mills, their design features and operation are discussed in the following paragraphs, and descriptions are given of individual mills, including sheared plate, universal plate, continuous and semi-continuous types.

Mill Operations—The sequence of operations for plate mills is covered by the following general subdivisions:

1. Heating the slabs for rolling
2. Descaling
3. Rolling plates
4. Leveling or flattening
5. Cooling
6. Shearing

Heating (General)—The era of the predominance of three-high plate mills was coincident with the development of open-hearth furnaces in which furnace capacities were relatively small, ranging from 40 to 100 tons. Ingots, with the exception of those poured for armor plates, were also relatively small and 40-inch maximum width dimensions were about the largest produced. The three-high mills, however, were not as restricted in the production of plate-size ranges as might be inferred from the restricted ingot sizes. Slabs were conveyed to the mill in a broadside position and after the requisite number of passes were taken to attain plate widths, turning hooks attached to rigid masts were employed to turn the stock 90 degrees by utilizing table traction. The stock was then reduced to final thickness in successive passes. Batch-type heating furnaces provided the additional flexibility of permitting slabs to be provided either for single units of the ordered plate size or for a range of multiples that would best conform to physical equipment limitations and operating economies.

Plate-producing plants were laid out so that slabs from the primary mills could be conveyed or transported to the batch furnaces as hot charges. Such layouts were motivated by the considerations: (1) that small ingots were not so susceptible as large ingots to being the source of major surface defects that carried through to the finished product; (2) that economical, intermediate-conditioning processes had not yet been developed; and (3) that the economies of increased production rates from hot charges would more than offset the surface rejections which might be incurred.

Heating in a fuel-fired batch furnace is essentially a process of transmitting heat by radiation and convection from the combustibles to the top slab surface and by conduction through the thickness of the slab to the bottom surface and the hearth. Even where the tem-

peratures of both the hearth and the charge are high in relation to the drawing temperature at the beginning of a cycle of charges and draws, the heating rates per square foot of furnace area are low in contrast with continuous slab-heating furnaces. When the cycle involves cold charges, the initial abstraction of heat from the hearth, which must be restored later in the cycle, slows the process to the point where continuous operation is impracticable, if the heating capacity was installed initially for hot charges. The process of heating from top to bottom through the thickness, in itself, suggests the existence of a temperature differential which can be minimized only by soaking. Changes which have occurred in the industry in the past thirty-five years have created a trend in the direction of cold charges and have confronted many plants with the necessity of determining whether to operate mills on an intermittent basis, add heating capacity, or build new mills.

Furnace-control equipment is intended not only to effect fuel economies, but also to improve the quality of heating. Slab heating is improved if slabs are brought to the plastic state most satisfactory for rolling, with a minimum of temperature variation in each slab without any part ever attaining an excessive temperature, and with the production of a scale coating conducive to the best surface finish. The thickness and the type of scale formation is dependent on temperature, time at temperature, and availability of oxygen. The temperature that must be used is dependent upon the required plastic state of the heated steel; time of heating is controlled by the heating capacity of the furnace; therefore, control of the fuel-air ratio is the only flexible practical determinant of the types of scale that may be formed, which may vary from extremely adherent to loose. Automatic furnace-control equipment was not available when the bulk of existing batch furnaces was designed. Some were modified later to permit installation of controls. Several furnaces, such as at Homestead District Works, built during World War II, were designed for and equipped with automatic furnace controls. The preponderant number of batch furnaces still in operation are dependent on manual control.

Most of the existing continuous slab-heating furnaces were designed for and are operated on cold charges. The slabs are charged into the low-temperature end of the furnace and progress toward the high-temperature end by being pushed progressively forward on water-cooled skids. Burners located above and below

the skid level furnish heat by radiation and convection to both the top and the bottom surfaces of the slabs until they reach the high-temperature zone, which is, in effect, a soaking zone. Only top burners are provided for the high-temperature zone. The slabs are at approximate rolling temperature when they arrive in this zone, and elimination of skid marks (visibly colder "stripes" on slabs caused by contact with the water-cooled skids) and other temperature non-uniformities, plus making up radiation losses to maintain the desired furnace temperature, are the principal heat requirements to be met. Slab-heating furnaces of this type either had furnace controls included in the original design or they were added later, inasmuch as such furnaces are very adaptable to automatically controlled operation. Scale formation, consequently, can be controlled closely with respect to both thickness and type.

Descaling Methods and Equipment—Prior to the introduction of the modern hydraulic pressure sprays, various expedients were resorted to for primary scale removal. Moistened burlap thrown on the slab at the entry side of the rolling mill embodied the same scale-removing principle as the more common use of salt. In both instances the materials were used to get moisture between the work rolls and the stock, where it vaporized instantaneously with explosive effects and removed the scale. Of all flat, hot-rolled products, hot-rolled sheet and strip must meet the most stringent surface finish requirements, and it was natural that scale-removal methods and equipment would first reach their maximum development in connection with those product classifications which were rolled on continuous mills. The trend of development moved in the direction of the use of a leading roll stand in which light drafts were taken for the initial breaking of the scale. Hydraulic sprays, impinging on both top and bottom surfaces, were placed on the delivery side of the scale breaker and on the entry side of the roughing stands and the first finishing stand. Hydraulic pressures were increased with various installations from the 600 to 800 pounds per square inch which prevailed for general plant hydraulic equipment, up to 1,500 pounds per square inch. It was found, however, that with proper heating, spray pressures of 1,000 pounds per square inch were adequate for the production of surface finishes to meet consistently the most stringent requirements. The continuous hot-strip mill method of scale removal was, therefore, adopted for new plate-mill installations.

SECTION 2

GENERAL TYPES AND OPERATING CHARACTERISTICS OF PLATE MILLS

In this section, general design and operating features of plate mills of several types will be compared. Detailed descriptions of existing plate mills will be given in Section 3.

Plate-rolling mills are generally considered in two very broad design classifications. One type includes the universal mills, which are characterized by vertical rolls preceding and following the horizontal rolls. The horizontal and the vertical rolls are integrated into a single mill unit and work the stock simultaneously. The purpose of the vertical rolls is not only to work the edges of the stock in the process of reduction, but also to produce a rolled width in conformance with specified standard tolerances.

The second type includes the sheared-plate mills,

some of which may include edge-working equipment. While some installations use the edging equipment for both edge working and approximate width sizing, final widths are attained by edge shearing. Sheared-plate mills, in turn, may be subdivided into the following mill types: (1) the single-stand two-high pull-over mill, (2) the single-stand two-high reversing mill, (3) the conventional three-high mill, (4) the four-high reversing mill, (5) the tandem mill, (6) the semi-continuous mill, and (7) the continuous mill. In addition to the distinct types mentioned, a variety of tandem combinations are in operation. If the characteristics of the five distinct types are understood, the working characteristics of a variety of possible tandem arrangements readily can be inferred.

TWO-HIGH PULL-OVER MILLS

The two-high pull-over plate mills were essentially an adaptation of the then existing sheet mills to plate rolling. Prior to the development of mechanical passers, single-stand hot sheet mills were drafted in one direction only, and the sheet packs were returned for successive passes by being lifted manually on to the top roll and pulled over by hand to the starting or entry side with the aid of the tractive friction of the top roll. Roll settings were altered after each pass by hand operation of levers, which rotated the head screws. Plates rolled on these mills were limited in size and weight by the ability to handle manually. This restricted their utility and soon rendered these mills obsolete for producing finished plate. Two-high non-reversing mills still are in operation, however, in the form of single-pass units, such as scale breakers and roughers, in tandem arrangements in which the finishing-mill unit is either a three- or a four-high mill.

TWO-HIGH SINGLE-STAND REVERSING MILLS

Hot-rolled sheet production by the pack method antedated volume tonnage production, and recognition of the impracticability of handling similarly the heavier plate weights prompted the introduction of reversing engines for two-high mills. These mills, likewise, soon reached their practical limit of application, particularly with respect to the production of the wide, lighter-gage plate. The fact that all passes from slab to finished plate were made on the same set of rolls accelerated roll wear, which is always greatest in the central portion of the roll body and in itself imposes a restriction on the gage that can be finished satisfactorily. Roll deflection is an important factor and, for the same roll diameter, increases with the body length for like reductions. An increase in the roll diameter to provide more strength and stiffness increases the separating force between the rolls for the same draft, because of the greater area of contact between the stock and the rolls. The lessening of roll deflection, therefore, progresses at a diminishing ratio with increasing roll diameters.

The art of casting massive rolls developed gradually. Increased roll separating forces for like reductions meant greater power requirements for rolling, as well as to overcome greater resistance to acceleration and deceleration. The design of reversing engines of increasing power also was a gradual development. Neither of these developments progressed sufficiently to meet expanding plate requirements prior to the invention of the three-high mill which overcame, at least to a degree, some of the principal drawbacks of the two-high reversing mill.

Operation of two-high single-stand reversing plate mills continued in England after they were largely displaced on the Continent and entirely displaced in this country. As they are obsolete in this country, a detailed description of any installation would have historical value only.

THREE-HIGH MILLS

The invention of the three-high mill provided design features which, to a degree, overcame some of the principal limitations of the two-high reversing type. In the three-high mill, the top and the bottom rolls are of large diameter, whereas the middle roll is friction-driven and usually about two-thirds of the diameter of the top and bottom rolls. The top roll can be raised and lowered in the housing by power-operated screws, and the middle roll can be brought into contact alternately with the top and the bottom rolls. In making the bottom pass, the stock passes between the middle and the bottom roll

while the top roll serves as a backup roll. The stock is raised on the delivery side by a tilting table for a return pass between the middle and the top roll, while the bottom roll serves as a backup roll. The sequence of alternate passes is continued until the stock is reduced to the desired finished plate thickness.

The middle roll is changed when combined roll wear produces a crowned plate which approaches the permissible tolerance limits. The replacement roll is itself crowned to compensate for the wear which already has taken place on the top and the bottom rolls. Successive replacements in the course of a week's rolling schedule are turned with progressively increased crowns to compensate for the continued wear of the top and the bottom rolls. During the weekly mill-repair shutdown, the top and the bottom rolls are either turned in place to their original contours or are replaced with newly dressed rolls. The cycle of replacement of middle rolls with progressively increased crowns is repeated in the following week.

The fact that one of the rolls in contact with the stock on each pass is of smaller diameter than is required for strength (the requisite strength being provided by the roll serving as a backup) reduces the total roll separating force for like drafts in contrast with a two-high mill. This principle, when applied to wide plate production, served to solve the problem of providing rolls of the required strength without being too massive. The fact that only one smaller-diameter roll needed to be replaced in the course of a rolling schedule for wear compensation, eased the roll-changing problem. The three-high roll arrangement, with tilting tables on entry and delivery sides, eliminated the necessity for using reversing engines. The tilting tables and the unidirectional main drives permitted a shortening of the rolling-time cycle and provided a temperature advantage in the finishing of light-gage plates. Continued trade demands for improved finishes, lighter gages, and closer and more uniform rolling to desired dimensions (width and thickness) for many applications, which the three-high mill could not meet consistently, hastened the development of the other plate-mill types.

FOUR-HIGH REVERSING MILLS

The development of the four-high reversing plate mill further increased the advantages which the three-high mill possessed when compared with the two-high reversing mill. The backup-roll to work-roll diameter ratios were increased to over two-to-one as compared with the three-to-two ratios prevailing in three-high mills. For like reductions, therefore, not only is the total roll separating force less, but also strength is provided for each pass by the backup rolls on both top and bottom sides. A four-high reversing plate mill was in operation in 1917 at the Coatesville, Pennsylvania, plant of the Lukens Steel Company; but the general adoption of the type for plate rolling was retarded until antifriction bearings for the rolls became available. Such bearings were developed for the hot-strip mills and the cold-reduction mills and their use was extended to four-high reversing plate mills. A concurrent development was the designing of multi-armature reversing motors for primary mills. The development of the latter motor types reduced the inertia effects of massive mill parts and their relation to acceleration and deceleration on reversals. The multi-armature motors replaced the reversing steam engines as prime movers.

TANDEM MILLS

Many of the original single-stand plate mills have been supplemented by an additional stand to form a

tandem plate mill. The various tandem-mill arrangements represent a wide variety of mill unit combinations, which achieve two principal objectives. When the total work of reducing slab to plate is divided between two mill stands, satisfactory surface finish and shape of rolls can be maintained for longer periods between roll changes. Secondly, since the work is divided between two units operating simultaneously, the required time interval for the reduction of a slab to a plate is reduced and the overall capacity of the unit is increased correspondingly.

A wide variety of tandem arrangements exists because they represent modifications of original rather than new installations, and because they were accomplished by the maximum utilization of existing equipment, the minimum expenditure for new equipment and a minimum of alteration to auxiliary facilities. The tandem arrangements, consequently, may include a two-high reversing rougher with a three-high finisher; a two-high reversing rougher with a four-high finisher; a three-high rougher with a three-high finisher; or a three-high rougher with a four-high finisher.

The two principal advantages of the tandem mills over the corresponding single-stand mills have been stated. No specific installation has mill or auxiliary equipment of engineering interest which is not covered in descriptions of more standardized arrangements.

SEMI-CONTINUOUS AND CONTINUOUS MILLS

These plate mills constitute the plate-mill groupings which include multipass, reversing, roughing units for the semi-continuous mills, and non-reversing roughing units for the continuous mills, coupled with two or more single-pass continuous units in which the plate is reduced simultaneously. Two-, three- and four-high stands with or without scale breakers, broadside stands, squeezers, and edgers are used as roughing units, while four-high stands are used as finishing units. The semi-continuous mill arrangement, although requiring a larger capital investment, has a number of operating advantages over both the single-stand and the tandem-mill types. The total reduction work is divided between individual stands to an even greater extent than in the case of the tandem mills. The roll wear of individual stands is, therefore, less than that of the prior mill types. The total time increment for reduction from slab to plate also is less, and the tonnage capacity per unit of time correspondingly is greater. The decreased time element permits sheets as well as plates to be rolled on these units.

UNIVERSAL MILLS

Universal plate mills are single-stand units which roll plates to width within standard tolerances. Although universal plate mills producing widths up to 60 inches have been in operation, the bulk of the installations has been of 48-inch width and under. The number of installations and the capacity has decreased rather than increased in the past several decades. This may be attributed to the development of the narrow as well as the wide continuous hot-strip mills, and to the development of the wide-flanged beam mills. The former mill types, particularly in the narrower widths, have displaced directly considerable universal-mill tonnage, owing to the improved surface finishes that can be obtained with the continuous hot-strip mills. The production of deep wide-flange beams displaced a large tonnage of fabricated beams and columns that formerly had utilized universal plates for the web portions. Consequently, the universal plate mills which are in operation today date back several decades to their installation

dates; and the unit which will be described in detail is typical in spite of its early construction date.

PLATE-ROLLING VARIABLES

The rolling of plates is subject to a number of variables. Control of temperature, if not properly effected, may cause variations in mechanical properties in steel from the same melt or lot of material. Inherent characteristics of equipment permit one mill to roll plates to closer limits for thickness, weight, width, length, camber (the greatest deviation from a straight line along a longitudinal edge), and flatness of plate after mill finishing-pass delivery and ordinary roller-leveler treatment.

Other variables affecting rolling are the **mill spring** (the total looseness under load of all mechanical parts) between the roll necks and the housings, which makes it necessary to increase spring allowances for light mill construction as compared with heavier or more rigid mill construction for the same loading.

Bending of rolls and resultant **crown** in the plates are also variables which affect the accuracy of rolling. When the rolls are subjected to the separating force of a plate being rolled between them, they are equivalent to beams supported at the bearing centers and subjected to a uniformly distributed load over the length. Uniformly loaded beams have maximum deflection in the center and a similar condition exists in the rolls. For the same drafts, small-diameter rolls are subject to less separating force than large-diameter rolls but the latter have greater resistance to deflection. The minimum roll deflection and plate crown are found in a mill with small-diameter work rolls backed up by large-diameter rolls. Crown (increase in thickness of the rolled center of the plate over its edges) is related to the amount of bending or deflection in the rolls. A minimum crown and uniform thickness of plates are desirable for many applications, especially forming operations.

Roll wear is an important factor in rolling plates. There is only one point in the arc of contact of the rolls with the stock at which the linear speed of the rolls and the stock is identical. As mentioned in Chapter 19, this is called the neutral point. There is backward slippage of the stock on the entry side and forward slippage on the delivery side of the neutral point. Slippage contributes to roll wear, and as the central portion of the roll face comes in contact with the stock on all widths, it is subject to the most wear and gradually develops a hollow contour across the roll face, accompanied by a surface roughness.

The effect of roll wear on plate crown and surface finish can be visualized readily. An additional effect relates to plate flatness and becomes more pronounced with the thinner plate gages. Roller levelers operate on the principle of imposing a slight extension, through the bending-roll action, on each increment of constricted plate surface as it goes through each bending action in the leveler. The extensions which can be imposed by the bending action are relatively slight and, even for these to be accomplished in one pass, a minimum state of plasticity (corresponding to that which obtains at the lower transformation temperature) generally is required.

Because edges of rolled plate cool faster than the center, the desired shape for entry into the leveler, particularly for light gages, is one with a slight edge wave. A mill which is too full will deliver light-gage plates with center buckles that require excessive edge extensions in the leveler to secure flat delivery. Conversely, a hollow mill will deliver light-gage plates with more

edge wave than can be compensated for by the limited extension of the central portions of the plate which can be accomplished in the leveler.

Temperature variation in the plate from the front end to the back is also a problem in rolling plates. Every roll pass involves a time interval for its accomplishment. In initial passes, when the stock is relatively thick and short, the time interval has no practical effect in creating a temperature differential from front to back. If the stock is being reduced to wide, light gage, and is elongated to long, thin lengths in the last several passes, the time interval will cause temperature differentials of such magnitude that the consequent gage differential must be anticipated in mill settings in order to retain the gage within tolerances. Differentials are minimized by shortening the time required for rolling.

Leveling (Flattening)—The amount of flattening required by the plate after leaving the rolls generally increases with decreasing thickness of the plate. The effectiveness with which flattening can be accomplished by a leveler increases with decreasing roll diameters and spacings and with increasing temperatures. For light-gage plates, therefore, a roller leveler with small-diameter rolls and with close roll spacings should be located near the mill finishing stand.

Heavier-gage plates, at corresponding temperatures, require greater strength and rigidity of rolls than lighter-gage plates. The heavier gages usually are delivered from the rolls at higher finishing temperatures and require accelerated cooling prior to leveler entry or the leveler must be located at a greater distance from the finishing stand to permit proper cooling before leveler entry. A compromise in roll diameters and leveler location may be satisfactory if the range of gages produced does not include extremes.

Levelers with small-diameter bending rolls backed up by short, rigidly supported backing-up rolls are used for cold releveling. They provide the requisite combination of severe bending and roll rigidity.

Cooling—Plates delivered from the roller leveler must

be cooled uniformly to avoid localized stresses that again would set up permanent localized distortions. As more heat is given up by the plate on contact with another metallic surface than by exposure to the atmosphere, it is necessary that cooling conveyors be so constructed that only momentary, staggered contacts are made with the bottom surface of the plate and cooling of the bottom side is effected primarily by radiation, similarly to the top surface. This cooling condition should be maintained until temperatures are reached at which the plates are no longer susceptible to distortion as a result of non-uniform cooling.

Shearing—All plates are sheared above atmospheric temperature in normal production. The most important factor in shearing plates to the desired size is the proper allowance for shrinkage. The cooling rate of plates decreases as they approach atmospheric temperature, consequently, extremely long cooling lines would be required to permit plates to cool sufficiently for accurate shearing to be done without the necessity for making allowance for shrinkage.

Manual layout of plates for shearing introduces some degree of deviation from theoretical accuracy. Mechanical layout machines and calibrated dials integral with the shearing equipment reduce the frequency and extent of deviations. The fundamental characteristics of the shearing equipment itself influence the degree of accuracy that can be attained consistently.

Identification, Inspection and Loading—Except for material in the lighter gages or narrower widths, each plate is marked by hand-stamping, painting or writing with chalk to show the heat number and any other necessary identification marks. It also is inspected for possible defects. If the order calls for special treatment such as annealing or other forms of heat-treatment, the plates are sent to the heat-treating department. Samples cut from the plates are usually subjected to the specified mechanical tests. If the plates meet the requirements, they then are loaded in railroad cars in accordance with standard loading practice.

SECTION 3

DESCRIPTIONS OF TYPICAL EXISTING PLATE MILLS

This section will describe several existing installations of plate mills of different types, beginning with a description of a three-high plate mill and its auxiliaries. The principle of the three-high mill and its advantages over earlier types has already been discussed in Section 2.

THREE-HIGH PLATE MILL

The 160-Inch Mill at Gary Works—The 160-inch plate mill at Gary Works, Figures 27—1 and 27—2, is typical of the single-stand three-high plate mills currently producing plates in a wide range of sizes. The furnace building parallels a conditioning yard which also is the delivery end of the primary mills supplying slabs to the plate mill. Slabs vary from 2½ to 24 inches in thickness, 50 to 66 inches in width and 57 to 130 inches in length with the weight ranging up to 44,000 pounds. Plates produced vary from ¼ to 14 inches in thickness, 48 to 148 inches in width and up to a maximum length of 720 inches. The maximum width of 148 inches is confined to the ¾- to 2-inch thickness range and is decreased progressively with lighter as well as heavier thicknesses. The maximum length of 720 inches is confined to the ½- to 3-inch thickness range and also progressively decreases with thicknesses outside of that range.

A minimum gage of ¼ inch rolled to a maximum width of 132 inches and a maximum length of 300 inches can be rolled satisfactorily for only a short time when roll conditions are ideal. A thickness of ⅝ inch is considered the minimum gage with rolls in the average condition.

The reheating furnaces are serviced by four 15-ton-capacity charging and drawing cranes with revolving mast, cage, and peel. The mill is serviced by ten in-and-out batch-type furnaces. Five of the furnaces are of the regenerative type, fired alternately from each end. These furnaces are controlled manually and, under continuous mill-operating conditions, develop a maximum heating capacity of 10 tons an hour per furnace at a fuel rate of 3,440,000 Btu per ton. One furnace, with a capacity of 8 tons per hour, is a continuous, recuperative furnace with a division wall dividing the hearth and the furnace into two sections, operating independently. The remaining four furnaces are of the recuperative design with full automatic control. These furnaces are divided and operate as two independent sections, similar to the furnace previously described. Capacities of 12 tons an hour per furnace were developed under the same conditions as previously stated, with a fuel consumption of 2,190,000 Btu per ton. All furnaces are

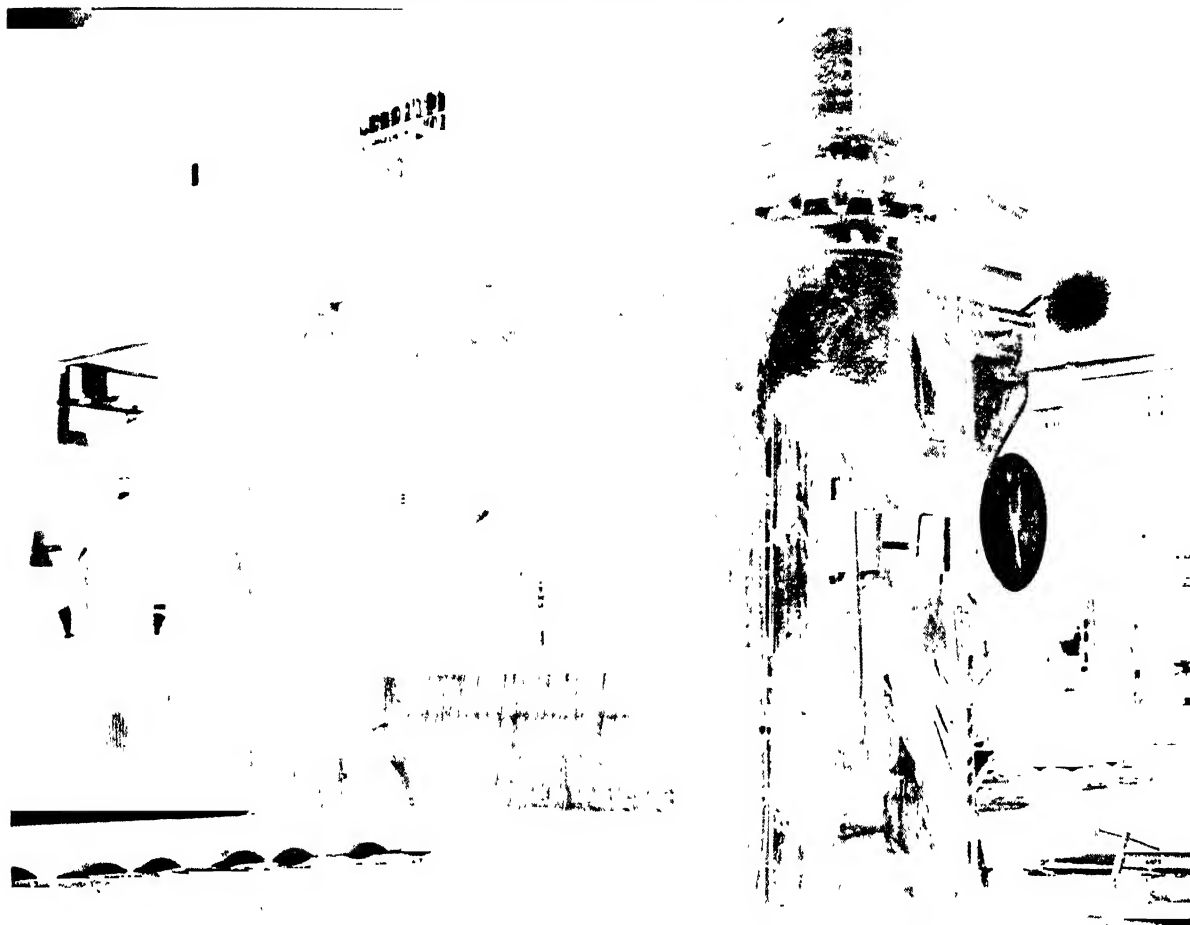


FIG. 27—1. The single-stand, three-high, 160-inch plate mill at Gary Works, showing the tilting tables at the entry and delivery sides.

fired with coke-oven gas as the primary fuel, with oil available as an alternate fuel.

Slabs are drawn from the furnaces and placed on a transfer car, which operates on trackage running the full length of the furnace building, and delivers the slabs to the mill approach table. This table extends toward the mill at 90 degrees to the furnace building. Thirty table rollers with 24-inch spacings convey the slab to the entry tilting table. Twenty-two table rollers with 21-inch spacings constitute the entry mill table which conveys the slab to the mill, and alternately receives the stock at the elevation of the top pass and re-enters the stock into the mill at the elevation of the bottom pass. The delivery mill table is of identical design and performs an identical function in addition to conveying the plate away from the mill after the final pass.

The top and bottom rolls, made from either grain or chill iron, are 44 inches in body diameter, 164 inches in body length, 30 inches in neck diameter, and 29 inches in neck length, and are turned with a $\frac{1}{64}$ -inch crown. Middle rolls made from chill iron are of 28-inch body diameter, 164-inch body length, 18-inch neck diameter, 29-inch neck length, and are turned with variable crown from $\frac{1}{64}$ to a maximum of $\frac{3}{32}$ inches.

The main roll necks are journalled in babbitt bearings with bronze inserts, and the middle roll necks are journalled in brass bearings. All roll-neck bearings are lubricated by an automatic forced-feed lubrication sys-

tem. The top roll is positioned through motor-driven screws and is balanced hydraulically. The middle roll is actuated through a leverage system and is balanced by counterweights. The middle roll is changed with the overhead crane and requires one hour for the operation. Main rolls are changed with a buggy-type roll-changing device, and a complete roll change requires an eight-hour period.

The main drive is a 7000-horsepower, 82-r.p.m. motor. Power is transmitted through a spindle to the middle pinion of a conventional three-pinion stand with pinion pitch diameters of 38 and 48 inches. Connecting spindles from the top and the bottom pinions transmit the power to the top and the bottom rolls.

Ninety-degree turning of the slab for width extension and turning again for resumption of length extension is accomplished by the manual operation of a large hook hinged on the entry side of the mill housing. Salt spread by hand on the slab surfaces in the initial passes, followed by high-pressure spray impingement in later passes, is the method of scale removal employed. Hydraulic pressure is supplied by motor-driven centrifugal pumps. Spray nozzles are directed at the bottom plate surface on the entry side, and on the top surface on the delivery side.

A 40-roll table conveys the plate from the delivery tilting table to the roller leveler. The leveler is a 9-roll machine with 5 bottom and 4 top rolls of 18-inch diameter and 13-foot 10-inch body length. The rolls are

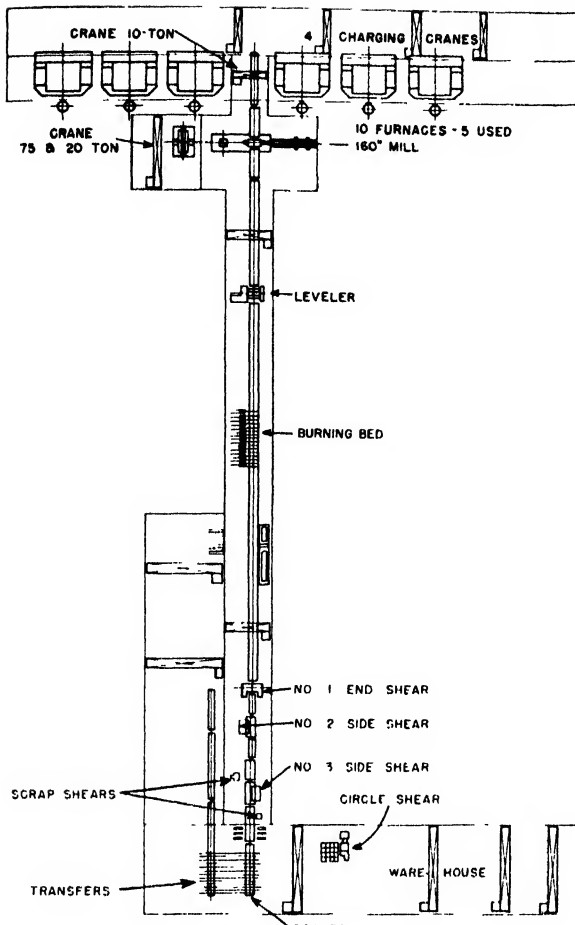


Fig. 27—2. Schematic layout (not to scale) of the units comprising the 160-inch plate mill at Gary Steel Works.

spaced on 18¼-inch centers, and the top rolls may be adjusted individually in a vertical position through a selective screw-down assembly mounted on the housing. The unit is driven by a 250-horsepower, 750-r.p.m. motor reduced through gearing to drive the rolls at 40 r.p.m., equivalent to a surface speed of 200 feet per minute. Hydraulic sprays operating at a pressure of 500 pounds per square inch on the entry side serve to remove secondary scale as well as to increase the cooling rates of heavy plates which are passed back and forth through them.

The plates are delivered from the leveler over a table of rollers to a continuing roller table. At a distance of approximately 425 feet from the mill center line is the beginning of a plate-marking machine which operates parallel with the adjacent conveying table. The sheared width is inscribed as two parallel lines on the plate. The plate lengths are measured off manually, but also inscribed on the plate by a transverse motion of the machine.

From the marking machine, the plate is moved to a hydraulically operated, down-cut guillotine end shear which has a 166-inch opening between the housings. The plates are then conveyed to side shears on opposite sides of the table. The side shears are similar in construction and thickness capacity to the end shear, with the exception that these shears can make a cut of 184-inch length per stroke. All three shears are equipped with magnetic holddowns and auxiliary guillotine

shears for side- and end-scrap disposal. The side-shear tables also are equipped with positioning magnets for properly positioning the inscribed lines on the plates along the knife edges. Unloading tables and beds on the delivery side of the second side shear facilitate the removal of plates which require further processing in the stock building. Plates for direct shipment continue on to a transfer which carries them to a table line, which in turn conveys them into a shipping building parallel with and adjacent to the mill building. The end 125 feet of this table consists of four unloading-table sections, which have tilting arms that raise and permit the plate to slide on skids. A gravity-feed bed is located at the end of this building area which permits returning plates from the storage area to the shear roller line for further processing. Completely processed plates are moved either into designated floor locations for accumulation of carload lots or directly into railroad cars.

Auxiliary processing facilities are located in a stock building, at right angles to and adjoining the mill and the shipping buildings. A rotary-type, motor-driven circle shear is provided for cutting circles ranging from 24 to 150 inches in diameter in a ½-inch maximum thickness. A brick-lined, gas-fired, slow-cooling pit is provided for retarded cooling of plates when the grade and dimensions make this necessary. A gas-fired, car-type furnace also is provided for annealing, normalizing, stress-relieving, and preheating for flame cutting. This furnace has four heating zones, with individual temperature controls. Two cars of 120-ton load capacity are provided in order that one car may be in the process of loading or unloading while the other car is in the furnace. A total of eight burning beds, three of which are equipped with two gantry cranes each, auxiliary equipment for flame cutting, and high-frequency electric hand-grinders constitute some of the additional available auxiliary processing equipment.

FOUR-HIGH REVERSING PLATE MILL

Homestead District Works 160-Inch Four-High Mill—The 160-inch mill at Homestead District Works, Figure 27—3, has a cold-slab storage yard parallel and adjacent to the slab-conditioning yards of the 45-inch slabbing mill. The gross area of 51,700 square feet provides ample storage area for stocking approximately 15,000 tons of slabs in accordance with a predetermined rolling sequence. Six gas-fired preheating pits are located in this yard; each has a capacity of approximately 10 slabs (50 tons) and is capable of heating cold steel to a temperature of 1200° F in 24 hours. Three gas-fired pre-heating hoods, each with a capacity of approximately 120 tons, can heat cold steel to a temperature of 1600° F in 24 hours. Slabs are transferred to the reheating-furnace areas by two standard-gage shuttle-car transfers, powered by 25-ton, Diesel-electric locomotives.

The slab-heating equipment consists of two continuous-type furnaces and two batch or in-and-out type furnaces. The batch-type furnaces are designed to heat slabs of a maximum size of 144 inches long by 20½ inches thick. They are two-zone-controlled, end-fired, recuperative-type furnaces with an effective hearth area of 12 feet by 46 feet. They are designed to operate on a mixture of coke-oven and natural gas and have a rated heating capacity of 15 tons per hour from cold to 2250° F. Each furnace is provided with two doors composed of three sections without jambs, and each section may be operated independently. Either half of the furnace front may be opened with no obstruction other than the center jamb. These furnaces are served by a 20-ton, stiff-legged, 3-swivel charging crane which charges the cold slabs from the transfer car into the

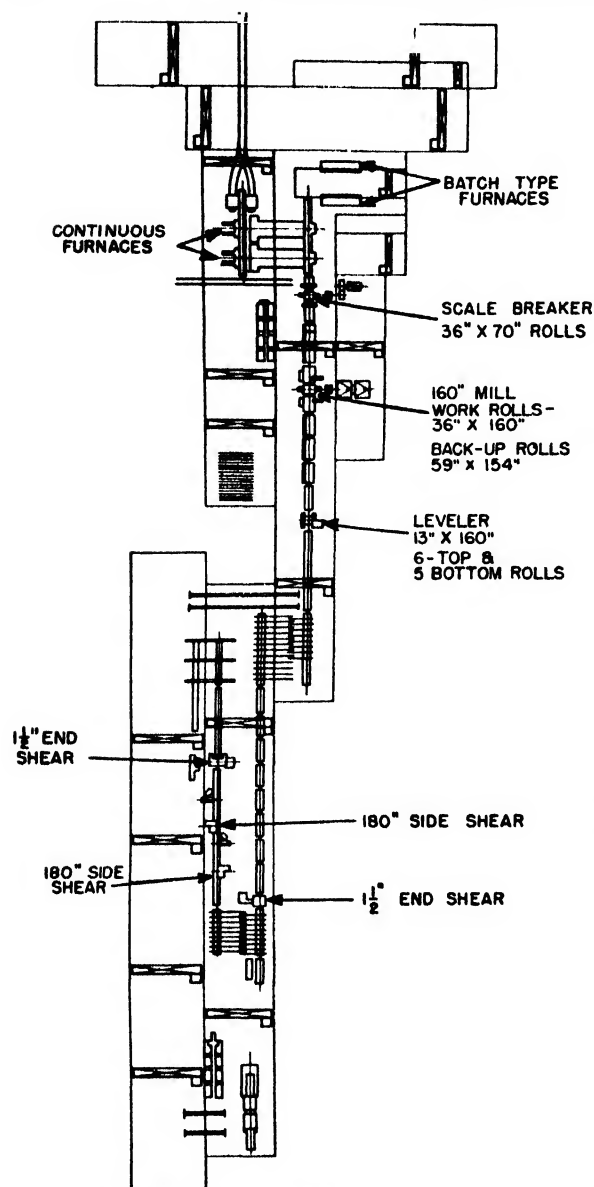


FIG. 27—3. Diagram (not to scale) of the physical layout of equipment comprising the 160-inch plate mill at Homestead District Works.

furnace and draws the heated slabs and delivers them on to the furnace delivery table.

The continuous-type furnaces are designed to heat slabs from 3 to 12 inches thick, 44 to 60 inches wide and 70 to 120 inches long. They are conventional two-zone-controlled, triple-fired, end-charged-and-discharged, recuperative-type furnaces and have an effective heating length of 80 feet and width of 23 feet. The top and the bottom burners of the primary heating zone are designed to burn a mixture of coke-oven and natural gas as a primary fuel and also can burn fuel oil as an alternate fuel. Only the gas mixture is burned in the holding zone. The heating capacity of each furnace is approximately 70 tons per hour from cold to 2250° F with full hearth coverage.

The continuous furnaces are served by a 30-ton electric overhead traveling crane which takes the slabs from the transfer cars by magnet or slab-handling tongs, and

places them on the furnace charging table. The slabs are fed through the furnaces by double rack-type pushers arranged to feed two rows of slabs through each furnace (Figure 27—4). Slabs are discharged from the delivery end, and slide down the furnace dropout skids to the furnace delivery table against spring-backed bumpers.

Heated slabs are conveyed on a roller table from the furnaces to a scalebreaker stand (Figure 27—5) which is a two-high mill with fluted alloy-steel rolls of 36-inch body diameter, 70-inch length, operating in water-lubricated composition bearings. Additional lubrication is provided by shots of water-soluble oil at 20-minute intervals timed by an automatic timing device.

The top roll is balanced by carrier bars supported from a yoke which is actuated by hydraulic jacks and accumulator to hold the top roll against the housing screw. The screws are operated by 50-horsepower motors through worm reduction gearing at a lineal vertical speed of 11 inches per minute. The rolls are changed at approximately one-month intervals under normal operating conditions. Approximately four hours are required to make the roll change with the use of a "C" hook and an overhead crane.

The mill is driven by a 1600-horsepower, 375-r.p.m. motor. Power is transmitted through a 15.68-to-1 double-reduction flywheel gear drive, a universal leading spindle, mill pinions and universal mill spindles, resulting in a lineal roll speed of 295.6 feet per minute. A meter in the pulpit gives the operator a reading of the opening between rolls at all times. Drafts of ½ inch may be taken on this stand, and the passage of the slab is guided to and from the mill by twin adjustable side-guides. Primary descaling is completed on the delivery side of the mill as the slab is passed through top and bottom hydraulic sprays operating at a pressure of 1,500 pounds per square inch.

The 160-inch reversing-mill stand (Figure 27—6) is separated from the scale breaker by a distance of 160 feet, spanned by 40 feet of 72-inch-wide table and 120 feet of 168-inch-wide table. All table rollers in this mill are journaled in antifriction bearings. A lift-and-turn table device is located on the reversing-mill side of the junction between the narrow and the wide table sections. The turntable is elevated and lowered by oil-hydraulic pressure and turned by a motor-operated rack and pinion. This permits either straightaway or broadside slab entry into the reversing mill. When spreading is required in the first pass, the slab may be entered following the preceding plate without a turning delay.

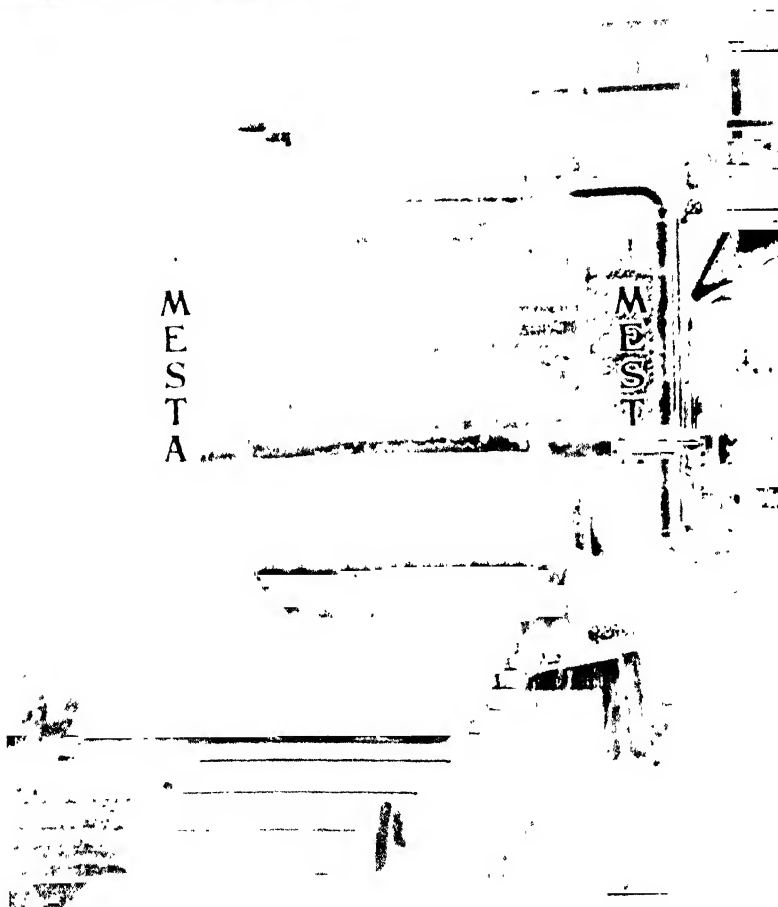
The crowned (0.024-inch maximum) alloy-iron work rolls of the four-high reversing stand have a body diameter of 38 inches, a body length of 160 inches, a neck diameter of 26.991 inches, and a neck length of 39 inches. The necks are fitted into the inner races of four-row tapered roller bearings. The alloy-steel backup rolls have a 59-inch body diameter, a 154-inch body length, a 33-inch neck diameter, and a 52-inch neck length. The roll necks operate in oil-film bearings at 17-pounds-per-square-inch oil pressure, through which the roll-separating force is transmitted. Thrust is taken by two-row steep-angle tapered thrust bearings. The top rolls are balanced by hydraulic jacks mounted in the bottom backup chucks, with the hydraulic system operating at 3700 pounds per square inch. An automatic forced-feed lubricating system is connected with all wearing surfaces on the mill other than the oil-film bearings.

Three sets of work rolls are used in the course of a normal weekly rolling schedule. Approximately one



FIG. 27-4. General view of the charging end of continuous-type reheating furnaces supplying hot slabs to the 160-inch plate mill at Homestead District Works.

FIG. 27-5. Close-up view of the scalebreaker stand of the 160-inch plate mill at Homestead District Works.



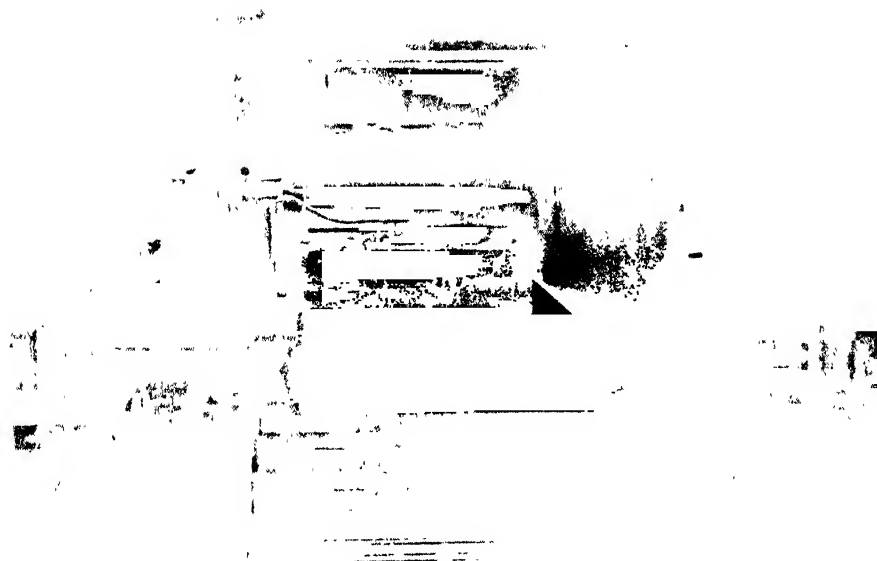


FIG. 27-6. The 160-inch reversing mill stand of the 160-inch plate mill at Homestead District Works, showing powered side guards for squaring and centering slabs and holding plates during the rolling operation.

hour is required for a work-roll change, which is accomplished by an overhead crane-suspended "C" hook, equipped with a motor-driven shifting device that positions the hook suspension over the center of gravity of the unloaded or loaded hook. The average time between complete roll changes, including the backup rolls, is three weeks. The complete roll change requires approximately ten hours, with the backup rolls being moved in and out of the mill on a motor-driven slide.

The main mill drive consists of two 5000-horsepower, DC, reversing motors, each directly connected to a work roll through universal spindles 36 feet long. The motor speed range is from 40 to 80 r.p.m., which provides a roll surface speed of 400 to 800 feet per minute. The mill screws are operated by two 150-horsepower mill-type motors, supplied by a variable-voltage motor-generator set. A 13-pass, 2-schedule, pre-set, automatic screw-down control, with optional manual control, is provided for this mill. Provision is made to set up one schedule while another is being run. The control is operated by the screw-down operator, who also reverses the mill drive and who can make manual adjustments to the screw settings by interrupting any automatic setting without affecting the accuracy of the following automatic stop. The mill is capable of producing the minimum gage of $\frac{3}{16}$ inch to sheared widths of 120 inches. Auxiliary facility limitations restrict the sheared-plate gages to a $\frac{3}{16}$ to $1\frac{1}{2}$ -inch range, widths to a $38\frac{1}{2}$ to 144-inch range and lengths to a 720-inch maximum. Flame-cut plates range from $1\frac{1}{2}$ to 15-inch thickness and a maximum of 144 and 480 inches in width and length, respectively.

The front and the rear mill tables are 33 feet long and have tapered table rollers with a taper of 1 inch in 14 feet. Even-numbered rollers all are tapered in the same direction and driven as a unit. Odd-numbered rollers are tapered in the opposite direction and also are driven as a unit. By rotating the alternate roller units in opposite directions, it is possible to turn a slab 90 degrees as desired. Heavy, powered side-guards are on each side of the mill for squaring and centering the slabs and holding the plates during the rolling opera-

tion. Hydraulic sprays on both sides of the mill, operating at a pressure of 1400 pounds per square inch, are utilized for top and bottom secondary scale removal.

The remainder of the 216-foot distance between the center line of the four-high mill and the leveler is occupied by a conveying table. Top and bottom water sprays span 180 feet of this distance to facilitate the cooling of heavier-gaged plates prior to leveler entry and to retard secondary scale formation. A section of the table on each side of the leveler is designed to permit plate removal by a "C" hook suspended from an overhead crane. The plates may be transferred to either one of two motor-driven transfer cars and subsequently transferred to the cooling area of the adjacent flame-cutting building or to the controlled-cooling facilities located in the shipping yard.

The leveler has two 16-inch diameter by 160-inch long entry pinch rolls and 6 top and 5 bottom 13-inch diameter by 160-inch long bending rolls. The machine is driven by a 250-horsepower, 400 r.p.m. motor, and positioning is accomplished through a selective screw-down assembly which is equipped with drum indicators located at all four corners of the leveler housing. The leveler can level structural-grade plates of $1\frac{1}{2}$ -inch thickness and can be raised to a maximum passage opening of 6 inches.

One hundred forty feet of the 275-foot distance from the leveler to a transfer stop is part of a chain-transfer cooling bed. This grid-type bed spans a distance of 120 feet and delivers the plates from the mill building onto a 292-foot disc-roller marking table located in the shear building. A motor-propelled automatic measuring and marking machine traverses a 140-foot distance parallel with this table. Indexed longitudinal and transverse movements enable the operator to measure and mark off mechanically any desired size and shape of a rectangular plate, and scribe a reference line which is used by the side-shear operator in the trimming of the side scrap. Beyond the marking machine, all required identification stamping and painting of the plates and test coupons are performed manually prior to entry in the crop shear.

The down-cut guillotine crop shear with knife blades 160 inches long and a knife rake of $\frac{1}{2}$ inch per foot is capable, in common with the main line shear units, of cutting 1½-inch gage to the maximum width produced. Powered mechanical manipulators are available for the square positioning of the plate. Oil hydraulic hold-down gags operating at 1700 pounds per square inch furnish positive clamping for the shearing stroke. The shear operates on the oil-hydraulic principle at pressures of 7000 to 9000 pounds per square inch, and has a self-contained pump with motor drive. The crop shear is used for dividing plates that are too long to pass over a second transfer, and for front-end shearing of heavy, wide plates, which must be free of shear bow.

Plates from the crop shear are delivered either to another chain transfer or to a 40-foot table section beyond the transfer, which is equipped with roll-off lifting arms and a side piling bed. This run-off and piling bed is utilized to divert flame-cut plates of such size and composition that will permit their conveyance to this point without the necessity of slow-cooling cycles. Plates to be finish-sheared are moved across the transfer to the transfer run-out tables, which double back through the shear building and are parallel to the marking line.

The main shear line consists of two side shears and one end shear. These shears are similar in design and construction to the crop shear, with the exception that the side shears have knives 180 inches in length. The side shears are staggered on either side of the table and are 78 feet apart. The side crop is sheared at the first side shear by positioning the plates to the reference line. A series of electromagnets with hydraulic vertical motion and screw drive transverse motion are utilized to position the plate so that the reference line is matched with a positioning line on the shear block. A mechanical width-gage guide is provided at the second side shear, which facilitates the positioning of the plate by working against the finished sheared edge; and its position in relation to the shear knife is recorded on a dial located on the shear housing. The end shear, located 84 feet from the second side shear, is a duplicate of the crop shear and is equipped with squaring pushers located on the entry side, and two motorized gage stops on the delivery side, which gage plates mechanically to predetermined lengths. One stop is used for short and the other for long lengths in the 60- to 720-inch sheared-length range.

The crop and end shear equipment consists of 150-horsepower motor-driven guillotine-type scrap shears capable of cutting a 33-square-inch cross-sectional area. A butterfly chute arrangement diverts test coupons, "block sheared," to a test box in which they are periodically delivered to a test-cutting shear for final handling. The test-cutting shear is also a motor-driven guillotine-type shear with a capacity of 27 square inches of cross-sectional area. The side shears are provided with 75-horsepower motor-driven alligator-type scrap shears with a capacity of 30 square inches of cross-sectional area. Side scrap is fed to the scrap shears by conveyors located below the table level, and the cut scrap is guided to removal buckets through gravity chutes.

Plates are delivered from the end shear to a 720-inch scale table, where they are weighed individually. They then proceed to a table section spanned by two transverse, selective, electromagnetic-type piling cranes. One crane can handle a 360-inch plate length, and the two cranes working together can handle a 720-inch plate length. Lifting force of the magnets can be regulated at will to provide a wide range in the lift combi-

nations. The cranes transfer the plates into the shipping building and deposit and pile them on gravity conveyor tables. The piles are moved by a conveyor-chain dog into positions from which they can be picked up by the overhead shipping cranes.

The shipping building is 110 feet wide by 1,100 feet long and is equipped with a shipping track extending its entire length. A connecting spur track of an 8-car capacity is located in an adjoining lean-to section of the building. This track is not serviced by overhead cranes and is used for preparing cars which require special blocking, either prior to or after loading. The shipping building is serviced by four 30-ton electric overhead traveling cranes of the double-hoist type. This building also is equipped with controlled-cooling hoods. Two additional inlets into this yard, other than the piling cranes just mentioned, are provided in the form of motor-driven transfer cars located at the extreme ends of the building. One car delivers directly from the mill building, and the other car delivers from the flame-cutting unit located in the end of the shear building.

The flame-cutting unit, located at the end of the shear building, is equipped with a flame planer, three pantographs, and sketch-cutting machines. The flame planer is designed primarily for side and end trimming as well as splitting rectangular-shaped plates. The pantograph machines are designed primarily for intricate shape cutting, although they can be used efficiently for cutting rectangular shapes. Magnetic tracing mechanisms, traveling over templates or operated manually over sketch drawings, guide the torch through the course desired.

Another flame-cutting unit is located in a building parallel and adjacent to the mill building. A continuation of this building merges with the end of the continuous-furnace building, and the two 30-ton cranes servicing this unit operate on a common runway with the crane servicing the continuous furnaces. This unit is equipped with three pantograph sketch-cutting machines, two large heat-treating ovens, and a heat-treating pit. The heat-treating facilities are used extensively in preheating, stress-relieving, normalizing and the controlled cooling of flame-cut products requiring these treatments. The heat-treating ovens also can be used for preheating slabs prior to charging in the slab-heating furnaces when such practice is necessary. Cooling skids are provided in this area for heavy-gage plates which are transferred hot from the mill by a 200-ton transfer car previously mentioned.

The greasing system for all table bearings, spindles, spindle carriers, jacks, slide joints, etc., is of the automatic "single-line" reversing type and contains eleven major pumping units.

Oil lubrication for the majority of the equipment, such as table line shafts, gears and bearings, leveler drives, screw-down drives, reduction gears, pinions, backup-roll bearings, and motor-room equipment, is accomplished by circulating systems of the single- and double-tank types.

SEMI-CONTINUOUS PLATE MILL

The 100-Inch Semi-Continuous Plate Mill at Homestead District Works—The 100-inch semi-continuous plate mill at Homestead District Works (Figures 27-7 and 27-8) has a slab-storage yard with an area of 66,000 square feet, having a slab-storage capacity of 18,000 tons stocked in rolling sequence. It adjoins two buildings of corresponding dimensions and area in which slabs produced by the 45-inch slabbing mill are cooled and conditioned. Two gas-fired, insulated, fabricated-steel preheaters, each having a capacity of 70

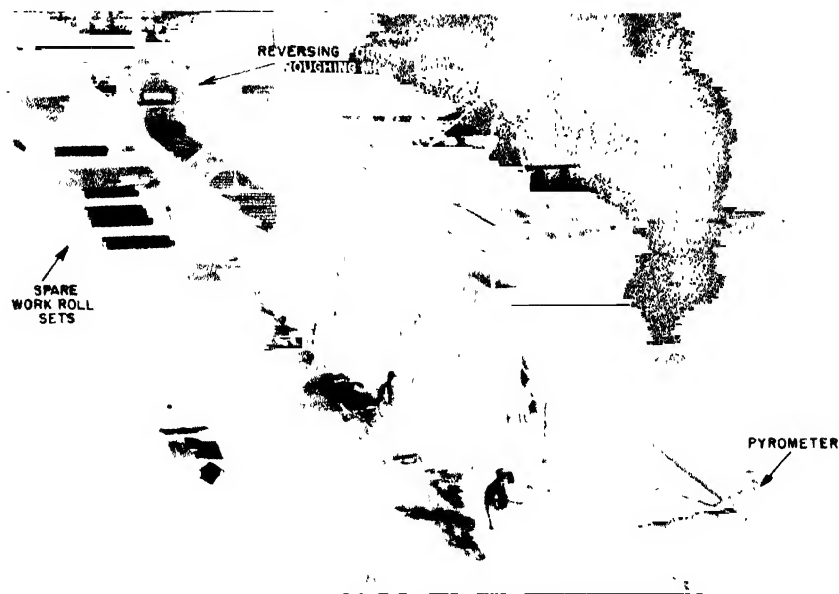


FIG. 27-7. General view of the 100-inch semi-continuous plate mill at the Homestead District Works.

tons, are located in this yard. Slabs are placed in lifts on a yard table which delivers them to a magazine feeder located in the slab-storage yard which serves as an intermediate unit between the yard and the furnace tables. These tables, as well as all others in this mill, are equipped with anti-friction bearings.

The feeder raises each pile of slabs to the level of the furnace charging table and discharges them one at a time onto the furnace table by an endless chain pusher. The slabs then are positioned in front of the furnace doors for charging by two pushers which can be operated independently or in unison for each furnace. The mill is served by four conventional, continuous, triple-zoned furnaces with non-metallic recuperators. The furnaces have an effective heating length of 76 feet, 10 inches. A mixture of natural and coke-oven gas is the primary fuel, although fuel oil can be burned as an alternate. The furnaces are equipped with combustion and temperature controls. They have heated, on a continuous basis, 68 tons an hour per furnace from cold to 2250° F. Slabs charged vary from 24 inches to 60 inches in width, 3¼ inches to 12 inches in thickness, and 65 inches to 92 inches in length.

The slabs slide down the furnace dropout against inclined spring-loaded bumpers or an improved shockless bumper onto the furnace tables. A skew table with movable side guards permits straightaway entry or may be used as a turnaround for broadside entry to the scale breaker. This is a two-high stand with smooth rolls, each having a 36-inch diameter body, 100 inches in length, with 23⅞-inch neck diameter by 26¾-inch neck length, operating in bronze-insert babbitt bearings. The mill is driven by a 1000-horsepower, 500-r.p.m. motor, transmitting power through a 24.4-to-1 ratio reduction gear and conventional pinion stand. Top and bottom descaling sprays operating at 1000 pounds per square inch are located at the delivery side of the scale breaker and at the entry sides of the reversing rougher and the first finisher.

The distance between scale-breaker and broadside stand is spanned by a conventional roller table into which is incorporated an electrically-operated lift-and-turn platen at the broadside entry. The broadside mill is a four-high nonreversing stand. Work rolls are of 42-inch body diameter, 120-inch body length, 30-inch

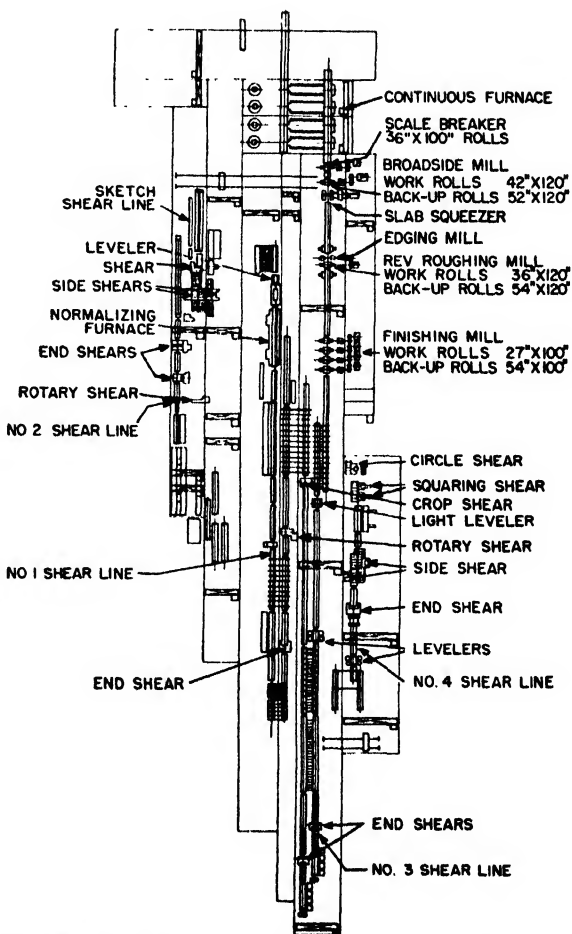


FIG. 27-8. Schematic arrangement (not to scale) of the production units comprising the 100-inch semi-continuous plate mill at Homestead District Works.

neck diameter, and 34-inch neck length. Backup rolls have a 52-inch body diameter, 120-inch body length, 36-inch neck diameter, and 44 $\frac{1}{2}$ -inch neck length. Both work and backup rolls operate in bronze-insert babbit bearings. The mill is driven by a 4500-horsepower, 370-r.p.m., AC motor, transmitting power through a 22-to-1 gear ratio and conventional pinions to give a lineal speed of 185 feet per minute at the work-roll face. Movable side guards on the entry side permit centering of the slab. Pushers on entry and delivery sides facilitate entry and return for another spreading pass if required.

A duplicate of the turn-around on the entry side is located on the delivery side in the table leading to the slab squeezer. The squeezer is a two-ram width-sizing machine located 30 feet from the broadside stand. The rams are supported above the mill table rollers and are positioned for the squeezing stroke in conformance with the plate widths being rolled. Forging power is supplied by a 500-horsepower, 500-r.p.m. motor, transmitted through gearing and mechanical linkage to actuate the forging rams. A hydraulic hold-down prevents bowing of the slab as the edges are subjected to the squeezing action.

The four-high reversing roughing mill is located 154 feet, 9 inches from the squeezer. On the entry side of this mill are movable side guards which guide the slab into a vertical edging mill. The vertical rolls have a 40-inch diameter and an 11-inch face with a 5-degree downward taper. They are driven by a 600-horsepower, 125 to 406-r.p.m., variable-speed motor. The work rolls of the reversing rougher are of 36-inch body diameter, 120-inch body length, 0.008-inch crown, 24-inch neck diameter, and 21 $\frac{1}{2}$ -inch neck length. The backup rolls are of 54-inch body diameter, 120-inch body length, 31-inch neck diameter, and 32 $\frac{1}{16}$ -inch neck length. The mill is connected directly through mill pinions to a 7000-horsepower, 40 to 80-r.p.m., variable-speed motor which provides a lineal speed range at the work-roll face of 376-752 feet per minute. Both work and backup rolls operate in anti-friction roller bearings. Heavy ram-type guides on entry and delivery sides and a turn-around on the delivery side permit the finish rolling of plates wider than could be finished through the continuous stands.

The first of four, four-high, continuous finishing stands is located 172 feet from the reversing rougher. The four finishing-mill stands are exact duplicates of each other. The work rolls are of 27-inch body diameter, 100-inch body length with crowns varying from 0.004 inch to 0.008 inch, 20-inch neck diameter and 15-inch neck length. Backup rolls are of 54-inch diameter, 100-inch body length, 31-inch neck diameter and 32 $\frac{1}{16}$ -inch neck length. Both work and backup rolls operate in anti-friction roller bearings. The finishing stands designated as No. 4 to No. 7 are each driven by a 5000-horsepower motor. No. 4 to No. 6 stands have a speed range of 110-250 r.p.m. and No. 7 has a speed range of 125-265 r.p.m. Reducing-gear ratios for No. 4 to No. 7 are, respectively, 2.32, 1.77, 1.46 and 1.35. Corresponding lineal speed ranges at the work-roll faces are: 333-760, 440-1000, 530-1200 and 655-1386 feet per minute. Pull-back chutes with a variable opening up to a 100-inch maximum guide the rolled plate through the individual stands. Steam top-side blow-offs are provided between stands in addition to the hydraulic sprays at No. 4-stand entry.

The scale-breaker top roll and the top work and backup roll of all the other stands are balanced by hydraulic jacks and accumulators. Overhead cranes and counter-balances are used to change scale-breaker rolls and the work rolls of the broadside and the finish-

ing stands. A roll-changing rig is used to change the reversing-rougher work rolls, and the broadside, the reversing-rougher, and the finishing-stand backup rolls.

The first 60 feet of the 280-foot run-out table is equipped with a series of cooling sprays. The far portion of the table is a part of a transfer, over the cooling grids of which the plates are moved laterally by a rope-and-carriage transfer 162 feet long and 18 feet wide. Located immediately beyond the transfer is the light leveler with 17 bending rolls of 8-inch diameter and 108-inch length. A cooling table with individually-driven disk rollers spans the 310-foot distance between the light and the heavy levelers. The heavy leveler is an 11-bending-roll machine with 12-inch-diameter rolls 108 inches long. After leveling, the plates may progress through either one of two alternate routes, to the No. 3 shear unit or toward the rotary-shear line.

Shear Units—The No. 3 shear unit consists of two similar shear lines, each one of which is a continuation of the dual transfer table. Each line consists of a shear approach table, a $\frac{3}{4}$ -inch by 100-inch motor-driven guillotine shear, a powered gage and gage table, and a "kick-off" table with a 60-foot stacking bed. A guillotine scrap shear of 16-square-inch capacity serves both end shears.

As all the kick-off tables and the stacking beds in this mill are of similar design, a brief description of one will suffice for all. The kick-off mechanism consists of a series of arms which, during a plate delivery from the shear, are positioned between and below the table rollers. The arms are of a double-bar design with small idler rollers free to rotate in the space between the bars. The stacking-bed ends of the arms are keyed to a pivot shaft which, on being rotated through a partial arc, will lift the arms correspondingly through a partial arc and permit the plates to slide down along the idler rollers.

The stacking beds consist essentially of a series of double beams between which movable stops may be shifted and secured in a position to correspond with the widths of plates to be received. A series of beams spanning about 20 feet is fastened to a bed frame and constitutes a stacking-bed section. The sections can be raised individually or in unison to receive the first plate of a stack and are then lowered in consecutive increments as the stack is built up. The stacks are removed from the beds by overhead cranes equipped with sheet carriers or spreaders.

The No. 3 shear unit is utilized for cutting sheet and light plate gages up to $\frac{3}{8}$ inch. The mill is capable of rolling material to thicknesses of approximately 0.10-inch minimum, 72 inches wide. The sheets and the plates sheared on the No. 3 unit generally are cut into multiples of the ordered lengths in order to keep ahead of the mill rolling rates. The mill has rolled a maximum of 2,137 tons in an 8-hour period. The multiple-length, side-untrimmed sheets and plates are transferred to the No. 4 shear unit by a 70-ton transfer car or by placing them on a gravity-feed table, which moves them under a magnetic depiler.

The No. 4 shear unit is housed in a building adjacent to and parallel with the mill building on the motor-room side. Two gravity conveyor tables, one located in the shear building and one in the mill building, move the stacks under a magnetic depiler. The stacks are depiled, and the plates are placed singly on the approach table of a backed-up roller leveler. The leveler has two 12-inch-diameter by 108-inch pinch rolls, nine 8-inch-diameter by 108-inch bending rolls and eleven 8 $\frac{1}{4}$ -inch diameter by 30-inch backup rolls. A marking

mechanism attached to the delivery side of the leveler scribes the ordered width on the plates.

The plates progress from the leveler to a $\frac{3}{4}$ -inch by 125-inch end shear equipped with a powered gage which can be set in the range of 48 inches to 510 inches. Delivery from the shear gage table is made to the caster bed of a left-hand $\frac{3}{4}$ -inch by 144-inch side-trimming shear and from there to the caster bed of a right-hand shear of otherwise identical design. A scale table and a kick-off table with a 60-foot stacking bed complete the main shear line.

Beyond the main shear line are located one $\frac{1}{2}$ -inch by 144-inch squaring shears and a circle shear serviced by gravity conveyors and a jib hoist. The circle shear has a 9-inch diameter by $2\frac{1}{2}$ -inch upper cutter and a $4\frac{1}{2}$ -inch diameter by $2\frac{3}{8}$ -inch, 45 degree-bevel lower cutter. It can cut 8-inch to 96-inch diameter circles from $\frac{3}{32}$ -inch to $\frac{3}{8}$ -inch, 0.30-carbon steel at a cutting speed of 50 to 100 feet per minute. A $\frac{3}{4}$ -inch by 30-inch guillotine scrap shear serves both the resquaring and the circle shears, and a similar shear serves each of the side shears.

Rotary Shear Line—No. 1 Shear Unit—Heavier-gage plates, after moving across the dual transfer, continue their travel in a reverse direction to the mill rolling direction, over spool-type marking and inspection tables toward a $\frac{3}{4}$ -inch by 100-inch crop shear. The distance traveled by a plate from the finishing stand to this crop shear is approximately 1,200 feet. The plates are cropped and are divided into multiples of the ordered lengths at this point and progress to a 123-foot by 41-foot roller-chain lift transfer located immediately beyond the shear. Required painting and stamping identifications are applied as the plates travel over this transfer.

Sketch plates and other plates which are beyond the capacity of the rotary shear are diverted from the delivery side of the transfer by a table extension and stacking beds. They are moved by overhead crane and transfer car to the No. 2 shear unit for cutting. Plates within the capacity of the rotary shear resume travel in the rolling direction toward that unit.

The rotary-shear approach table is equipped with magnetic manipulators to position the plates for the shear entry. The shear is a double-rotary, side-trimming shear driven by a 300-horsepower motor. The top knives are 15-inches in diameter by 2 inches thick, the bottom knives are 60-inches in diameter by $2\frac{1}{2}$ inches thick, and the scrap cutters are 12 inches by $3\frac{3}{4}$ inches by $1\frac{1}{2}$ inches. Scrap from each side is guided from the main cutters through chutes to the scrap cutters located below the large knives. The scrap is cut by a rotary, eccentric motion and is dropped through chutes into disposal buckets. The shear has a capacity for cutting 20-inch to 90-inch widths, $\frac{3}{32}$ -inch to $\frac{3}{4}$ -inch gage, of 0.30-carbon material at a cutting speed of 79 to 237 feet per minute. The cutters may be set to $\frac{1}{16}$ -inch increments within the width range.

The trimmed plate may be sent to either one of two end shears for final cutting to ordered length. The near or No. 2 end shear, located in the shipping building, is reached by traversing a 127-foot by 27-foot rope-and-carriage transfer located immediately beyond the rotary-shear delivery table. The far or No. 3 end shear approach table is immediately beyond the transfer referred to, and is reached by direct, continuous travel from the rotary shear. Both shears can handle material $\frac{3}{4}$ -inch thick by 100 inches wide, and each is driven by a 150-horsepower motor. They are followed by gage tables equipped with motor-operated, tilting, and traveling plate stop-gages with a 48-inch to 720-inch travel

range from the knife edge. The gage carriage travel speed is 50 feet per minute. Each end shear is serviced by a $\frac{3}{4}$ -inch by 20-inch guillotine scrap shear with a 15-square-inch cutting capacity.

The sheared plates from the No. 3 end shear are transferred into the shipping building by a 74-foot by 27-foot roller chain lift transfer. Travel direction is reversed to move over a scale table with an automatic weight-recording device and continue on to a 110-foot kick-off table and stacking bed. Plates from No. 2 end shear travel over a similar scale table and on to a 150-foot kick-off table and stacking bed. The kick-off table connects with the conveyor feed table of the continuous normalizing furnace. The shipping building in which the stacking beds are located is a 90-foot by 1,540-foot structure with a net shipping space of 60,000 square feet.

Continuous Normalizing Furnace—Plates to be heat treated in the normalizing furnace are placed in stacks on a gravity feed table which moves them under an unpiler. The unpiler is a motor-driven traverse bridge and hoist with five selective magnetic lifters having a total capacity of ten tons. It operates on a structural framework runway, 35 feet long with a 40-foot span. The runway extends over a 75-foot, pipe-roller, roller-bearing, gravity conveying table which also connects the No. 1 shear line kick-off table with the normalizing furnace.

The furnace is of the straight conveyor type, divided into five zones. All zones, with the exception of the entrance or heating zone, are equipped for recirculation to improve temperature uniformity. There is an alloy baffle between the heating and the recirculating zones. The furnace is 130 feet long, 9 feet wide, and the roof is 2 feet 10 inches above the conveyor. The conveyor consists of 5 sprocket-driven chains with vertical flights. A variable speed range with a maximum of 75 feet per minute is provided. Pressure burners for all zones, burning a mixture of coke-oven and natural gas, with temperature and fuel-air ratio controls, provide the heat input. A stack at each end of the furnace provides natural draft.

Plates emerging from the furnace may be quenched in a 20-foot quenching hood prior to leveler entry. The backed-up leveler is a duplicate of the one described in the No. 4 shear unit. Plates from the leveler traverse an 80-foot by 28-foot cooling bed and are stacked in conventional stacking beds at the far side of the traverse.

No. 2 Shear Unit—The No. 2 shear unit has two main shear lines, consisting of a rotary shear line housed entirely in the 80-foot by 860-foot shear building, and the sketch shear line, which has the feeding table and two shears in the shear building and one shear, scale table, and stacking beds in the No. 2 shipping building. The No. 2 shipping building, adjacent to and parallel with the No. 1 shipping building, is an 80-foot by 1,136-foot structure with a net shipping space of 55,000 square feet.

The sketch shear line, as its designation implies, is used to shear irregularly shaped plates. It also is utilized to shear structural-grade, rectangular plates heavier than $\frac{3}{4}$ -inch gage, and alloy plates of thinner gage which, because of composition, are beyond the cutting capacity of the No. 1 shear line. Such plates are diverted from the No. 1 shear line beyond the No. 2 transfer and moved to the No. 2 shear unit by transfer car, as previously described.

Plate stacks for the sketch shear line are placed on a 90-foot gravity feed table by an overhead crane. An overhead, selective, magnetic unpiler puts the plates singly on a marking and layout, five-chain conveyor

table, 140 feet long. Both sketch and rectangular plates are laid out manually on this table. The first shearing unit in the line, a 1½-inch by 110-inch end shear, cuts the plates to length. The plates may be turned 180 degrees on the caster beds immediately beyond and returned, if necessary, to have both ends front-end cut for the elimination of shear bow. A continued, manually propelled movement of 55 feet over caster beds brings the plate to a 1¼-inch by 144-inch side shear for edge trimming. A lateral movement of 44 feet into the shipping building brings the plate to an opposite hand but otherwise identical shear for the trimming of the opposite edge. Further manual propulsion opposite to the original travel direction moves the plate back to powered traction on a 35-foot scale table and a 60-foot kick-off table and stacking beds. Two scrap shears serve the three major shearing units in this line.

When the mill rolling rate on ¾-inch to ¾-inch gage plates exceeds the capacity of the No. 1 shear line, the excess multiple-length plates are diverted at the same point and are transferred to the No. 2 shear unit in the same manner as sketch plates. The plates are spread on a 160-foot chain-conveyor marking table and move over an approach table to a double-rotary side-trimming shear. In general design features, the shear is similar to the one in the No. 1 shear line. Shear-knife diameters are 14 inches and 58 inches for top and bottom, respectively. The capacity ranges from 30 inches to 96 inches in width and ⅜ inch to ¾ inch in gage at a cutting speed of 50 feet per minute. The scrap cutters are of the revolving-drum type. A duplicate of the end shear in the sketch shear line is located 71 feet from the rotary shear. It is followed by a 60-foot back shear table, a 30-foot scale table and a 60-foot kick-off table with stacking beds. A guillotine scrap shear of 32½-square-inch capacity serves the end shear.

Auxiliary shearing equipment in No. 2 shear unit consists of a resquaring shear, a circle shear, and a test-cutting shear. The squaring shear is a ⅜-inch by 144-inch, motor-driven unit. The circle shear, driven by a 110-horsepower motor, has a capacity ranging from 20-inch to 150-inch diameter, ⅜-inch to 1¼-inch gage, in structural grades at a cutting speed of 56 feet per minute. Both top and bottom knives are 16½ inches in diameter and 2 inches thick. The guillotine-type test-cutting shear has a capacity of 16 square inches, equivalent to cutting 20-inch by 1¼-inch thick plates.

An individual plate inspection unit is made available by feeding from the shear building and repiling in the shipping building. It includes two parallel sections of gravity table for plate entry and two parallel sections for plate delivery. Each section is 90 feet long with pipe rollers operating in roller bearings. Located between the gravity table lines is a motor-driven roller table over which individual plates are moved and are tilted for inspection by eight lifting arms to an angle of 75 degrees. The unit is served by one unloading unit and one piling unit. Each traverse and hoist, with five selective magnetic lifters, operates on a structural framework 40 feet long with a 40-foot span.

CONTINUOUS PLATE MILL

The 96-Inch Four-High Continuous Plate Mill at South Works—The four-high continuous plate mill which will be described in detail was placed in operation in the early part of 1931 and, therefore, represents an early stage in the development of the wide four-high continuous mills. The wide, four-high, continuous hot-strip mills, which have been placed in operation in the intervening period and which are described in another section of this book, are more representative of the

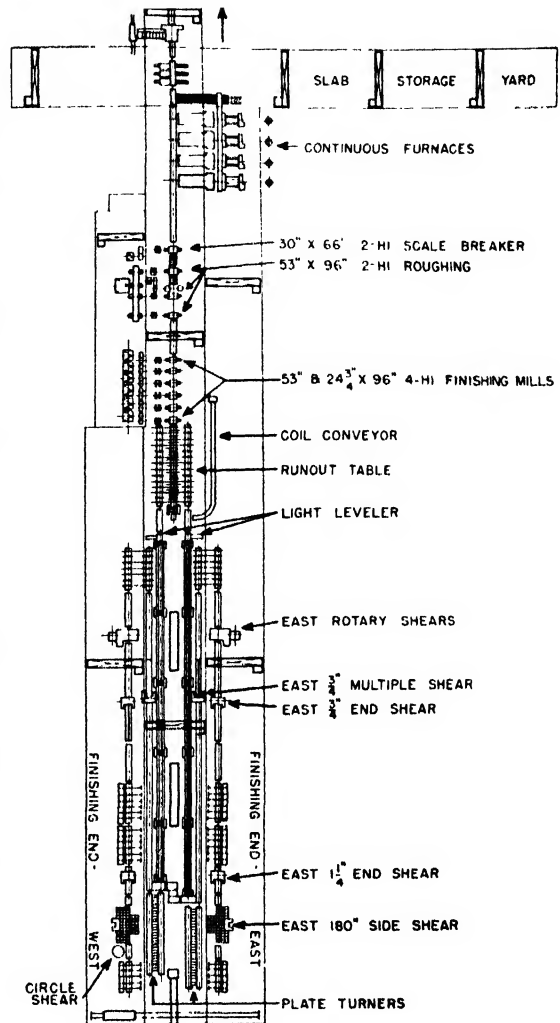


Fig. 27-9. Diagram (not to scale) of the layout of the mills and auxiliary equipment comprising the 96-inch four-high continuous plate mill at South Works.

potential productive capacities, the surface finish, and the width and gage uniformity which can be achieved with a modern mill of this type designed primarily for plate production.

The 96-inch continuous plate mill at South Works, Figures 27-9, 27-10 and 27-11, receives the bulk of its charge in the form of cold slabs from a multi-purpose slab-yard building running at 90 degrees to the mill center line and serviced by four 20-ton, single-trolley, single-hoist, overhead cranes. The 96-inch plate mill and its supplying unit, the 44-inch slab mill, have a common and continuous center line. The slab mill shear delivery and piler tables form a continuous table line with the 96-inch mill furnace and mill approach tables so that hot slabs may be table-conveyed directly from the slab-mill shears into the 96-inch mill. A pusher and chain transfer adjacent to No. 1 reheating furnace permits the diversion of hot slabs from the mill table and their transfer to the charging ends of the reheating furnaces. It also is utilized for the recharging of furnace kick-outs.

The slab mill has a rated capacity of 140,000 product tons per month. It supplies all of the slab requirements

FIG. 27-10. General view of the 96-inch continuous plate mill at South Works.



of the 96-inch mill, with the bulk of its excess productive capacity allocated to the slab requirements of the 80-inch hot-strip mills at Gary Sheet and Tin Mill, and a smaller percentage allocated to the 30-inch universal plate-mill slab requirements and semifinished trade-slab requirements. Virtually all the slab-mill production is removed from the run-out tables by three sectional pushers and pilers located in the slab yard. Slabs in ingot stacks are removed from the pilers by overhead cranes. Slabs for the hot-strip mills are placed in a cooling area west of the run-out table in conformance with the principle of utilizing fully the conditioning capacity of the western extremity of the slab

yard for this product. After cooling, the slabs are loaded into standard-gauge cars placed on tracks running transversely through the extremity of the building, for shipment to the conditioning yard of the 80-inch hot-strip mill at Gary.

All 96-inch plate mill slabs are placed in the central portion of the slab yard, adjacent to the pilers, for cooling. After cooling, they are conditioned, and piled in rolling sequence in an area approximately 70 feet by 100 feet located near the pilers. This area has a storage capacity of 5,000 tons. A portion of the slab tonnage produced for other applications and cooled in the central area also is conditioned here and is loaded



FIG. 27-11. General view of inspection and marking operations on plates at the delivery end of the 96-inch continuous plate mill at South Works.

out on standard-gauge tracks. Cool slabs in excess of the yard conditioning capacity are sent to other yards for conditioning.

The plate-mill furnace-charging tables extend into the slab yard. The slab charger is a side unpiler consisting of a skidded deck with a vertical screw motion and a pusher operating transversely to the table travel direction and at the table-top elevation. Slab stacks are placed on the unpiler deck in its down position, and the slabs are pushed singly on the charging table as the unpiler is moved upward in slab-thickness increments. The slabs are positioned at the furnace charging doors and are charged into the furnaces by double-row pushers which can be operated singly or in unison.

The heating equipment for this mill consists of four continuous-type, two-zone, triple-fired furnaces equipped to burn natural gas or fuel oil through all burners. One of the originally installed furnaces of 13 by 59 foot inside dimensions is manually controlled, is not equipped with either recuperators or regenerators and is only used for standby service. When this furnace is used, it is single-row charged with slabs of 83-inch maximum length. Three furnaces installed in 1950 are 16 by 60 foot 6-inch inside dimensions, are equipped with automatic heating controls and metallic recuperators. Slabs charged into these furnaces vary from 3½ to 6½ inches in thickness, from 28 to 60 inches in width, from 50 to 83 inches in length for a double-row charge and up to a maximum of 178 inches for a single-row charge. Each of these furnaces has a rated heating capacity of 50 slab tons per hour.

The two-high scale-breaker stand is located 156 feet, 3 inches from the center line of the near furnace. It is driven by a 600-horsepower, 488-r.p.m. motor, driving through a gear reducer and conventional pinions of 34-inch pitch diameter. The steel rolls of 20-inch neck diameter, 18¾-inch neck length, 30-inch body diameter, and 66-inch body length, operate in bronze-insert, babbit bearings. The top roll is counter-weight balanced. Drafts on this stand are limited to ⅜-inch maximum. Top and bottom hydraulic sprays operating at 1000 pounds per square inch are located on the delivery side of the mill.

The roughing train consists of three duplicate two-high stands with spacings of 35 feet between the scale breaker and No. 1 stand, 41 feet, 5.3 inches between No. 1 and No. 2 stands, and 32 feet, 2 inches between No. 2 and No. 3 stands. All three stands are driven by a 6000-horsepower, 370-r.p.m. motor, driving through a special gear-reduction set with two flywheels on the high-speed shaft and through conventional pinion stands. No. 1, No. 2, and No. 3 stands operate, respectively, at 8.5, 10.1 and 15.15 r.p.m., corresponding to lineal surface speeds of 116, 138 and 206 feet per minute. Only one slab can be undergoing reduction in the roughing train at any time. The cast-steel rolls for these stands have a 35-inch neck diameter, 36½-inch neck length, 53-inch body diameter, and 100-inch body length. The rolls operate in bronze-insert, babbit bearings. The top roll is balanced hydraulically. The screw-downs are operated by two motors tied in with a magnetic clutch so that they may be operated in unison for draft settings or individually for roll alignment. High-pressure hydraulic sprays are located at No. 1 and No. 3 delivery and at No. 4 entry.

Motor-driven sectional side guards on the skew roller table between the scale breaker and No. 1 stand may be positioned to permit either a straightaway entry to No. 1 stand or to form a turning pivot for a 90-degree turn and a broadside entry to No. 1 stand. Therefore, No. 1 stand may be utilized as either a straightaway or

a broadside mill. A rack-type carriage pusher operating on a structural framework above the table provides for a square entry for the broadside pass. Powered side guides at the entry of each roughing stand, as well as the first finishing stand, permit centered entry.

A similar arrangement of side guards and skew table on the delivery side of No. 1 stand makes possible either continued straightaway progress of the slab or a 90-degree turn after a broadside pass.

The finishing train consists of six duplicate four-high stands spaced on 21-foot centers, with the first or No. 4 stand spaced 72 feet from the center line of the No. 3 roughing stand. Each mill is driven by a 3500-horsepower, 165-330-r.p.m. motor through a gear-reduction unit, conventional pinion stand and wobbler-type spindles. The speeds of the work rolls of the various stands in r.p.m. and f.p.m. are listed below:

Stand No.	Speed Range	
	Rev. per min.	Ft. per min.
4 Finishing	20 to 49	128 to 315
5 Finishing	30 to 72	193 to 463
6 Finishing	39 to 94	250 to 600
7 Finishing	49 to 118	315 to 760
8 Finishing	54.5 to 132	350 to 850
9 Finishing	60 to 146	385 to 940

Grain-iron work rolls in No. 4 and No. 5 stands, and chilled-iron work rolls in the remainder of the stands, have 16⅞-inch neck diameter, 41½-inch neck length, 24½-inch body diameter, and 100-inch body length. The steel backup rolls have 27-inch neck diameter, 44¼-inch neck length, 52-inch body diameter, and 96-inch body length. Both work and backup rolls operate in anti-friction roller bearings. The top backup and work roll are balanced hydraulically as an assembly, with the work roll being held against the backup roll by two spring suspension take-up rods positioned through yoke extensions. Draft settings are made by 100-horsepower motors driving the screwdowns through worm reduction gearing. The two motors for each stand are connected by magnetic clutches and are operated in unison for draft settings and singly, when the stock is in the mill, for camber correction.

The spacings between the mill stands are each occupied by a two-sectional retractable table which in a normal operating position presents a continuous, smooth iron liner surface to plate travel. Adjustable, powered side guides center the plates for admission at the point of entry, and top and bottom stripper-guide assemblies are attached to the delivery section. A looping roll normally is positioned below the table surface and is raised in an arc by pneumatic cylinders when it is necessary to take up stock slack while speed adjustments are being made. Prior to a roll change, the two table sections are retracted so that the liners and the supports of one section overlap the other in the center of the spacing. In this position both the entry- and the delivery-guide assemblies are free of the rolls.

An overhead-crane-suspended sleeve, into which a work roll is fitted as a counterweight, is used for work-roll changes. The frequency of work-roll changes is extremely variable and is dependent on the particular roll stand as well as on the preponderance of product gages rolled. In the rolling of sheet gages, the leader and the finisher may require changing at four-hour intervals. When the schedule includes only plate rolling and the preponderance of the tonnage is in the heavier plate gages, the work rolls in No. 4 and No. 5 stands may last out the weekly schedule or require changing only once to meet wide-plate rolling requirements in

the latter part of the week. Work-roll changes are made in 25 minutes.

A cast counterweight secured on an integral arm and sleeve which fits over the roll-neck extension, is used for the changing of the backup rolls, whereas a sleeve and another roll as a balancer are used for changing the roughing rolls. The frequency of backup-roll changes is also variable and dependent on the roll-stand position and gages rolled. Since one roll is changed per week, the average time interval amounts to 12 weeks. No. 3 roughing rolls are changed each week, No. 2 at weekly or bi-weekly intervals dependent on schedules, and No. 1 at 3 to 4-week intervals. Backup-roll changes require an average of $2\frac{1}{2}$ hours per roll, and roughing rolls require $3\frac{1}{2}$ hours a set.

Product is delivered from the finishing stand onto a central runout table, 132 feet long, consisting of individually-driven disk rollers which protrude through openings in cast alloy-iron aprons. Plates are stopped on the runout table by raising the lifting aprons above the table rollers. The plates are moved 18 feet, 4 inches laterally over iron gridwork by a cable-carriage dog transfer to either one of two duplicate, parallel finishing lines.

The 126-foot leveler-approach tables convey the plates to the light levelers, the first processing units in each line. They are located immediately beyond the hot transfer and have seventeen bending rolls, 8 inches in diameter by 100 inches long. Delivery from the leveler is made to the first of five 125-foot sections of sprocket-driven chain spool conveyors. These conveyors span the distance to the heavy levelers and also serve as cooling and top surface inspection tables. The heavy leveler is a 13-roll machine with seven top and six bottom, 14-inch diameter by 100-inch long, rolls. From the leveler the plates are discharged on a 132-foot combination table and turnover device. The turnover consists of two series of arms keyed to pivot shafts located between two parallel roller tables. In an idle position both sets of arms are below the roller-top levels of the two tables. When the plate is delivered from the leveler, the arms are moved toward each other in an arc, with the sending arms passing through a greater arc. This results in a transfer of the plate to the receiving arms as they approach a vertical position. The lowering of the arms transfers the plate to the adjacent parallel table which reverses travel direction toward the multiple shear.

Five 70-foot approach table sections serve as a bottom surface inspection and marking table. The first identification is made by chalking on the plate surface. Multiple lengths to be sheared are also marked on the plate with allowances for necessary tests. The multiple shear is a $\frac{3}{4}$ -inch by 100-inch, motor-driven, open-throat, downcut shear. A chain scrap conveyor is common to the multiple and the adjacent end shear and carries the scrap from both to an alligator scrap shear.

Continued table travel of 178 feet brings the multiple plates to a 64-foot cold transfer. This is a chain lift transfer with a 32-foot span from the center line of entry and delivery, tables. During the crosswise travel, physical identification of product and test pieces is completed by manual painting and stamping. Routing information for placement in the shipping area or direct car loading also is indicated on the plates.

Travel in the rolling direction is resumed on the 60-foot approach table to the rotary shear. The finish-shear lines are located in the shipping buildings which are adjacent and parallel to the mill building. Three magnetic manipulators traveling in a direction trans-

verse to that of the table are available for positioning the plates for the rotary-shear entry. An electromagnet, traveling with the table direction, holds the plate in a fixed lateral position. The double rotary shear has 15-inch diameter top and 60-inch diameter bottom knives. Cutting capacity ranges from $\frac{1}{8}$ -inch to $\frac{3}{4}$ -inch gage and 20 inches to 96 inches in width at a cutting speed of 79 to 158 feet per minute. Width settings may be made in $\frac{1}{16}$ -inch increments. Side trimmings are guided to rotary scrap cutters, cut into short lengths and dropped into a hold for removal by magnet.

Two 46-foot table sections that serve as delivery and entry tables for the rotary and the end shears respectively have powered side guards to guide the delivery from the rotary shear and facilitate square entry into the end shear. The end shear is a duplicate of the $\frac{3}{4}$ -inch by 100-inch multiple shear. A motor-operated lift plate gage with a travel speed of 50 feet per minute can be set from a 6-foot to a 60-foot distance from the shear knife. The 60-foot delivery table is followed by a 68-foot scale table and two sections of 64-foot push-off tables and side pilers. The side pushers travel in transverse ways across the tables to stack the plates in the pivoted side pilers, the table ends of which are lowered in conformance with the height of the plate stack.

Plates heavier than $\frac{3}{4}$ -inch and up to $1\frac{1}{4}$ -inch gage are laid out manually on the push-off tables and continue over a short approach table to a $1\frac{1}{4}$ -inch by 100-inch end shear. The shear is a conventional motor-driven downcut type followed by a short depressing table and a delivery table with an overall length of 26 feet. The plates then are moved manually over caster beds to a $1\frac{1}{4}$ -inch by 180-inch side shear located 58 feet from the end shear. The plates are turned 180 degrees on the caster beds and moved back to the side shear to trim the opposite edge. Side-trim scrap is moved manually to an alligator scrap shear for cutting into short lengths.

A circle shear is located beyond the side shear of the west shear line. The shear has a cutting capacity of $\frac{1}{8}$ -inch to $\frac{3}{4}$ -inch gage and 16-inch to 84-inch diameter at a cutting speed of 20 to 60 feet per minute. A transfer car at the extreme end of the building permits the transfer of circles and plates between the various buildings.

UNIVERSAL PLATE MILL

The 30-Inch Universal Plate Mill at South Works—The 30-inch universal plate mill at South Works, Figure 27-12, was erected in 1907 to produce rolled-edge plates ranging from $\frac{3}{16}$ inch to 6 inches in thickness and 6 inches to 30 inches in width. Slab sizes utilized to produce this range vary from $2\frac{1}{2}$ inches to $11\frac{1}{2}$ inches in thickness, 5 inches to 31 inches in width and 55 inches to 120 inches in length.

The conditioning and slab-storage yard serving this mill is under an open crane runway of 83-foot span and 280-foot length, serviced by two single-trolley, single-hoist, 10-ton cranes. The yard consists of four sections, for slab receipt, conditioning by hand scarfing, stock storage, and current charge storage. Charge slabs are piled in upright racks in the proper sequence. Forty-five hundred tons can be stored in the charge and stock racks.

Reheating Furnaces—Two single-zone, end-fired, hydraulic-pusher-charged, continuous reheating furnaces service the mill. The burners located above the discharge doors are designed for changing quickly to natural gas or oil for combustion. Neither gas nor air is preheated, inasmuch as neither checkers nor recupera-

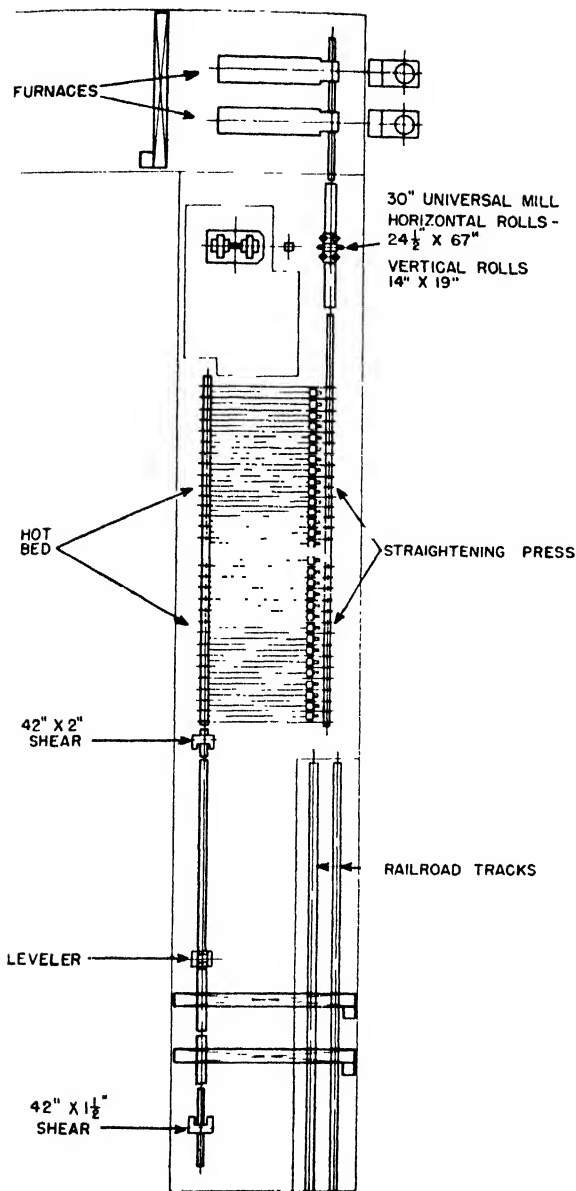


Fig. 27—12. Layout (not to scale) of the 30-inch universal plate mill at South Works.

tors are incorporated into the furnace design. Air for combustion is provided under pressure by a fan which draws directly from the atmosphere. Fuel-air ratio and furnace pressure controls are operative for natural-gas combustion. The furnaces are rated at 20 tons an hour per furnace for full hearth coverage and a cold charge. Heated slabs and billets slide down the dropout skids onto the mill approach table.

30-Inch Universal Plate Mill Stand—The single-stand two-high reversing mill, with vertical edgers front and rear, is located 68 feet from the center line of the near furnace. The chilled alloy-iron horizontal rolls are $24\frac{1}{2}$ inches in diameter and 67 inches in body length; the chilled-iron vertical rolls are $14\frac{1}{2}$ inches in diameter and 19 inches in body length. The top roll is balanced by counterweights, and is positioned for draft settings by a 50-horsepower motor driving both vertical screw-downs through worm reduction gearing. The maximum available horizontal roll opening is 10 inches. Initial roll

alignment is provided by placing liners as required under the fabric bearing chucks of the bottom-roll.

Additional roll-alignment adjustment required in rolling is obtained by the manual operation, during pass intervals, of a device which varies the distance between one screw and bearing chuck.

The vertical-roll assemblies at each side of the mill are positioned by a 30-horsepower motor. A drive shaft extends across both main housings and has pinions meshing with a spur-gear train on the outer vertical face of each housing. Bronze nuts, driven by the gearing, move upper and lower screws transversely, which in turn move the vertical-roll housings. A yoke connects the outer square ends of the screws to prevent their turning. It also has an adjustable rod attached with a nut to the yoke center that extends through an opening in the housing and is pinned to the vertical-roll yoke. The yoke and the rod serve as a pull-back when the roll spacing is widened. Available roll opening varies from a 5-inch minimum to a $31\frac{1}{2}$ -inch maximum. The weight of the vertical-roll assemblies is supported on an inverted V slide at the base of the main housings.

The mill is driven by a 4000-horsepower, 150-r.p.m. reversing motor, the first reversing-mill motor to be built and placed into successful operation. Power is transmitted through a leading spindle to the bottom pinion of a five-pinion, three-high, cast-herringbone pinion stand. The bottom and the second pinions, of sixteen teeth each, are connected directly through spindles to drive the top and the bottom horizontal rolls. The top pinion of fifteen teeth meshes with the middle pinion below it and with one fifteen-tooth pinion at each side of it with a common center-line elevation. All five pinions are journaled in babbitt bearings in the pinion housing. The side pinions drive the vertical-roll square shafts on which the square bores of the bevel driving gears for the vertical rolls slide for roll positioning, and which transmit power to the mating gears keyed to the vertical rolls.

Rolling—Slabs intended for plates in the 6-inch to 8-inch width range are provided 1 inch narrower than the finished plate width. A slab 6 inches thick and 8 inches wide rolled down to 1-inch gage will finish $9\frac{3}{4}$ inches wide on this mill if it is allowed to spread freely. Therefore, on the basis of slab provision for a 1-inch-gage, 8-inch wide plate, a 1-inch spread is allowed; and the edges actually are worked down $\frac{3}{4}$ inch. Slabs provided for over 8-inch to 12-inch widths are $\frac{1}{2}$ inch narrower than the finished size; slab and finished widths are equivalent in the over 12-inch to 17-inch range; and slabs $\frac{1}{2}$ inch over finished size are provided for the over 17-inch to 30-inch range. Edging drafts are taken alternately with the vertical rolls on the delivery side of the horizontal-roll drafting passes. As the horizontal-roll drafts are variable, vertical and horizontal-roll diameters must be kept matched so that the peripheral speed of the vertical rolls exceeds that of the horizontal rolls. Main-roll changes average three per week for a normal schedule and require 55 minutes to complete. Vertical rolls are changed at three-week intervals and require $3\frac{1}{2}$ hours to change both sets. Spray nozzles operate at 650 pounds per square inch pressure for descaling on both sides of the mill.

Hot Bed—The finished plate is delivered by a 69-foot delivery table to a two-section lift chain hot bed, each section being 77 feet long by 59 feet wide, with transfer chains spaced on 5-foot 6-inch centers. When delivered to the hot bed, the plate is moved off the hot-bed entry table by the transfer chains and positioned against fixed raised castings or anvils which are

parts of a straightener-press used to remove delivered camber, and to prevent camber resulting from the initial cooling stages. The straightener-press contains 30 individual blocks, each of which consists of a 3½-inch diameter worm screw, a bronze nut, a head casting attached to the worm screw, and the fixed table casting that serves as the anvil. The blocks are on 5-foot, 6-inch centers and are located between the transfer chains and adjacent to the hot-bed entry table. Two worm line shafts, each 5 inches in diameter and 89 feet, 7 inches long, drive the individual worm screws. Each line shaft, in turn, is driven by a 50-horsepower motor through reduction gearing. The castings which serve as anvils are integral parts of troughs through which cooling water circulates. When the worm screws are operated, the movable heads push against the edge of the plate and press it straight against the anvil castings.

Plates are delivered to the two hot-bed sections alternately. When the plates are released by the straightener-press, they again are picked up by the lift transfer chains and moved across the bed in plate-width space increments. In the course of their traverse progress the plates are identified by painting and stamping and laid out for the lengths that are ordered.

Required tests also are laid out and identified. The final chain movement places the plates on the combination bed-delivery and shear-approach table.

Finishing—The end shear, located immediately adjacent to the hot bed, is a 42-inch by 2-inch guillotine downcut shear with a counterweighted upper-knife holder, and is operated by an 11-horsepower, 800-r.p.m. motor. A chain conveyor moves crop scrap to a chute which guides it into a scrap box for disposal. The sheared plates go over a table to a roller leveler which has 6 top and 5 bottom bending rolls, 8 inches in diameter and 44 inches in body length. Plates up to 2-inch thickness are pulled off the table line on the delivery side of the leveler. They are pulled off manually with the use of hooks into stacking cradles. Heavier plates are pushed manually on an idler table to a horizontal gag in which they are straightened and pulled off in a similar manner on the delivery side.

Auxiliary equipment includes flame-cutting equipment for heavy plates, hand-grinding tools and powered hack saws. Storage racks are provided for carload-accumulation storage in the combination shearing and shipping building. Additional storage and shipping facilities are available in an adjacent building.

Chapter 28

RAILROAD RAILS AND JOINT BARS

SECTION 1

ROLLING OF RAILROAD RAILS

Dating from the invention of the steam locomotive, the railroad rail represents one of the first sections to be rolled. The railroad rail is a most vital part in railroad operations and represents a difficult section for the roll designer and roller. With the advancement in speed of travel and weight of loads carried, more and more has been required of the rail, until today no product is subject to more severe service conditions. Exposed to the weather at all times, it is subjected, under constantly varying conditions, to high compression and bending stresses, impact, vibration, friction and wear.

The railroad rail should be designed to have the greatest possible transverse strength, to provide an abundance of metal for wear, to present a wide base for fastening to the cross tie, and, for the sake of economy, it should be of the lightest section possible. The American Tee Rail best meets all of these requirements and represents the accepted design.

Historical Development of Rail Sections—The history of rail development is indicated in the sketches of Figure 28—1. The first running surfaces for the early railroad rolling stock consisted of strap rails comprised of cast-iron plates approximately 4 inches by 1¼ inches by 5 feet long, which were attached to a wooden base. The first strap rails were used around 1767. Various types of cast and malleable iron rails were used until about 1820 when the first iron rails were rolled. These were supported by cast-iron holders, called chairs, attached to stone supports. In an effort to eliminate use of the expensive chair required for this type of iron rail, a rail with a wide and relatively heavy flange on the bottom was rolled in 1831. The difficulty of rolling the flange led to the better balanced Locke rail of 1837, the bull head and U-shape rails of 1844, and the pear head rail of 1845. Then came the compound rail of 1856 (not shown in Figure 28—1) and another form in 1858, which was the U-shape of 1844 with the lower parts closed in and welded to form a web. As neither of these forms proved serviceable, a demand for more metal in the head for wear forced a final return in 1858–1868 to the tee shape with wide thin flange. Subsequent to 1858–1868 the quality of the steel, the design of the rails and rolls, and the equipment of the mills have improved continuously. Present standard railroad rails are being rolled in various sizes ranging in weight from 65 lb. up to and including 155 lb. per yard. Rails 60 lb. per yard and lighter are classed as light rails. Representative chemical compositions for rails are shown in the American Railway Engineering Association (A.R.E.A.) tabulation given in Table 28—I.

Mills for Rolling Rails—There does not seem to be much accurate information available about the first mills which rolled rails. It is probable that existing mills designed to roll bars were utilized with such alterations

and additions as were necessary. Credit for rolling the first steel rail in 1857 is given to the Dowlais Plant, Wales, while the credit for rolling the first steel rail in this country is given to Captain Ward's North Chicago Rolling Mills, where the first Bessemer steel rails were rolled experimentally in 1865 from blooms made of hammered ingots produced at Wyandotte, Michigan.

Rails were originally rolled on the pullover mill, and later on the reversing mill. In this country rails have been rolled for many years on the three-high mill, which was usually made up of a single train of three stands driven by one engine. Sixty-nine mills were reported to be rolling rails of various weights in 1874. During this great railroad-expansion period, rails were in such demand that even this large number of mills could not supply the demand. These mills were scattered around the country with one as far west as Laramie, Wyoming. The tonnages produced on the old mills were small in comparison to present-day tonnages. Since about 1900, rail mills have been forced to undergo many changes because of the ever increasing requirements of the railroads with respect to quality of the steel, size of the rail section, length of rail, freedom from internal and surface defects, exactness as to dimensions, and a variable demand as to quantity. At the end of 1954, there were nine mills in the United States and Canada which were producing railroad rails. Additional rail mills on the North and South American continents are at Monterey, Mexico and Volta Redonda, Brazil. These mills are far from being standardized in layout. Some have a large number of stands, others only a few, some are two-high throughout, while others are three-high. No one rail mill may be cited as an example typical of all, but a brief general description of a mill which requires reheating of the blooms and another mill which rolls direct will illustrate operating conditions of representative mills.

In the first example, that of a large rail mill in which the blooms are reheated, the ingot first passes through a 48-inch, two-stand, two-high, tandem bloomer with two passes in each stand. After the ingot has passed through once, it is returned to the first stand by a run-

Table 28—I. Representative Chemical Compositions for Rails

Constituents	Nominal Weight in Lb. per Yard		
	70/90	91/120	121 and Over
Carbon	0.64–0.77%	0.67–0.80%	0.69–0.82%
Manganese ...	0.60–0.90	0.70–1.00	0.70–1.00
Phosphorus (maximum) .	0.04	0.04	0.04
Silicon	0.10–0.23	0.10–0.23	0.10–0.23

THE MAKING, SHAPING AND TREATING OF STEEL

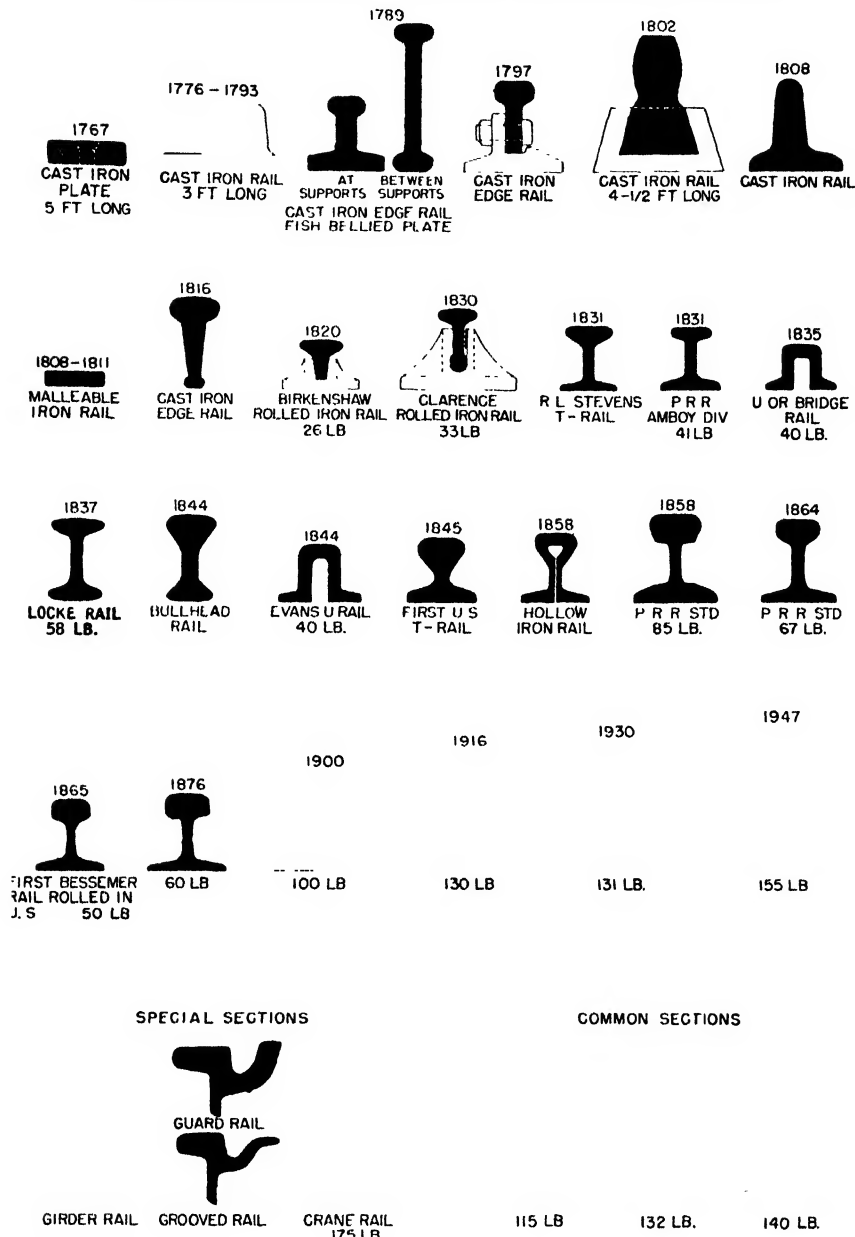


FIG. 28-1. Sketches of cross-sections of rails from the earliest periods of railroading until the present, showing the evolution of the modern railroad-rail design. Certain special rail sections are shown at the lower left.

around turn table for the second pass. After passing through the stands twice, the piece goes on to a three-high 40-inch blooming mill, where it is given seven passes. After being sheared into bloom lengths and reheated, the blooms go on to a four-stand rail finishing mill, the first part of which consists of two engine-driven three-high stands. The bar is given five passes in the first set of rolls and four passes in the second set and then moves straight forward to a three-high stand driven by its own engine, where it is given three additional passes. From this stand, the rail moves on to the two-high finishing mill for the final pass. There are thirteen passes from the bloom to the finished rail.

The second example, a mill in which rails are rolled directly from ingots, is represented by the largest rail

mill in this country, the layout of which is somewhat similar to the cross-country type. This mill starts with four tandem stands of 40-inch, two-high rolls. In these stands, the ingot is given one pass per stand, and is turned after each pass. The passes are of diamond, diamond-square, and box-pass design. After the initial passes, there is a 40-inch three-high mill with five box passes, the final pass being slightly shaped to give the hitherto rectangular bloom a form more suitable for subsequent rolling. Following this three-high blooming stand is the bloom shear and then the cross-country arrangement of seven stands of rolls arranged in two groups of three stands each and one separate stand. The leader stand has a vertical roll working on the head of the rail; the finishing stand one for the rail base.

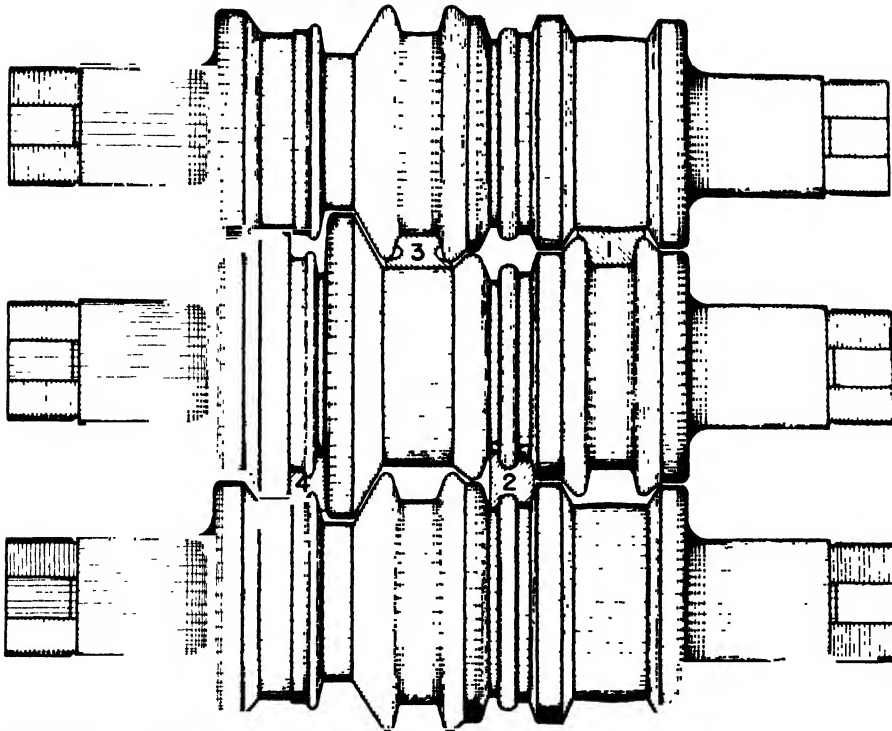


FIG. 28—2. Sketch of rolls used in second roughing stand of a rail mill rolling by the tongue-and-groove, flat, or slab-and-edging method, showing shape produced in each of the four passes.

Rails are formed by two general methods of rolling, known as the tongue-and-groove, flat or slab-and-edging, and the diagonal or angular method. Several of the rail mills combine these two methods, some of the passes being designed to form by the first method and the remaining passes to roll the section by the second

method. In the tongue-and-groove, flat or slab-and-edging method, illustrated by the second roughing stand in Figure 28—2, the axis of symmetry of the rail coincides with the pitch line and is parallel to the train line of the rolls.

The diagonal or angular method of rolling is repre-

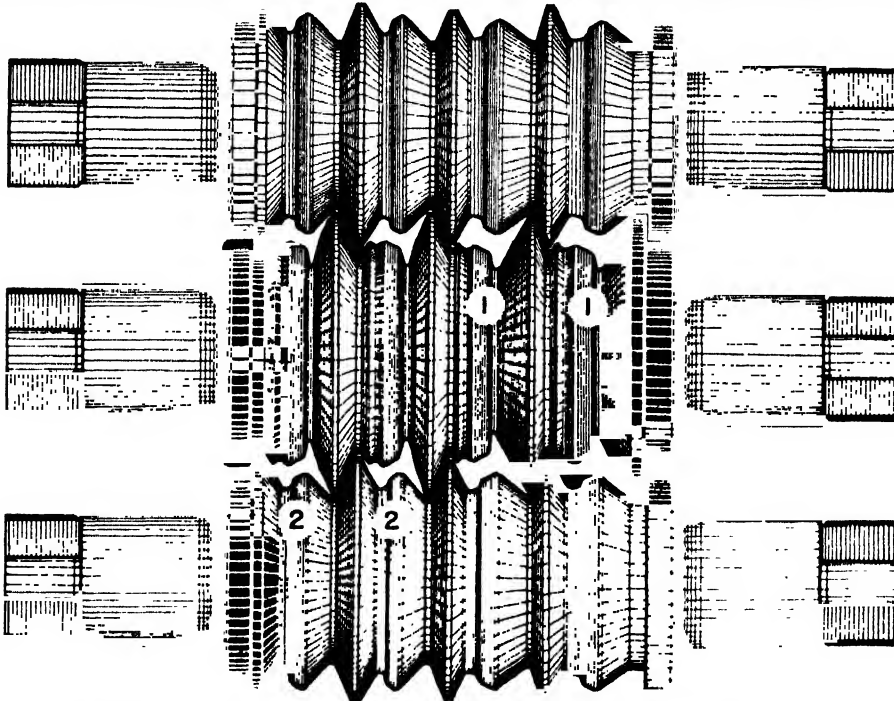


FIG. 28—3. Sketch of rolls used in the roughing stand of a rail mill rolling by the diagonal or angular method.

sented by the roughing stand shown in Figure 28—3. It differs from the slabbing method in that the shaping of the rail is begun with the first pass in the roughers and instead of first compressing the bloom to a smaller size and then forming the section partly through compression and partly by spreading, the process is one of compression from beginning to end.

DESIGNING THE ROLLS FOR RAILS

The first consideration in designing the rolls for rails is to produce a finished piece of the correct size and form. This objective can be accomplished only by directing and controlling the flowing, spreading and bending of the steel. The ease with which this forming is done depends on the plasticity of the metal, which in turn is affected by the temperature. With the speed of the rolls fixed, the temperature confined to a narrow range, and the grade of steel given, the only means of control remaining to the roll designer is the size and shape of the passes, and in part of these, at least, the size will be governed by the size of the bloom. In designing the passes, a good designer will endeavor to work the steel in such a manner that the quality of the product will be benefited, and no defects will be developed. The defects that should be carefully avoided are fins, laps, overfills, and underfills. Laps may result from fins or a collaring of the piece in the rolls; overfills, from worn rolls, poor or improper design; and underfills either from poor design or incorrect adjustment of the rolls.

Stages of Reduction—The formation of the rail from the bloom may be considered as taking place in three steps or stages. The first stage, called the **roughing**, is merely one of preparation; in it a large amount of work is done, but this work is expended mainly in reducing the size of the section and elongating the piece. The intermediate stage proceeds with the forming of the rail and involves a combination of **slabber**, **dummy**, **former**, **edger** and **leader** passes, dependent on the mill layout. The finishing pass completes the formation of the rail.

The Section—No original designing of section is done by the roll designer. The first requirement in the rolling of a new section is that the roll designer be supplied with a drawing or print of the section, which must be accompanied with all the dimensions, preferably indicated on the print. The weight of rail desired or expected should also be given. Here the matter of dimensions is

of extreme importance, for the designing of the templets cannot be started until each and every dimension required is given. These dimensions not only include linear measurements, such as height of rail, width and thickness of parts, but also the radius of all curves, and amount of slope on inclined surfaces expressed in degrees or percentages.

Roll Preparation—With all the necessary information available, the first step taken by the roll designer is to prepare a drawing for the cold templet. This drawing is constructed on the axis of symmetry of the rail, which is the vertical line drawn through the center of the head, of the web, and of the flange. On this line the section of rail is symmetrically constructed to the dimensions given on the drawing, all the dimensions being made with extreme care and accuracy. Upon completion of this very accurate drawing, the area of the section is measured with a planimeter in order to check the weight of the section. Any discrepancies between dimensions or weight are clarified with the customer after which the cold templet is prepared from either brass or steel.

The next step, which is really the first step in designing the roll passes, is the making of the hot templet. This templet is similar to the cold templet, but larger in size, as it represents the section of the rail at the finishing temperature of rolling.

From the hot templet the various passes are designed successively as the experience and judgment of the designer dictates. Roll designers may prepare the various pass templets by several methods. Usually the pass templets from a similar section are used as a guide, since many of the sections currently being rolled have been perfected through many modifications from initial designs for the passes. Frequently, the designer constructs each pass outline in a drawing showing the different passes superimposed upon each other. As a preliminary step toward designing passes back to the bloom from the finished templet, a table is usually prepared. This table will consist of various passes, cross-sectional area of the head, web, and base, per cent reduction, and spread in inches. The designer must constantly keep in mind the danger of forming fins. In order to avoid these defects, the designer may arrange the passes so that each side of the piece alternately enters an open and a closed side of the groove when rolling by the tongue-and-groove, flat or slab-and-edging method. Even with

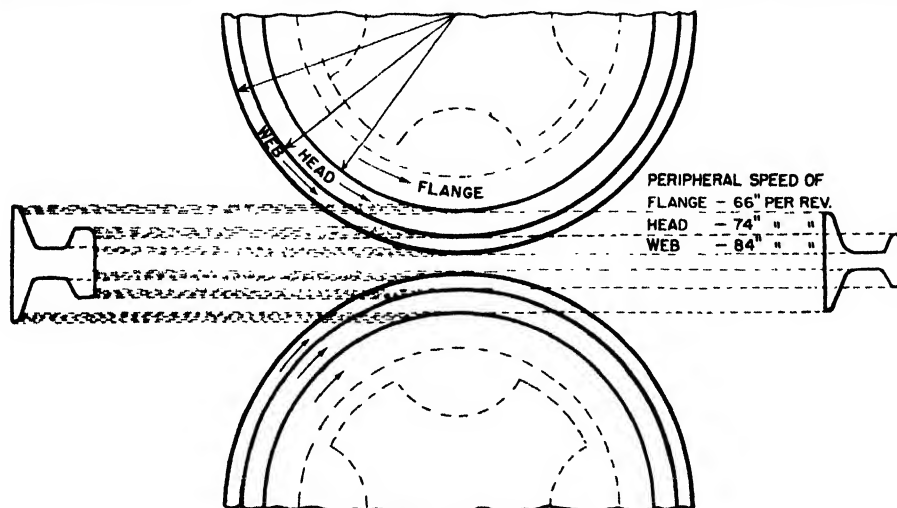


FIG. 28—4. Differences in peripheral speeds at various points in the roll pass contacting, respectively, the flange, head and web of a rail during rolling.

this arrangement, fins would still be formed if the passes were not properly designed. To avoid the possibility of forming fins, two modifications of design may be used. In the leader pass, the corner of the head, which is to come opposite the openings between the rolls in the finishing pass, is well rounded off, so that the spread or flow of the metal will be taken up in filling out this rounded corner and none will remain to be forced into the clearance. For the same reason, that half of the flange on the same side of the rail is left much shorter. This provision is made in many of the passes. Great care is necessary in distributing the reduction of each part to prevent the metal flowing away from parts where it is needed. For example, if a too great reduction in the web takes place in one pass, it will produce a flow of metal away from the head, causing the latter to be underfilled. The cause for much of the trouble of this kind lies in the different diameters of the pass, which cause a different roll-surface speed for head, web, and flange, and hence different rates of elongation. If the elongation produced through compression is at the point of less speed, the section will be imperfectly formed, or cracks will result. The accompanying illustration (Figure 28—4) will help in understanding this point.

The roll designer strives to keep the roughing passes of such shapes and sizes that the same set may be used for a large number of different rail sections. The pass contours for producing the 132-pound RE rail are illustrated in Figure 28—5.

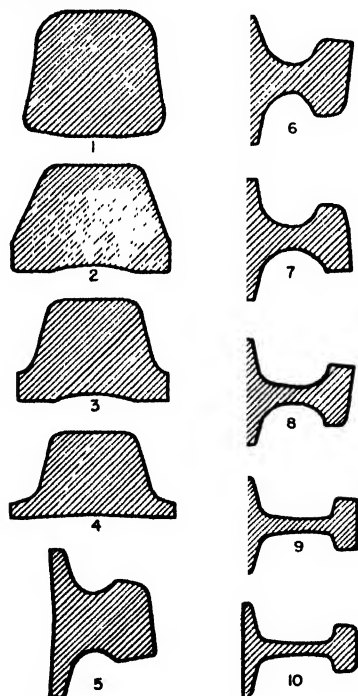


FIG. 28—5. Roll-pass contours for producing a 132-pound RE rail.

Upon completion of the pass templets, including both male and female for the cold templet, they are sent to the tool shop where they are used as patterns in making a set of tools for turning the rolls for the section. As many as 24 different tools may be required for the last six passes of each size rail. After shaping these tools a little over-size, they are heat treated and then redressed to exact size before they are used to turn the rolls. When ready, templets and tools go to the roll shop, where the work of turning the rolls is done. Rail-mill rolls may be

sand iron, alloy iron, or alloy steel rolls. Frequently more than one grade of rolls will be used on the same mill.

ROLLING PRACTICE

After the rolls have been properly turned, they are placed in the housings in their proper position and carefully lined up. A trial rolling with one bloom will then be made with the roller checking the piece closely to see that it goes through the mill in a satisfactory manner. The trial rail thus produced is gaged by templets which permit checking of base over-all, head width and contour, flange length and thickness, head radius and fish. (Fishing is a mill term for the dimensions of the theoretical contour of the under side of the head, the side of the web, and the upper side of the base.) By comparison with a master or cold templet the height and web thickness can be checked. If this section is found to be correct, the mill is then ready to begin the rolling. At frequent intervals, the rails are gaged and examined for defects such as collar marks, underfills, roll marks, overfills, guide marks, cracks, seams or pickups. One of the things that cannot be avoided in the rolling is the wear of the rolls which is aggravated by the slippage associated with the irregular section. The grooves tend to wear down rapidly, which will produce a "loose fish." "Rocking base" rails may result from worn rolls.

Standard specifications for rails require that the cold templet shall conform to the specified section as shown in detail on the drawing of the purchaser and shall at all times be maintained perfect. The rail section shall conform as accurately as possible to the templet or drawings furnished by the purchaser. A variation of $\frac{1}{64}$ -inch less or $\frac{1}{32}$ -inch greater than the specified height is permitted. A variation of $\frac{1}{16}$ -inch in the width of either flange is permitted, but the variation in total width of base shall not exceed $\frac{1}{16}$ -inch. No variation is permitted in dimensions affecting the fit of the joint bars, except that the fishing templet approved by the purchaser may stand out not to exceed $\frac{1}{16}$ -inch laterally.

Girder Rails—Among the special types of rails rolled are the girder rails of the plain, guard and grooved type, used principally by electric surface railways. These rails represent some of the most difficult sections to roll and are usually produced on structural mills. Demand for these sections has diminished to the point that the rolling of many sections has been discontinued, with replacements being made with standard T-rails. Contours of typical girder, guard, and grooved rails are shown in Figure 28—1.

Crane Rails—Crane rails, in three sections of 104–105 pound, 135 pound, and 174–176 pound, are currently being produced. These rails in general have a heavier head and web than the railroad rail in order to withstand the heavy loads imposed in service. These rails ordinarily are rolled on rail mills and are made by the same manufacturing practices as railroad rails.

Light Rails—Light rails are rolled on several types of mills—bar mills, light structural, and, in some cases, mills built and operated exclusively for light rails. Light rails are produced from heavy rails or from billets. In the rerolling of light rails from standard rails, either passes of the edge and flat design or of the diagonal design may be used. The rolling of light rails from billets is accomplished in much the same manner as the rolling of standard rails. An average of nine passes is generally used, although in a few extreme cases as few as five passes have been used.

FINISHING OPERATIONS FOR STANDARD RAILS

Cutting and Cambering—Railroad rails are ordinarily cut to produce the standard 39-foot rail. Lengths of 30

and 33 feet, which represent the old standard lengths, are produced in small tonnages, and long rails of lengths from 45 feet to 78 feet are produced in limited quantities. Finishing equipment is designed for standard lengths and longer lengths are handled with inconvenience and at additional cost. The rails are cut to desired lengths by hot saws operated singly or in gang, mounted over the runout table beyond the finishing stand. In cutting the hot rails, proper allowance must be made for linear thermal contraction, which, for a 39-foot rail, is approximately 8 inches. The exact amount of the contraction depends upon the temperature at which the rail is sawed. It is necessary to set the saws very accurately as the first rail from an ingot as well as the last rail from the same ingot must be within the length tolerance of plus or minus $\frac{3}{16}$ -inch. Vertical alignment of the saws must also be accurate and the saws must be maintained in sharp condition in order to meet a $\frac{1}{32}$ -inch maximum off-square tolerance. Drop-test and the nick-and-break samples are obtained from the crop ends cut at the hot saws. After sawing, the rails pass under a stamping machine which marks the heat number, ingot number and position of the rail in the ingot. The latter is designated by letters beginning with A at the top of the ingot. Between the stamping machine and cooling beds is a cambering machine which consists of a set of horizontal rolls with a vertical roll on each side, all in one housing and set to bend the rail slightly so as to make the top surface of the rail convex from end to end. This bending is done to compensate for the camber produced in a straight rail as it cools on the cooling bed, this camber being caused by the different rates at which the head and the base cool. A scale located near the end of the delivery table is used for checking the weight of the rails as often as desired, before they are advanced to the cooling beds. Rail specifications state that a variation of 0.5 per cent from the calculated weight of section as applied to the entire order is permitted.

Marking and Branding—One of the mandatory requirements for standard railroad rails is that each rail shall be legibly marked for complete identification. So far as practicable, this branding is done by numbers, symbols and letters cut in the bottom finishing roll to give raised characters on one side of the web of the rail. Markings that cannot be thus rolled into the web are marked intaglio on the opposite side of the web by a hot stamping machine following the hot saws. The nature of the information required and the methods of marking are as follows:

Method of Marking	Kind of Mark	Nature of Information
Engraving on bottom finishing roll	{ Raised characters }	Weight or section number
		Type CC (for control cooled) Manufacturer's brand Year rolled Month rolled
Hot stamping	{ Intaglio }	{ Heat number Rail letter Ingot number }

Controlled Cooling—Controlled cooling of rails on a production basis was begun between 1935 and 1937. From the rather crude start with various types of containers and covers, refinements have been made and specifications prepared so that a uniform controlled-cooling practice now is obtained.

Present specifications for rails intended for railroad service require controlled cooling of the rails within the

temperature range between 725° and 300° F. This is a process especially developed to prevent the formation of shatter cracks, also called internal thermal ruptures, flakes or internal thermal cracks. In mill operations, the rails are allowed to cool to under 1000° F and then are placed in either stationary or movable insulated containers. The rails must be placed in the container within the temperature range between 1000° and 725° F, which is checked by a radiation pyrometer. One or more thermocouples of the chromel-alumel type are placed between tiers of rails in the slow-cool container in order to obtain temperature readings during cooling. The containers are insulated to meet the specified cooling cycle of not dropping below 300° F in 7 hours for rails 100 lb. per yard in weight or heavier, from the time that the bottom tier is placed in the container, and 5 hours for rails of less than 100 lb. per yard in weight. Rails must remain in the cooling container for a minimum of 10 hours. Complete records of the cooling cycle to 300° F are maintained on each container.

Testing of Standard Rails—Testing of standard railroad rails includes both chemical and mechanical tests. Two ladle test samples, representing the composition of one of the first three and one of the last three ingots applied, are obtained. The drop test or the nick-and-break test are relied upon to supply the only mechanical-test information required. Crop ends 4 to 6 feet long are cut from the top end of the "A" rail from the second, middle, and last full ingot of each heat. ("A" rails are those rolled from the upper portion of the ingot, "B," "C," "D," etc., rails being rolled from steel in successively lower parts of the ingot.) These specimens are placed upon supports and subjected to the impact of a tup weighing 2000 lb., falling from a height of 17 to 22 feet, depending upon the weight of the section. For rails 106 lb. per yard or less in weight, the supports are placed 3 feet apart and for rails over 106 lb. per yard this distance is increased to 4 feet, with the tup striking the rail midway between the supports. If a specimen breaks, all the "A" rails of the heat are rejected and the "B" rails must be similarly tested. Failure of a "B" rail specimen causes all "B" rails to be rejected in addition to the "A" rails and similar tests must be made in "C" rails. Failure of a "C" rail specimen is cause for rejection of the whole heat. One specimen from a heat is given a number of blows to determine the ductility.

In progressive nick-and-break testing, if an interior defect shows on the fracture, the top end of the top rail is nicked and broken back. If a fracture free from interior defects is reached at a point permitting the rail to be finished to an acceptable length, the rail and the following rails of the ingot are accepted. If not, the rail is rejected and a test piece cut from its bottom end to represent the second rail of the ingot. The second and succeeding rails are tested in the same manner when so required.

Finishing Operations—Finishing operations for standard rails include preliminary inspection upon unloading from slow-cool cars or containers, removal of saw burrs, straightening, drilling, grinding of ends, beveling of heads when specified, inspection, classification, and painting. The purpose of the preliminary inspection is to check identification and lengths and to locate harmful surface defects and thus save the expense of finishing a rail that would be rejected. The removal of burrs made by the saws on the ends of the rails usually occurs prior to the straightening, although in some instances burr removal occurs after the straightening operation. The burrs are cut off with chisels and subsequently smoothed with a file or a grinder.

The rails are straightened on gag presses, each of

which is provided with two bottom supports located in the table proper on which the rail rests, and a top block with die attachment which moves up and down with a fixed stroke between the supports. The stroke of the block is of such length as not to touch the rail by about two inches at its lowest point. On each side of the press at spaced intervals are located two or three stands, each with a roll, to allow bringing the rail to the press and manipulating the rail back and forth while it is being straightened. The die has a double face, each side of which slopes to the center line. The gag, a rectangular block of steel slightly beveled to fit the die, is inserted between the rail and the die. The die form, in combination with the different dimensions of the gag, makes it possible to control the amount of bend the rail receives and to adjust the press to the several rail dimensions. To straighten a rail, one man called a gagger is stationed in front of the machine and a second referred to as a straightener at the end. By sighting along the rail, the straightener at the end locates the areas of rail which require straightening and brings them under the blocks while the gagger before the machine, acting under directions from the straightener at the end, inserts the gag in such a way that the stroke of the machine will bend the rail enough to straighten it, which requires that the rail be bent beyond its elastic limit in order to give a permanent set.

The rails are next moved to the drilling machines, which are arranged in pairs and so spaced that when one machine has completed the drilling on one end of the rail, it is moved under the other machine which drills the holes in the opposite end. These machines are provided with three drilling spindles, the middle one of which is fixed, and the outside two adjustable, and may be made to drill from one to three holes at one time. After drilling, the rails are moved to inspection beds where they are walked, or inspected, ends checked, ends beveled when specified, classified, painted, and separated for loading. The finished rail inspection is made by mill and purchasers' inspectors to satisfy themselves that the rails are according to specifications. The rails are inspected twice, one time with the base up and the second time with the head up. The location of defects or rails not satisfactorily straightened are marked with chalk. If these defects are located near the end, that portion of the rail may be sawed off and the rail still applied on the order as a short of first grade; shorts are accepted in limited quantities. If the defects are many or are near the center, the rail is either classed as a number two, or scrapped. Number two rails may contain slight imperfections which do not make them unfit for service. Number one rails must be free of injurious defects and flaws of all kinds. Rails which are not straight are transferred to a restraighening press for restraighening whereas rails required to be sawed are transferred to a recut unit which is equipped with slow-speed saws. Off-square ends are corrected either by use

of end-milling machines or hand grinders. Rails are classified into the following groups and each group is loaded in separate cars.

Classification	Painting on Rail Ends*
No. 2 rails.....	White
"X-Rayls"	Brown
"A" rails	Yellow
No. 1 rails of less than standard lengths....	Green
No. 1 rails of a heat whose carbon content is in the upper five points of the carbon range specified	Blue
No. 1 rails of a heat whose carbon content is lower than the upper five points of the carbon range specified.....	None

* Individual rails shall be painted only one color, according to the order of precedence listed above.

End Hardening—End hardening is a heat treatment given to the top end surface of a rail head which makes it more wear-resistant. The heating of the rail end is accomplished by either gas hoods, gas burners, or induction heating coils. Compressed air is used for the accelerated cooling. Usually the hardened pattern, including the hardened and transition zones, is about $\frac{3}{4}$ -inch in depth at the rail end, tapering to zero depth at a distance approximately $3\frac{3}{4}$ inch back from the end of the rail.

Some type of production-control testing is generally performed on end-hardened rails, usually the standard Brinell test, using a 10 mm. ball, made on representative samples.

Finishing and Inspection of Crane Rails—Crane rails are subjected to the same testing and processing as standard railroad rails. In the lighter sections, crane rails have been heat treated by quenching in oil and tempering with very satisfactory service results where heavy loads are involved. Crane rails in the control-cooled and the heat-treated conditions, with welded joints, have been placed in service in numerous locations.

Finishing and Inspection of Girder and Light Rails—Girder rails are not subjected to slow cooling, and straightening is usually done on a machine straightener, supplemented by gag straightening. Standard specifications cover girder rails of the plain, grooved and guard types, and finishing operations are comparable to standard rails. The impression test to check hardness is used rather than drop testing.

The finishing and inspection of light rails are different from the same operations for heavy rails. Because of the relatively small tonnage of most orders, light rails cannot be handled as separate heats and many light rails are rolled from billets of lower carbon content than heavy rails. Light rails are supplied in a variety of lengths and are usually straightened on a machine straightener. Bolt holes may be punched or drilled.

SECTION 2

THE ROLLING OF RAIL-JOINT BARS

Types of Rail Joints—Paralleling the development of rails to present-day standards has been the development in rail-joint bars and new techniques, such as welding for joining rails. Through the years, rail-joint bars have been known by various names, such as splice bars, angle bars, and fish plates. A great number of rail-joint bar designs were in use around 1925, such as (1) the Duquesne rail joint; (2) the 100 per cent rail joint; (3) the Weber rail joint; (4) the Hatfield rail joint; (5) the

Wolhaupter rail joint; (6) the Barschall rail joint; (7) the Bonzano rail joint; (8) the Abbott rail joint; (9) the Williams reinforced rail joint; and (10) the reinforced angle bar. Only a few of these rail joint bars are in use at the present time. In recent years there has been a trend toward a standard design of joint bar.

Present Rail-Joint Bars—Even with one type of joint bar, there must be many different designs, for the joint bar must fit accurately between the head and the base

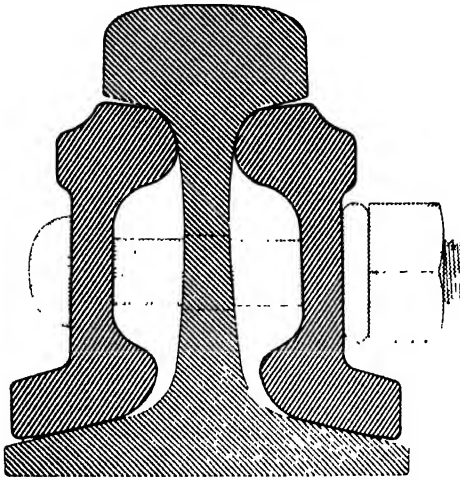


FIG. 28-6. Cross-sectional diagram of the head-free type of conventional rail-joint bar.

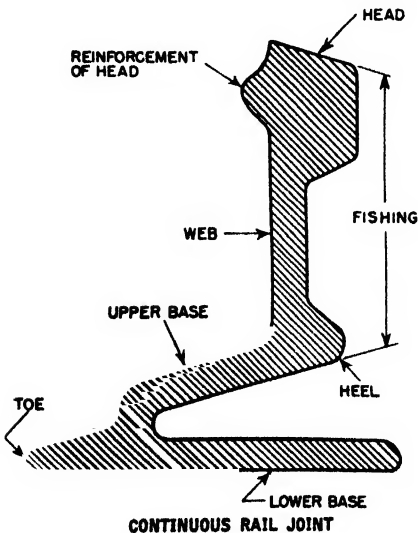
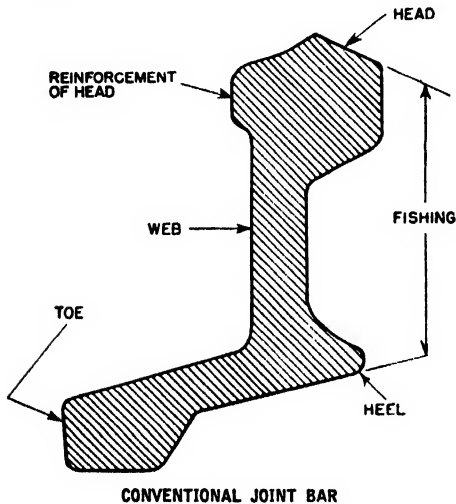


FIG. 28-7. Diagrammatic cross-sections of two types of rail-joint bars, with nomenclature of principal components.

of the rail, and each change in these dimensions of the rail requires a change in the joint bar. The most popular rail-joint bar type at the present time is the short-toe joint bar of either the head-free or head-contact design. A sketch of the conventional rail-joint bar of the head-free type (with short toe) is shown in Figure 28-6. The head contact type of conventional rail-joint bar is identical with the sketch shown in Figure 28-6, except there is contact of the under surface of the head of the rail with the head of the joint bar. The head-free design is considered to have some advantage with respect to a lower rate of bolt-tension loss and lower resistance to expansion movement of rail ends. The principal advantage of the head-free design, however, is the fact that its use permits the desired thickening of the upper web and lengthening of the fillet radius. Nomenclature for the present conventional rail joint (with long toe) and the older type of continuous rail joint is shown in Figure 28-7. The continuous joint bar shown is still in limited use. This section is rolled with the lower base flared, the flare being pressed into a size to fit the rail base when the bar is punched. Other types of joint bars, with or without reinforcements in the web, are in very limited use at the present time.

Problems in Rolling Rail-Joint Bars—Passes for rolling three typical joint bars are shown in Figure 28-8. Joint bars are usually rolled from billets or blooms within the carbon content range of 0.30 to 0.60 per cent, although a lower carbon range has application for the cold-worked joint bar. The conventional joint bar and the joint bar with long toe are subject to all of the drawbacks of the rail section and to many others in addition, because of their irregular section and lack of symmetry. In the conventional joint bar, the angles at which the section is rolled are limited by possibility of undercuts, and the shape of the passes in which the piece is necessarily reduced are favorable to the formation of laps and seams. If the joint has a depending flange, or long toe, these difficulties are multiplied, while the excessively protruding parts of a joint, such as in the continuous section (Figures 28-7 and 28-8), often prevent the piece from entering the pass properly, by striking the rolls first or being a trifle colder than the rest of the piece. For similar reasons, it is difficult to make guides that will properly handle such sections, and they are prone to become cobbled or caught in the rolls of the

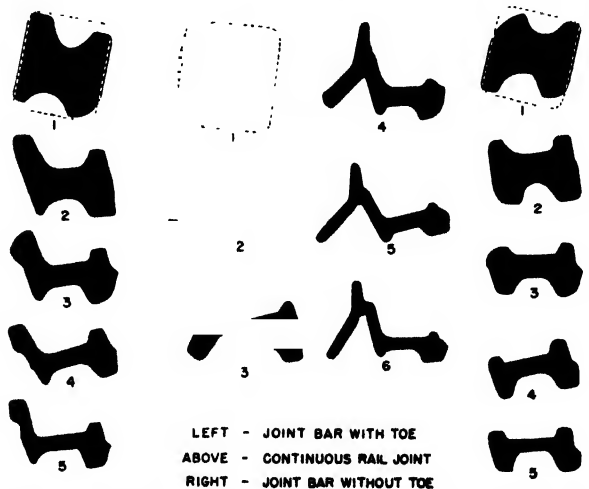


FIG. 28-8. Successive roll passes for the production of three typical rail-joint bars: (left) joint bar with toe; (center) continuous rail joint, and (right) joint bar without toe.

tables. The rolled section of joint bar is cut into convenient long lengths for handling and sent to the joint bar shop for processing into finished joint bars.

FINISHING JOINT BARS

In general, rail-joint bars may be finished in one of three ways: First, all the operations of shearing to length, straightening, punching and slotting may be performed upon the cold pieces without heating in any way; the finished pieces then are referred to as **cold-worked joint bars**. Second, the bars may be heated, after shearing to length, and the work of punching and slotting be done while they are hot, after which they are allowed to cool in air. In this case they are called **hot-worked joint bars**. Third, instead of cooling the bars in the air after hot working they may be cooled by immersing them in oil, when they are designated as **hot-worked and oil-quenched bars**. It will be observed that in this latter method, as in the other two methods, the bars are sheared to length cold, hot shearing being impractical, and hot or cold sawing too expensive. The hot-worked and oil-quenched joint bars possess such superior mechanical properties that practically all joint bars used on trunk-line railroads are finished by this process.

Cold-Worked Joint Bars—The bars rolled in long lengths are first sheared to length, punched and slotted and then inspected for straightness and flaws. The dies of the shears are, in all cases, made to conform to the shape of the bars, so that a clean cut is made without in any way deforming the bar. For a similar reason, the bottom blocks, or dies, of the punching and slotting machine support the web while the punches descend from above, pushing the material through conforming openings in the blocks. All dies are made of the highest grade of special tool steels and are kept in the best possible condition. In this way, the hole is made as smooth as it is possible to make it by punching. Bars of low carbon content are not adversely affected by the cold working but cold punching can injure bars of higher carbon content. One of the direct results of cold working high-carbon steel by cold punching is to increase the hardness of the metal about the hole; the worst effect, which applies only to bars of high carbon content, is the formation of very small cracks which extend into the metal along lines perpendicular to the surface of the hole. The difficulties in punching, as well as the deleterious effects of cold working, increase as the carbon content of the steel increases. As a rule, cold punching is applied only to the smaller joint bars with carbon content under 0.25 per cent.

Hot-Worked Joint Bars—The order of procedure for the manufacture of hot-worked joint bars is: shearing to length, heating, punching, slotting, straightening, cooling and inspecting. The "continuous" bar requires the additional operation of bending the base, which is usually accomplished on a combination folding and straightening machine.

The furnaces employed for hot working are of the continuous type, operating temperatures of modern installations being automatically controlled. After being sheared to length, the cold bars are laid upon water-cooled skid pipes and pushed into the furnace from the rear by electrically-operated dogs. The length of the furnace and rate of charging is such that about two hours are consumed in pushing each bar through the furnace. This time is sufficient to bring the bar to a working temperature of about 1500 to 1550° F (810 to 840° C) for 0.40 to 0.45 per cent carbon joint bars. The skids end near the front of the furnace, and the bars descend to a hearth or grate, whence they are removed

with tongs or hooks through the doors of the furnace.

During hot punching of a bar, in order to avoid spreading of the metal and consequent distortion of the bar, it is necessary to employ a confining die; that is, the cutting die must be enclosed in a die block or frame, the upper surface of which, together with the die itself, conforms in shape to the inside surface of the bar. The straightening machines are presses provided with a set of dies for each size of each section. One die conforms to the size and shape of one side of the section and the second die to the other side, and both are set in the press so that at the end of the stroke the space between the dies is of the same shape as the bar and just equal to it in thickness.

The harmful effects of cold working are entirely avoided by hot working, and, in addition, the reheating to a point just above the critical, or transformation, range tends to refine the grain structure and make it more uniform.

Hot-Worked and Oil-Quenched Joint Bars—The hot-worked and oil-quenched joint bar, at the present writing, is the only heat-treated rail-joint bar in commercial use. Reheating of the bars after hot working is not required because the operations of punching, slotting, and straightening can, with proper equipment, be done in a very short period and the bars delivered to the oil bath at a temperature satisfactory for quenching.

A joint-bar shop is usually equipped to handle any type of rail-joint bar by any one of the approved methods. To expedite handling of the bars, the shop is provided with a cold shear for cutting the bars to length, a reheating furnace for heating prior to working, a punching machine for punching the bolt holes, a similar machine for slotting the bars, a press for straightening them, and an oil tank for quenching. All these appliances are arranged in series in the order given, so that each bar may be passed easily and quickly from one to the other. The punches and press are set close to the delivery end of the reheating furnace. An endless-conveyor chain running at a uniform speed is employed to carry the bars from the straightener to the oil tank. The tank is rectangular in shape and provided with a link-chain conveyor which slowly carries the bars through the oil, the speed of the conveyor and its direction of travel being so regulated that the bars, upon entering at one end of the tank, are carried down into the oil, across the tank, and up to the opposite end. Adequate cooling and circulating facilities for the quenching oil are included so that the oil may be maintained at an optimum quenching temperature.

Welded Rail Joints—The joining of rails by welding the joints is being used by many railroads. The thermit process, the electrical method and the butt pressure method have all been used with the butt pressure method being most generally accepted for railroad installations. One result of the application of welded rail joints has been to increase the demand for rails in long lengths of 60 feet or more, which are produced at the present time in limited quantities. A typical example of welding rail joints as practiced by one of the major railroads in this country is as follows: Rails to be welded are held together end to end and a saw cut made at the junction of the two rails in order to produce parallel, clean and smooth end faces. A heating head with multiple orifices brings the two rail ends to welding temperature and then pressure is applied through clamps placed on the rails. After the welding is completed, the upset metal at the weld is scarfed off using a frame to protect the rail surface adjacent to the weld. Subsequent to the removal of the welding flash, a normalizing treatment is applied to the weld area.

Chapter 29

STRUCTURAL AND OTHER SHAPES

SECTION 1

EQUIPMENT FOR PRODUCING SHAPES

In rolling-mill parlance, the word "shape" is used interchangeably with the word "section" in describing forms of rolled material (except for geometrical shapes, which are known as rounds, squares, hexagons, etc.). Shapes, or sections, are normally divided into two classes, structural and other sections. Structural sections include standard items, such as I-beams, channels, angles, and wide flange beams, and special sections such as zees, tees, bulb angles, and car-building center sills. Other sections include such miscellaneous shapes as sheet piling, tie plates, cross ties, and those for special purposes.

The production of shapes, as enumerated above, involves a number of processes which are generally common to all of them. These processes include heating of the bloom, rolling to proper contour and dimensions, cutting while hot to lengths that can be handled, cooling to atmospheric temperature, straightening, cutting to ordered lengths, inspecting, and shipping.

The heating of the bloom for large sections is done in either of two types of furnaces, the in-and-out, or the continuous, which are described in Chapter 21. The in-and-out furnace is the more common of the two and serves nearly all of the older structural mills. A typical mill uses three furnaces of this type, having hearth areas about 18 feet by 36 feet. The newer mills tend to use continuous furnaces because of the greater economy, one or two continuous furnaces being sufficient. Practical widths of this type furnace can accommodate blooms up to 30 feet long, and one furnace of proper length, designed according to the cross-section of blooms to be heated, can have sufficient capacity to supply a mill. To hold heat loss to a minimum, furnaces are usually located adjacent to a bloom-storage yard, or the delivery table from a blooming mill, or both, and at a minimum distance from the first stand of the mill on which shapes are to be rolled.

A typical mill for the production of structural sections is shown schematically in Figure 29-1. It has a two-high reversing breakdown stand in which the initial shaping is accomplished, followed by a group of three stands, arranged in train, where the rolling process is completed. The first of these three stands, known as the rougher, is three-high. The middle stand, which is known as the intermediate, is also three high, and continues the formation of the shapes to almost finished dimensions. The finishing stand, which is usually two-high, establishes the final shape of the rolled section. Under some conditions it is desirable to have a three-high stand for the finisher. The two-high breakdown stand is fitted with rolls of cast steel, either carbon or alloy, with pitch diameter typically 36 inches, and body length 80 inches. The roughing, intermediate, and finishing stands, which are sometimes referred to

collectively as the finishing mill, usually employ rolls with a pitch diameter between 28 inches and 33 inches, and a body length of 68 inches. Cast rolls are used on these three stands, usually of carbon or alloy steel, grain or sand iron (see Chapter 23), the selection being based on service requirements.

In modern mills all of these rolls are driven by electric motors, with direct-current variable-speed reversing motors being essential for the breakdown stands, and preferred for all stands. On breakdown mills, the motor drives through a flexible connection to a two-high set of pinions, and through spindles to the rolls. Most structural finishing mills drive the three stands from a single motor, through a single three-high set of pinions, the drive being carried by spindles from the pinions to the roughing rolls, and by other spindles from the roughing rolls to the intermediate rolls. Two more spindles, connected to the middle and bottom rolls of the intermediate stand, drive the finishing rolls. Horsepower requirements vary, with 5000 horsepower being typical for breakdown stands, and 6000 horsepower typical for the finishing mill.

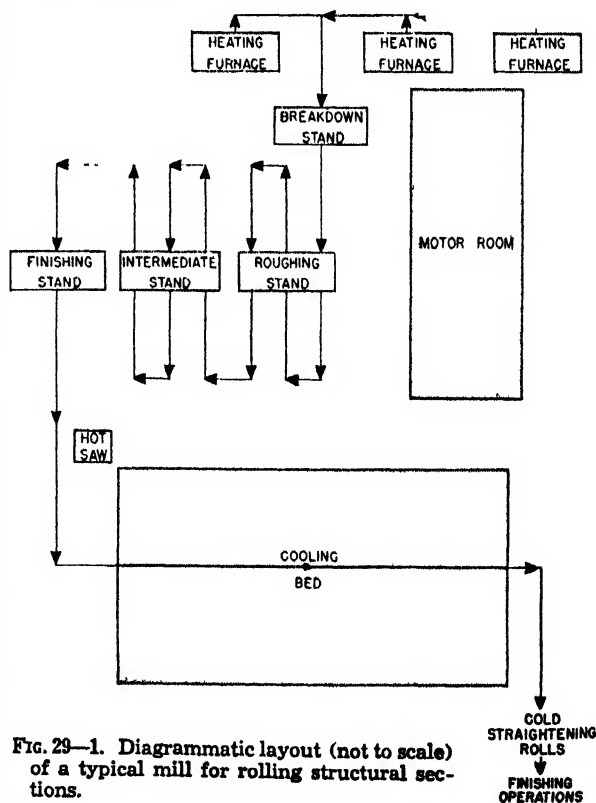


FIG. 29-1. Diagrammatic layout (not to scale) of a typical mill for rolling structural sections.

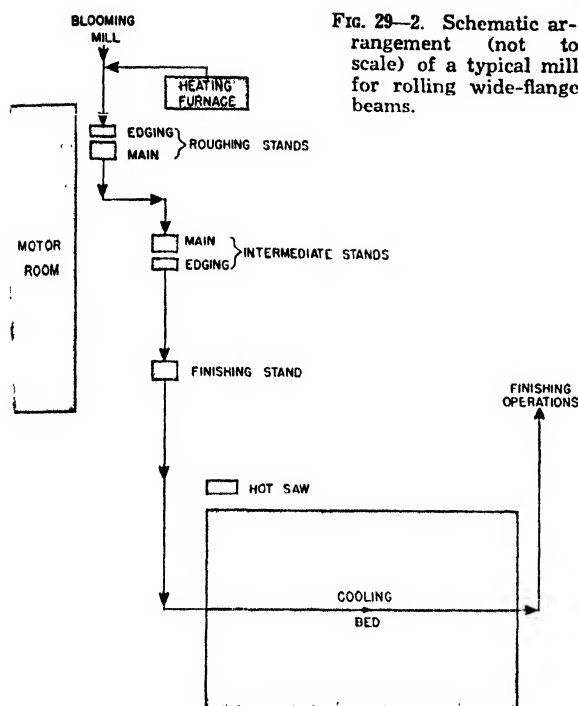


FIG. 29-2. Schematic arrangement (not to scale) of a typical mill for rolling wide-flange beams.

The breakdown stand is normally served by stationary roller tables equipped with mechanical manipulators. The shape is conveyed into, and received out of, the finishing mill passes by traveling, tilting roller tables.

The mill equipment for the production of wide-flange beams is substantially different from that used in the production of other shapes. Two or three groups of roll stands may be used, with each group containing more than one set of rolls. Figure 29-2 shows the schematic arrangement for a typical mill. This mill has a roughing group of stands which consist of a two-high roughing edging stand, closely associated with a roughing main stand, which has two horizontal main rolls and two friction-driven vertical rolls in a single vertical plane. The rolls in the roughing edging stand have a nominal diameter of 30 inches at the working zone and a body length of from 20 inches to 48 inches varying according to the depth of section rolled. The roughing main rolls have a nominal body diameter of 52 inches with a body length matching the edging rolls. Vertical rolls in the roughing stand, having a nominal diameter of 42 inches and a face width of 18 inches, are friction driven by the shape about a roller bearing in their bore. The intermediate group of stands is identical in all of the above particulars to the roughing group, with the exception of the edging stand being on the opposite end of the main stand. The finishing stand resembles the roughing and intermediate main stands in that it has main and vertical rolls of the same size in like arrangement, but does not have edging rolls. The general arrangement of the mill is such that the hot shape from the blooming mill or reheating furnace enters first into

the roughing edging stand and before clearing that stand enters the roughing main stand. The intermediate group of stands is placed a minimum distance to the side and one maximum shape length beyond the roughing group. The main intermediate stand is on the side toward the rougher and is closely followed by the intermediate edging stand, again arranged so that a single shape is in both stands of the group simultaneously ($12\frac{1}{2}$ foot centers). The finishing stand is in line with the intermediate group and 186 feet beyond the intermediate edging stand. Stationary roller tables are provided throughout with a short transfer mechanism to move the shape from the line of the roughing group of stands to the line of the intermediate group. All of these stands are driven by direct-current reversing motors, each individual stand having a separate motor, two-high pinions, and the necessary spindles. The roughing and intermediate main stands are driven by 7000-horsepower motors, and their respective edgers by 2000-horsepower motors. The finishing-stand motor is rated at 4000 horsepower.

Rolled shapes are further processed with equipment which is substantially the same for shapes produced on the standard type of structural mill as on the wide-flange-beam mill.

Removing ends not fitted to proper section, commonly called "crop ends," and cutting the hot shape into lengths which can be handled in further processing, is done with a hot saw. This equipment consists of a toothed circular saw blade mounted on a sliding frame and driven at high speed by an electric motor. Blades range up to 66 inches in diameter and copious water cooling helps maintain the cutting edges. The saw and its drive are moved on a sliding base at right angles to the hot shape so that the revolving blade cuts through the stationary shape.

Cooling of the rolled shapes is accomplished by placing them on a cooling bed which is a steel structure, arranged to support a continuous layer of shapes, while providing for a maximum circulation of air upward around them. A mechanism is provided to pull the shapes sideways into place and to slide them across the bed onto the discharge table.

Large shapes are straightened by roller-type straightening equipment, or a gag press. The former normally consists of seven or eight cast-iron or cast steel rolls assembled in a single housing with a single drive, driving either part or all of the rolls. The top rolls are placed midway between the bottom rolls and may be moved vertically by screws. All rolls are arranged for axial adjustment. The gag press consists of a horizontally-reciprocating ram midway between two support points on a platen which is so arranged that it can be moved closer to or further away from the ram, thereby varying the amount of bend made in the shape.

Cold cutting to final length is accomplished by shearing or cold sawing. The shears consist of a stationary bottom blade over which the shape is positioned, and a top blade which is forced down on the top of the shape to cut it through, either with a simple shearing action or by punching a slug, or short piece, out of the shape. The cold saw is similar in design and action to the hot saw, but is fed at a substantially slower rate.

SECTION 2

ROLLING METHODS AND PROCEDURES

The practice of heating blooms for rolling into shapes varies with the quality of the steel, the size of the bloom, and the temperature at which the blooms are

charged. A typical mill charges ordinary carbon-steel blooms received from the blooming mill at about 1800°F and heats them to 2250°F in approximately 45

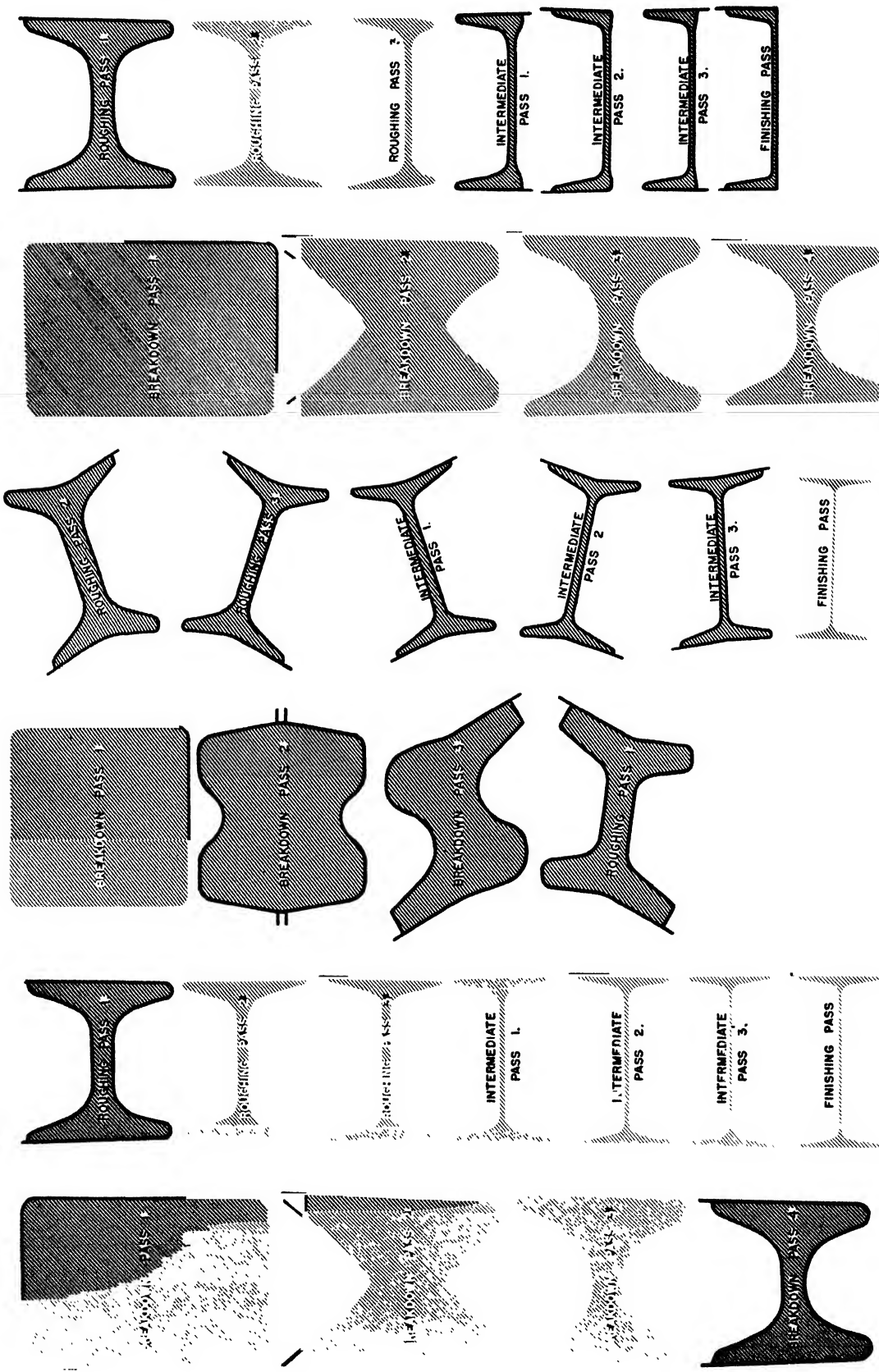


FIG. 23-5. Roll passes for producing a channel, using early passes shared for rolling beams.

FIG. 23-4. Roll passes for rolling beams by the diagonal method.

FIG. 23-3. Roll passes for rolling standard beams by straight-flanged method.

minutes. On a mill using three in-and-out type furnaces, normal operations find one furnace being charged, one heating, and the other being drawn, at a given time. Charging of the single layer of blooms in a furnace is begun as soon as the drawing has been completed. When steel at atmospheric temperature is charged, the heating to 2250° F requires about 2½ hours for the average size bloom. Handling the blooms into and out of the furnaces is accomplished by charging machines which grip the blooms on their sides. This necessitates a space between adjacent blooms of some six or eight inches as clearance for the charging-machine tongs.

Rolling—Heated blooms are deposited by the charging machine on the breakdown-stand approach table, which conveys them to the rolls. The manipulators, on the entry side of the mill, are brought into play to align the bloom with the first pass and to turn it about its longitudinal axis, if necessary. When properly aligned, the table rollers are revolved to feed the bloom into the first pass of the rolls. The rolls for the breakdown-stand generally have three or more pass grooves, some of which may be rectangular blooming passes. The number of different shaped grooves in the rolls, and the number of passes made in each groove, vary with the section being rolled. After making the required number of passes through the first groove, the bloom is re-aligned for subsequent passes by the manipulators.

Blooms processed through the breakdown-stand progress over the stationary rollers of the delivery table to the traveling tilting table which carries the bloom into line with the first pass of the roughing stand. Since the finishing-mill rolls are not reversed during operation, only a single pass is made through each groove. Similar tables on the opposite side of the finishing mill receive the shape from the first groove and enter it into succeeding grooves. Roughing- and intermediate-stand rolls usually contain from two to five pass grooves, while finishing-stand rolls are generally limited to a single pass on a shape. Finishers usually have duplicate grooves for the same section, or provide grooves for a variety of sections.

The type of section resulting from the rolling process is determined by the shape of the various pass grooves through which the material progresses, with optional groups of pass shapes frequently being available for the production of a given section. In the case of standard beams, at least three different systems of passes may be used.

The most common pass shapes used in rolling standard beams are those of the straight-flanged method which are shown in Figure 29-3. This method owes its popularity to the large range of web thicknesses which can be produced from a single set of rolls, the ability to use the early passes for the production of channels as well as beams, and the small thrust loads transmitted by the rolls to the bearings. The versatility of the weights produced results from the relatively small total taper in the live or open flanges which permit rolls to be separated to produce heavier webs with a minimum thickening of the live flange. The most undesirable feature of this method is the relatively slight taper in the pass sides which reduces the gain on dressing and results in passes getting progressively wider and flanges relatively heavier when dressed, thereby limiting the life of the rolls.

The butterfly method of rolling beams closely resembles the straight-flanged method. The outstanding difference is the fact that the live flanges are bent out to a much greater degree in all passes preceding the finishing pass. This results in greater ease of restoring

the open flanges by dressing, but imposes a serious limitation on the range of web thicknesses that can be produced in a given set of passes.

Figure 29-4 illustrates the passes used in the diagonal method of rolling beams. This method makes good provision for the restoring of both flange thickness and pass width by dressing, but involves serious restrictions in the web thicknesses that can be satisfactorily produced from a single set of rolls, and vastly increases the thrust loads that are imposed on the bearings which support the rolls. Since thrust-bearing wear permits longitudinal motion of the rolls with corresponding changes to the thickness of the open flanges, rolling difficulties result.

The method of rolling a channel using early passes shared with a standard beam is illustrated in Figure 29-5. This plan results in a smaller roll inventory, good flange control in the production of a large range of weights, narrower passes and stronger collars than can be had if the butterfly method is used. Figure 29-6 shows the butterfly design. It has the advantage of producing channels which are filled to proper section very close to the ends of the rolled bars. However, this method of production results in relatively thin weak collars on the rolls and almost precludes the rolling of multiple weights. This is caused by the extreme thickening of the flanges in the early passes when the rolls are separated to produce a thicker web.

Angles also offer the roll designer a choice between a number of proven rolling methods. Those most generally approved today may be referred to as the butterfly and the flat-and-edge methods. Typical passes for the butterfly angle are shown in Figure 29-7. The current popularity of this method is due to the relatively small crop loss on the shapes, the absence of vertical rolls or turning for edging, and the ease of controlling the bending, which is slight in any one pass. Undesirable features of the design are the lack of proportional thickening of the various parts of each leg when rolls are separated to make heavy weights, and somewhat poorer control of leg length.

The flat-and-edge type of design can be seen in Figure 29-8. This method is popular on those mills having vertical edging rolls and is sometimes used where the shape is turned 90° for edging in a pass in horizontal rolls. Leg lengths are readily controlled in the edging passes and thicknesses remain uniform when rolls are set to produce heavy weights. These advantages are frequently offset by the difficulty in maintaining proper entry alignment of the section in the later passes where bending is drastic. Turning long lengths of hot, flexible steel for edging, where that is necessary, is also difficult.

The special structural sections include shapes rolled rarely or for a single purpose. The passes used for the production of zee bars involve the same principles as the rolling of angles by the butterfly method, with each half of the pass resembling an angle pass, and with the two halves being fitted together reversed. The half center-sill section, used in making modern railroad cars, is a special type of zee bar, being irregular in both thickness of the members and flange lengths. Passes for its production are included here (Figure 29-9) because of the relatively great demand for the section.

Modern practice tends towards the production of tees by splitting a beam of proper size through its web to make two of the required sections. Some tees, however, are still rolled as such. The rolls are unduly complicated and the actual rolling is difficult. Where a choice is offered, the rolling of tees is usually avoided because of the necessity of turning the section between

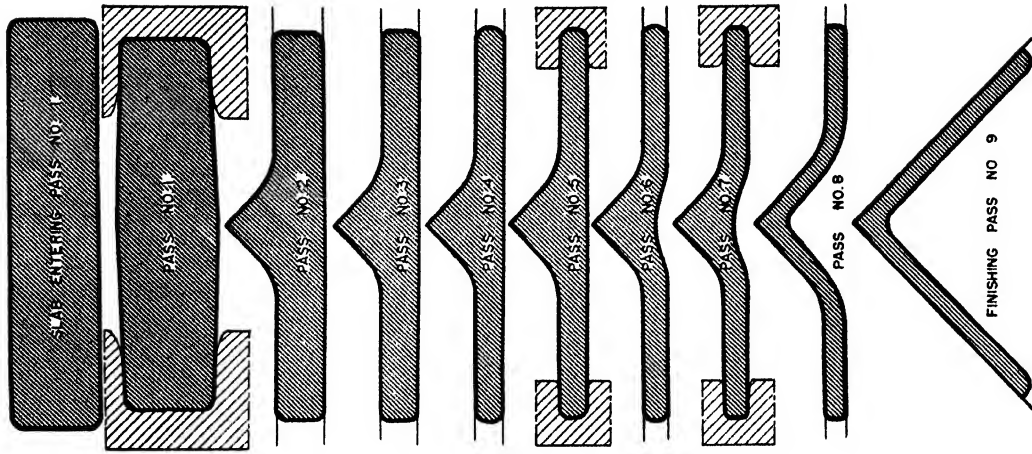


FIG. 23-8. Roll passes for producing an angle by the flat-and-edging method.

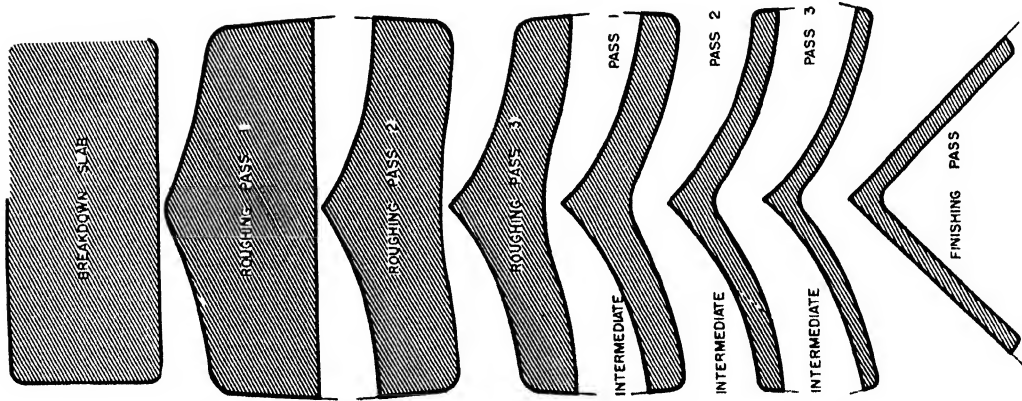


FIG. 23-7. Roll passes for rolling an angle by the butterfly method.

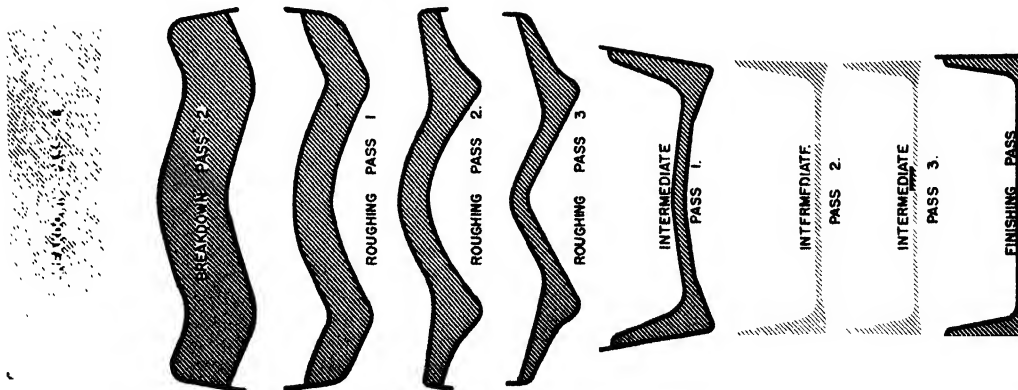


FIG. 23-6. Roll passes for rolling a channel by the butterfly method.

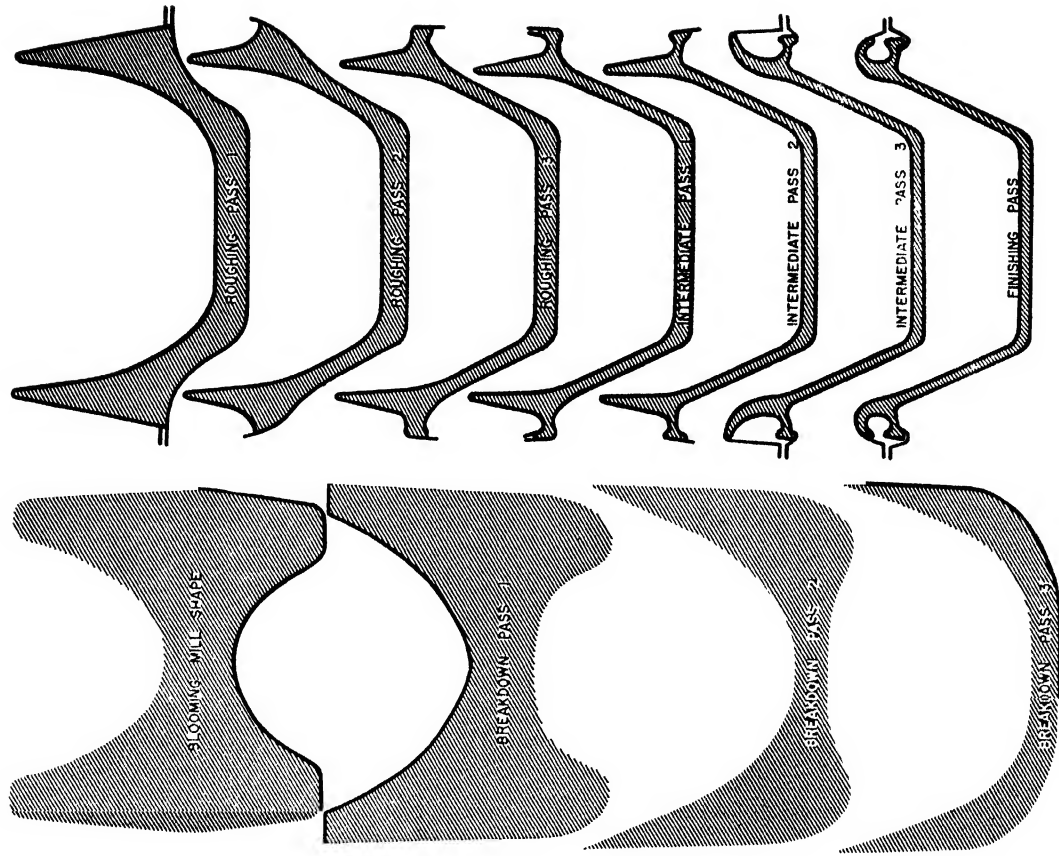


FIG. 29-10. Roll passes for rolling an arch web section of sheet piling, showing how flange is bent to accurate configuration in the leader and finishing passes.

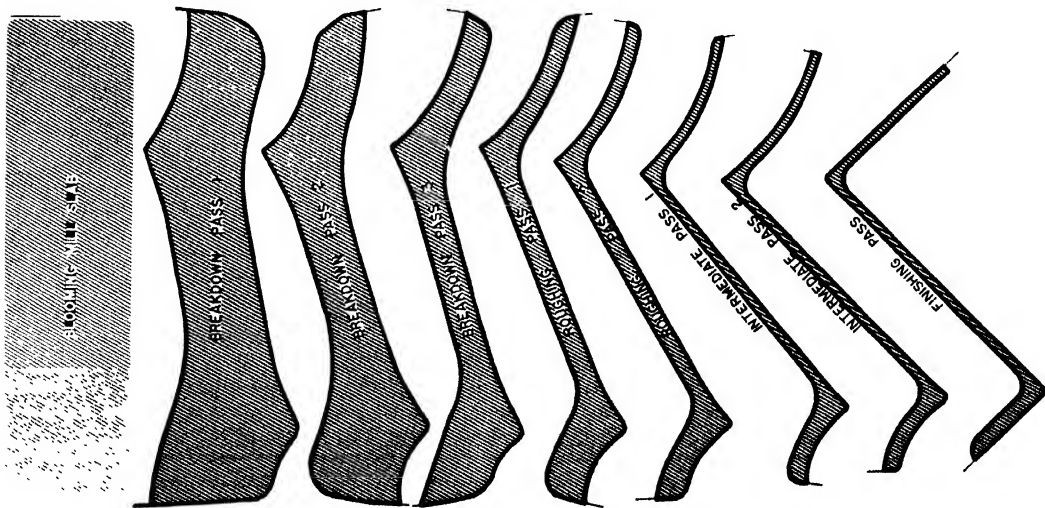


FIG. 29-9. Roll passes for the production of a half center-sill section, a special type of zee bar.

passes to permit each of the two tapered flanges and the parallel stem to be worked alternately so far as possible, in open and closed grooves.

One of the most interesting groups of sections produced upon standard structural mills is the sheet-piling group. Usually produced in three general types, known as straight web, arch web, and zee, these sections have been adapted as wall members in both permanent and temporary structures requiring strong vertical walls for lateral support, such as coffer dams, piers, dykes, breakwaters, etc. In the rolling of piling sections, the usual problems are complicated by the necessity of bending the flange which has been rolled straight to form the interlock. Precise control of the bend is necessary since proper clearance within the interlock must be maintained and the resulting opening between flange tip and thumb must be within close limits. The particular arch web section for which passes are shown in Figure 29—10 accomplishes this bending of the flange in two steps in the leader and finishing passes. Attention should be given to the fact that the shaping of the section starts in the blooming mill. This is typical of most piling sections and large beams, and is the result of the large size of the sections coupled with their intricate shape which precludes getting enough passes into the limited space of the breakdown and finishing mills to properly form the section.

While the rolling procedure for a section on the standard type of structural mill does not normally employ the same pass groove for more than one reduction in the shape, multiple passes being limited to the breakdown stand and seldom exceeding three passes in a groove, the opposite situation exists on the wide flange beam mill. Here as many as fifteen successive passes are taken through the main and edging stands of the roughing group. In a typical layout, the main and vertical rolls of the roughing main stand and the rolls of the roughing edging stand are moved to new settings with respect to each other by an automatic screwdown control. With this mechanism, the operator sets in advance of rolling the reduction to be taken on each pass. Then, pass by pass, he advances the lever of the master control switch a notch, with each of the three sets of rolls of the roughing group moving closer together simultaneously to the new settings. The control equipment is so designed that each of the three sets of rolls can be made to move different amounts, thus permitting proportional rather than like reductions on flange and web.

The equipment of the intermediate group of stands is practically the same as the roughing equipment. As in the rougher, fifteen passes are provided for in the control equipment. In practice, the passes are divided be-

tween the roughing and the intermediate stands in a proportion that gives equal duration of rolling cycle in each of the two stands. Since the shape at the roughing stand is relatively heavy and short, more passes are required there to balance the relatively few passes made on the elongated shape at the intermediate stands. Passes in actual use range from fifteen roughing and nine intermediate on sections requiring heavy overall work, to five roughing and three intermediate on the sections requiring a minimum of work; nine roughing and five intermediate passes represent the average condition. In all cases the intermediate passes are followed by a single finishing pass. In rolling wide-flange beams it is customary to roll a bloom which has, as nearly as possible, the same proportions as the finished beam. Each succeeding pass in the rolling cycle strives to maintain these proportions, resulting in reduction, pass by pass, being proportional. Since at no time after leaving the blooming mill are the flanges rolled in a closed groove, the loss in flange length is very small, and the rolling of beams with extremely high flanges is entirely possible.

Rolled material from either of the described structural mills is delivered by roller table to the hot saw. The blade of the saw is normally revolved in such a direction that the cutting edge is moving downward toward the shape. The action of the rapidly revolving toothed blade combines a mechanical sawing and melting of the steel, the chips when thrown clear frequently being molten. Cuts are usually made at the hot saw to remove the crop ends, to part the usable material into lengths that can be handled for further processing, and to provide short test pieces. Tests are cut for the guidance of the roller in correcting defects in the section and for various mechanical tests which are part of the inspection procedure.

Cooling is accomplished by the natural circulation of air about the shapes on the cooling bed. The rate of cooling can, to some extent, be controlled, and is of relative importance. Cooling can be accelerated by maintaining space between shapes on the bed, and by providing ample space below the bed for the passage of cool air. Retarding of the cooling cycle results from putting the hot shapes on the bed in a solid touching layer. If cooling is too drastically retarded, the size of the hot bed must be increased or rolling delays will result. On the other hand, too rapid cooling may result in mechanical properties outside the ordered range. Non-uniform cooling results in warpage of the shapes as they cool. This must be avoided in symmetrical shapes, if possible. Most non-uniform shapes warp during cooling, sometimes to the point that they are difficult to convey to the straightening machine.

SECTION 3

FINISHING AND INSPECTION

When the shapes have been properly cooled they are conveyed again by roller table to the straightening machine. For the products rolled on the standard type of structural mill and the smaller wide-flange beams, a roller straightener is normally used. The first shape to be straightened is stopped in the machine while the rolls are adjusted laterally to it. The top rolls are then moved downward to deform the shape with each successive top roll making less deformation than the preceding one. This shape is backed out of the machine and then run through for most of its length. If not straight to the eye of the experienced operator it is

again reversed in the machine. Further adjustment is made with this process being repeated until a straight shape is obtained. Subsequent shapes will require only slight alteration to the setting of the machine to compensate for wear and other variables. The larger wide-flange beams and some few standard mill products are straightened in the gag press. Here the process consists of advancing the stationary platen and the shape towards the reciprocating ram until a bend is accomplished in the shape. The operator has full control of the point on the shape at which the bend will be made, the amount of the bend, and by turning on the table,

the direction of the bend. Bending strives only to offset initial crookedness.

Cutting to final length is accomplished by one of several means. The most commonly used process is cold shearing. It is applicable to angles, channels, standard beams and the smaller sizes of wide-flange beams. It is relatively inexpensive and results in good uniformity of length with minimum distortion of the sheared end. Cold sawing is next in popularity, and is used on most of the wide-flange beams, the larger sizes of standard beams, and on sections, such as piling, where shear distortion of the ends would make the section unusable. Cold sawing is more expensive than shearing and results in a ragged burr which must be removed by chipping. Flame cutting is available for extremely heavy sections and unusual applications. Where overall length must be accurate, as in the case of columns, the ends may be milled. This results not only in close length tolerance, but in a reduction in the ends' out-of-square allowance as well.

Before the products rolled on the various mills may be shipped, they must be inspected for defects and tested for certain mechanical properties. This is accomplished by two general methods. First, the test cuts taken at the hot saw are sent to the laboratory where tests are made to determine the mechanical properties of the material, such as tensile strength and yield point. The other method of inspection is visual inspection of the material on an inspection bed where the shapes are inspected for metallurgical defects, such as pipe, blister

and scabby surface, and for dimensional variations such as length, straightness, and out-of-square condition. Dimensions such as web or flange thickness, flange height, weight per foot, etc., are checked on tests taken at the hot saw by the mill inspector, and the roller then adjusts the mill to correct improper conditions. Special sections may require additional inspection. An example is the slot and thumb on sheet piling. Chemical composition of all material is known, as a record is kept of all rolled products showing the number of the heat from which it was rolled.

The final steps are to separate and assemble for shipment. The material then is loaded in freight cars, river barge, or truck, and shipped to its destination.

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Chapter 30

MERCHANT-BAR PRODUCTION

SECTION 1

MILLS AND THEIR PRODUCTS

The name "merchant mill" is said to have been given in the early days of rolling to those bar mills which carried a stock of their product from which merchants selected bars at their convenience. The name has been carried over from that time and is still applied in the designation of modern bar mills, particularly those on which some of the small shaped sections are rolled. In other plants, the mills are designated simply as bar mills.

Other designations sometimes used are "hand bar mills" and "guide mills."

In the early mills, rounds and sharp-cornered squares were rolled by hand, that is, the bars were prevented from turning over in the passes by a tongsman, who forcibly held the bars in the proper position in the grooves throughout their passage through the rolls. As a general rule, the finishing stand of rolls was two-high

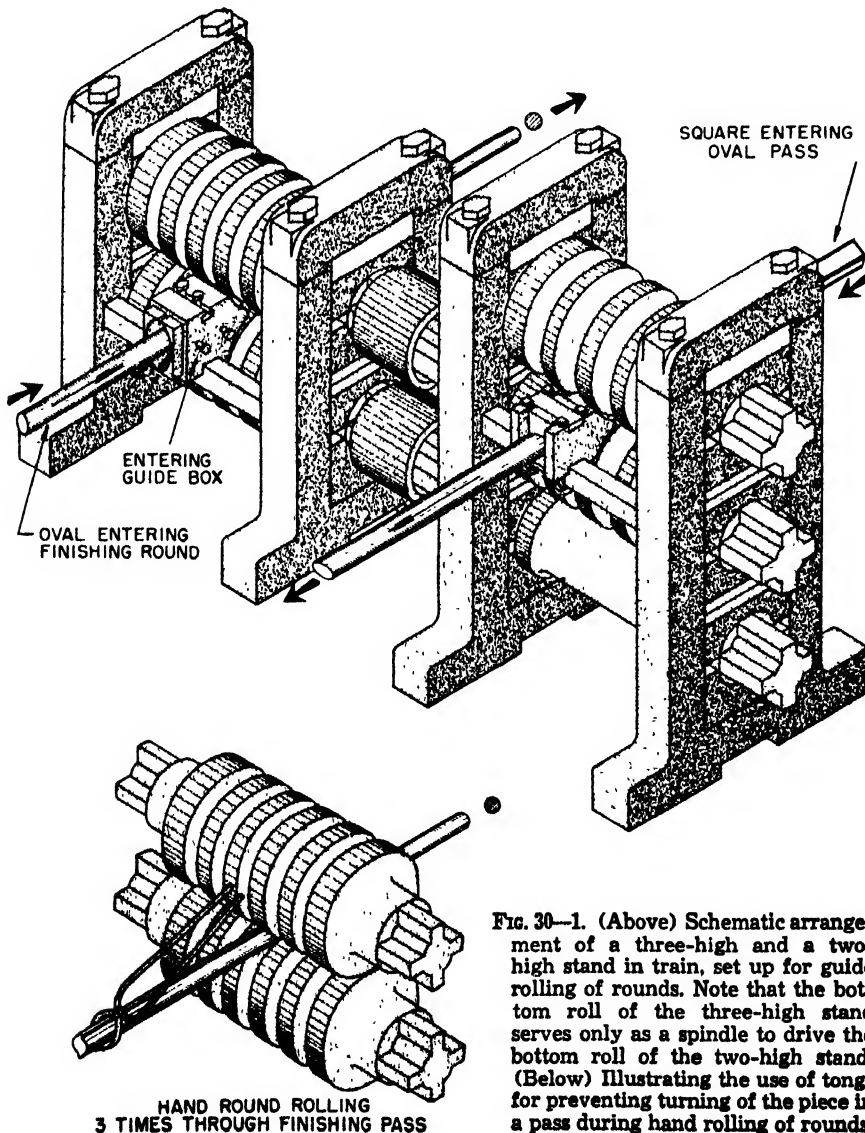


FIG. 30-1. (Above) Schematic arrangement of a three-high and a two-high stand in train, set up for guide rolling of rounds. Note that the bottom roll of the three-high stand serves only as a spindle to drive the bottom roll of the two-high stand. (Below) Illustrating the use of tongs for preventing turning of the piece in a pass during hand rolling of rounds.

in these mills, though in rare instances a three-high stand was used. In the two-high mill, the bar delivering from a given pass was returned over the top roll for entering into the next pass, and so on until the section was finished. This type of rolling was arduous and the length of bar that could be rolled under these circumstances was limited to about 16 to 20 feet.

This limited length that could be rolled, in combination with the heavy labor involved, led to the adoption of the so-called guide mills, in which rounds and sharp-cornered squares were held in the passes by close-fitting entering guides. To do this for rounds, it was necessary to devise a new pass design in which a square (or gothic square) was entered into an oval pass and the resulting oval was then entered on edge into a round finishing groove. By using an oval, closely fitting entering guides could be used that would hold the oval section in its proper position as it passed through the round groove.

In the older hand-round method, bars entering the finishing groove were so nearly round that no enter-

ing guide could be devised which would prevent the bars from turning over in the grooves. Thus the guide-round design permitted the rolling of long lengths and this method has practically superseded the hand method, although, for various reasons, there are mills in which rounds still are rolled by hand. Figure 30-1 illustrates both the hand and guide method of rolling rounds.

The mills in which rounds and squares were rolled by hand were referred to by the early operators as "bar mills," whereas the mills using guides to hold up the bars were referred to as "guide mills." What can be called the first bar mill was the two-high mill designed by Henry Cort in the latter part of the 18th century. Many of the early mills were driven by water wheels, and it is known that two water wheels were used in some cases, one for each roll. Later, steam-engine drives became common.

Crude as these early mills were, they were vastly superior to those in use prior to Henry Cort's time. With these early mills as a start, the bar mill gradually be-

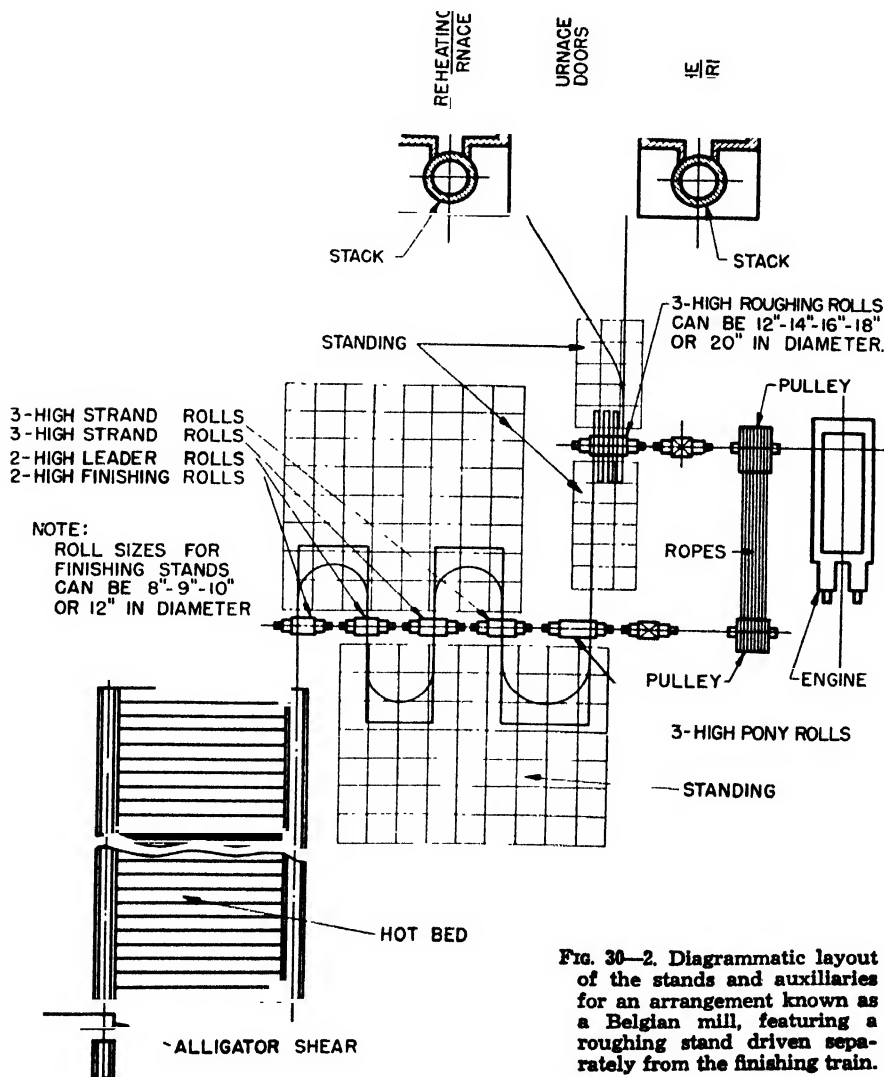


FIG. 30-2. Diagrammatic layout of the stands and auxiliaries for an arrangement known as a Belgian mill, featuring a roughing stand driven separately from the finishing train.

referred to as a **Belgian mill**, as it was said to have originated in Belgium. The mill could roll longer lengths and finish them at a higher rate of speed and, consequently, could produce greater tonnage than earlier mills.

As the next innovation, the Belgians and Germans introduced the looping of bars. In this practice, the bars in the finishing train, when they reached a stage in their reduction at which they were relatively flexible, were caught by the tongsmen on the front end as they emerged from the rolls and pulled around in a half circle and entered into the next pass. As a result of this practice, rolling time for a bar was appreciably reduced and more bars could be rolled per hour, producing greater tonnages. Some of the more improved mills of this kind had two or three roughing stands in train, on the same housing shoes, which permitted still greater production because the roughing passes could be divided among the two or three stands.

After the looping of bars became common practice, means were sought to eliminate, so far as possible, the manual looping of bars. A trough called a repeater was devised to conduct bars from one stand to another. The first successful repeater was put into use by W. McCallip, superintendent of a small bar mill in Columbus, Ohio, in 1877. For the most part, repeaters are used to direct the bar out of a square pass into the succeeding oval pass, while the tongsmen direct the ovals into square passes. Some mills also use repeaters for oval into square, in which case the oval must be twisted so that its longer axis is in a vertical position as it goes around the repeater; otherwise it will jump out of the trough.

In Europe, the production of the Belgian-type bar mills was further increased when two Austrians perfected a repeater for three-high roughing stands. This repeater was a trough which conducted the bars, as they emerged from the passes, around a half circle and back into the next groove in the same roll stand, obviating the use of a tongman who could not, in any case, work in such close quarters as would be necessary on these roughing stands. The repeaters were used on the last two or three passes in the roughing operation, when the bars were small and long enough to provide the necessary flexibility. Figure 30-3 shows a three-high mill repeater in use on the roughing stand of a mill in which the finishing stands have been divided into two groups, the second group being run faster than the first for increased production. The finishing stands employ two-high repeaters for looping square into oval. Some mills in the United States use three-high repeaters of this type.

Further improvement of the bar mills became possible with the introduction of the continuous roughing principle in 1882. Mills employing this principle consisted of an eight-stand tandem* continuous roughing mill and four finishing stands in line. Although such mills were first built for the rolling of rods, about 1900 several were built to roll bars from a 4-inch by 4-inch billet. Figure 30-4 shows an improved type of this mill with two additional stands added for the rolling of smaller bars.

The early mills of the type shown in Figure 30-4 had some serious disadvantages. As discussed in Chapter 19, since the cross-sectional area of a piece is reduced during rolling, the piece leaving the rolls has a higher velocity than that at which it entered, due to its being elongated. The continuous roughing stands of the early mills had fixed speeds, with each successive stand running faster than the preceding one by an amount cal-

culated to compensate for bar elongation. The fixed speeds made it imperative to have perfect balance between speed and elongation among the several stands, otherwise either the bars would be stretched or a loop would form between stands, either of which was undesirable. At the same time, the distances between stands were relatively short, causing heavy pressure on the delivery guides used to twist the bars 90 degrees for entrance into each succeeding stand. This heavy pressure often resulted in injurious scratching of the bars by the guides and these scratches, when rolled in, gave the bars a seamy appearance.

In the finishing mill the four roll stands, being in one line (in train), are driven at the same speed. As there is a reduction in cross-sectional area of the bar in each succeeding pass through the mill, and consequently a corresponding elongation of the bar, a successively longer loop forms between each stand. As a result, the back end of the bar becomes colder than the front end because of more contact with the floor and longer exposure to the air, a condition which introduces a variation in size of the ends of the bar as compared with the middle.

A number of mills were designed to overcome these disadvantages. Some had the roughing line split into

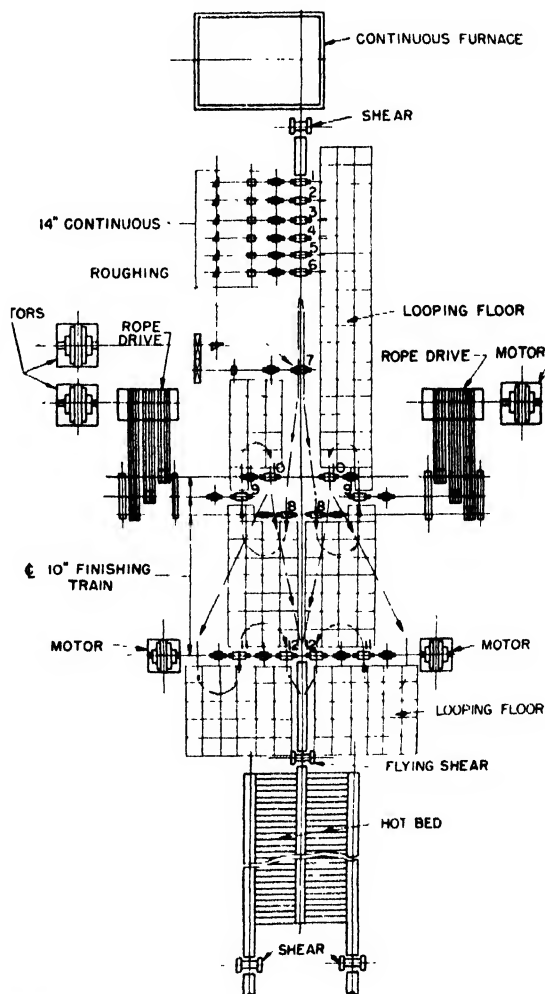


FIG. 30-5. Arrangement of a bar mill employing a 6-stand tandem continuous roughing mill, followed by a double finishing train. All stands are two-high.

* Rolling mills are said to be "in tandem" when they are arranged so that the rolled piece can progress from one stand to the next in a continuous straight path, as in the continuous roughing stands in Figures 30-4 and 30-5.

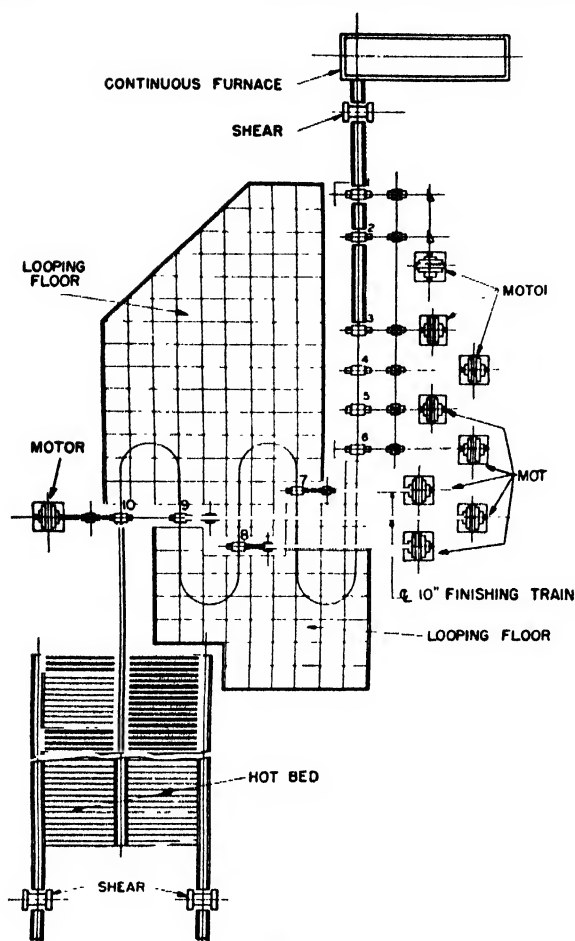


FIG. 30-6. Bar mill designed with a semi-continuous roughing arrangement followed by a four-stand continuous roughing mill, supplying roughed-down billets to the four stands of the finishing mill. All stands are two-high, and each is separately driven except for the two stands of the semi-continuous roughing set-up.

two or three groups so that the bar would run out to its full length between the groups. The finishing stands were also separated into groups of two or more roll stands, each group being separately driven and at a higher speed than the preceding one. Thus, between each group of rolls, the loop could be kept at a minimum, though they still formed between the stands which were in train. Space does not permit showing all of the arrangements devised to eliminate long loops.

A mill designed to overcome some of the disadvantages of these older mills is shown in Figure 30-5. In this mill, which has a double finishing train, the finishing stands have been divided into groups, with no more than two roll stands driven at the same speed. In each of these groups of two stands driven at the same speed, the second stand has rolls of larger diameter than the first in order to keep the loop between stands at a minimum. In a mill of this type the continuous roughing mill has sufficient capacity to supply both finishing trains, thus increasing production. Two billets can be sent through the roughing train, one for each finishing train.

Figure 30-6 shows a mill with a semi-continuous roughing arrangement in which the first two stands are driven by one motor and the bar runs free after each

stand. These two stands are followed by four continuous stands driven by individual variable-speed motors. The finishing unit is comprised of four stands, each one driven by an individual variable-speed motor at a successively increased speed. This mill, through the use of separate motors, permits excellent control over push and pull in the roughing stands, except in the first two, and, as the stands are a good distance apart, the heavy pressure on the guides experienced in the older close-coupled mill is relieved. In the finishing stands, the speed control provided by the separate motors permits close control of the length of the loop between any two passes. This mill was a considerable improvement over the older types.

Figure 30-7 shows a bar mill of the cross-country type which rolls a wide range of products. The mill has two hot-beds and finishes the large sizes out of stand No. 10. The smaller sizes are finished in stand No. 14 or No. 16. The mill is further distinguished by having a vertical roll stand immediately following stand No. 14 and continuous with it. The vertical stand provides the means for rolling off all over-sizes or shoulders formed in the horizontal stand, giving a more perfect round. So-called precision rounds, which are rounds rolled to approximately one-half standard tolerances, are finished through these two stands. The vertical stand is removed when finishing in stand No. 16 or when rolling to standard tolerances.

Figure 30-8 shows another cross-country design. This layout does not have any continuous stands and the bars run out free after each pass. The sections rolled on this mill range from $\frac{1}{2}$ -inch to $5\frac{1}{8}$ -inch rounds, squares

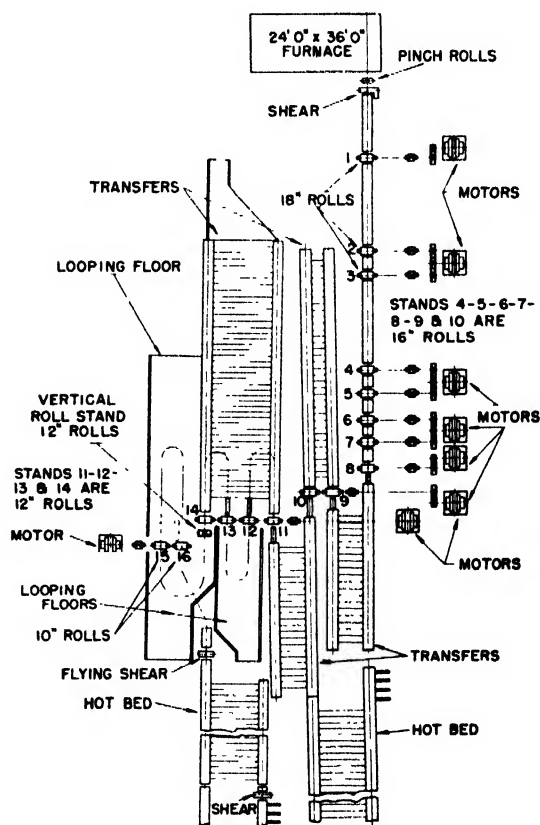


FIG. 30-7. Schematic layout of the stands and auxiliaries for a bar mill of the cross-country type, employing the continuous principle in the latter roughing stands.

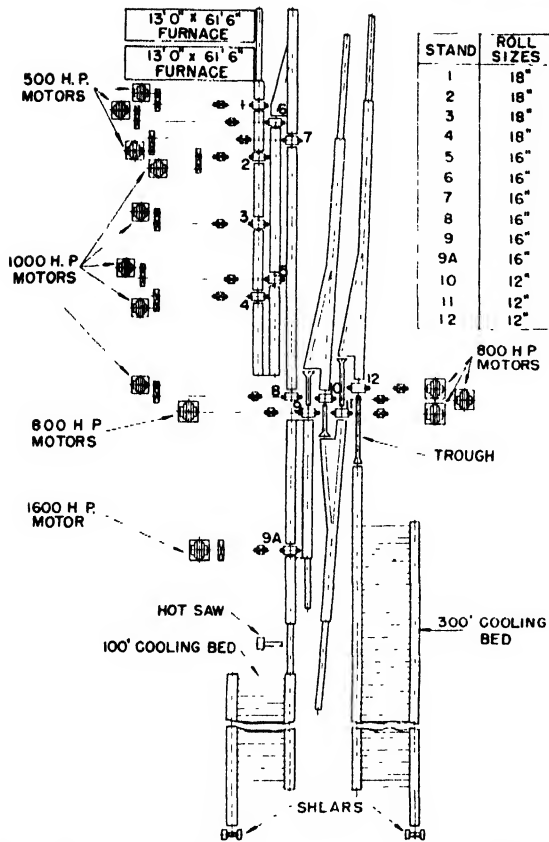


FIG. 30-8. Bar mill of the cross-country type differing from that of the preceding illustration in that no continuous stands are used and the bars run out free after each pass.

in corresponding sizes, flats from 1 inch to 9 inches in width and other products of comparable dimensions. The larger size products are finished from stand No. 9-A, which has 16-inch rolls, and the smaller sizes are finished from stand No. 12, which has 12-inch rolls. The distinguishing feature of the mill is that each stand has a variable-speed motor.

The most modern of all the bar mill designs is the continuous mill with alternate horizontal and vertical rolls, an arrangement which, curiously enough, was embodied in the first continuous mill built by Bessemer in 1862. The use of alternate horizontal and vertical stands obviates the necessity for twisting the bars, and the twist guides, with their tendency to scratch bars, are eliminated. The stands of the mill are spaced far enough apart so that a slight loop can be formed between stands. Individual variable-speed drives for practically every stand permit regulation of speed relationships to take care of bar elongation between stands, and the slight loop eliminates all push and pull between stands. The general principle of employing alternate horizontal and vertical stands was discussed in the description of the continuous billet mill at Lorain Works of National Tube Division in Chapter 25.

Mills for Rolling Light, Narrow, Flat Material—Flat thin material of narrow width is produced on various types of bar mills. This material is known as narrow strip, band iron, hoop, and cotton tie. When first produced, this type of product was rolled on mills similar to the one shown in Figure 30-2, but as mills of this

type could not roll any large tonnage of such light weight material, better mills were soon developed for the purpose.

Figure 30-9 shows a modern mill designed to roll this thin material. This continuous mill produces comparatively large tonnages of hoop, cotton tie, and narrow strip. In mills of this kind, the very narrow widths of the thin gauge materials are obtained by rolling a multiple width of the material and then slitting the rolled piece (after it is cold) into the desired widths. This permits the mill to maintain a rate of production that could not be realized if very narrow widths were to be rolled.

Rail-Slitting Mills—Rail-slitting mills were first developed when Bessemer rails were being rolled in large quantities and before open-hearth furnaces were common. Bessemer rail scrap was a drug on the market, as it could not be used in any quantity in the Bessemer vessel. The rail-slitting mills, a number of which are still in use, solved the problem of utilizing this scrap by rerolling it into other and useful sections.

Figure 30-10 illustrates a rail-slitting mill. These mills take a heated rail, usually 10 to 16½ feet long, and run it through a two-high stand of rolls with sharp collars which cut through and separate head, web, and base. Each one of these parts is then rolled into a bar of some kind directly after it is slit. The products include rounds, squares, fence posts, angles, concrete-reinforcing bars, flats, and a variety of other small sections. The continuous stands shown on the drawing are used to bend and form a flat, rolled from the base of the rail, into a cylinder which is then welded at the joint to make a tube or pipe.

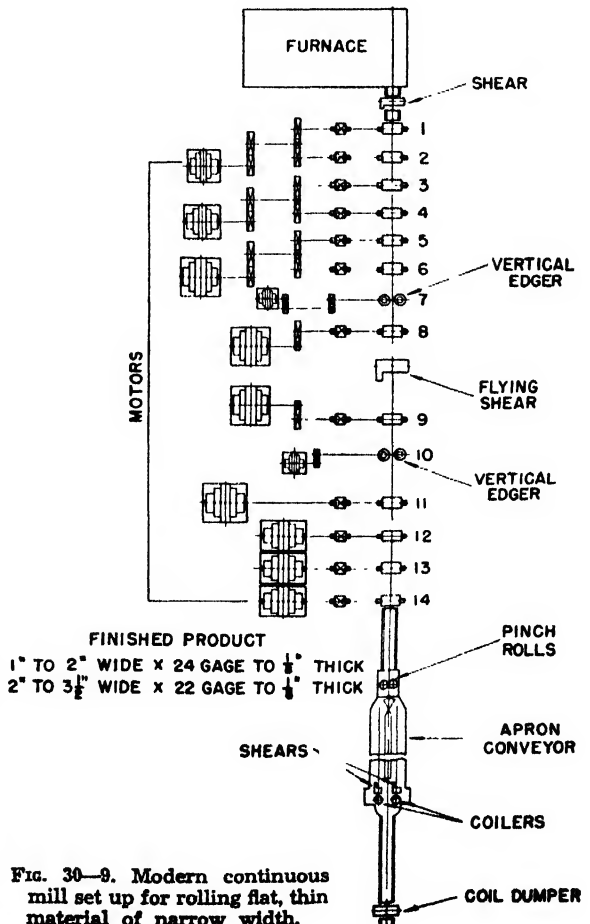


FIG. 30-9. Modern continuous mill set up for rolling flat, thin material of narrow width.

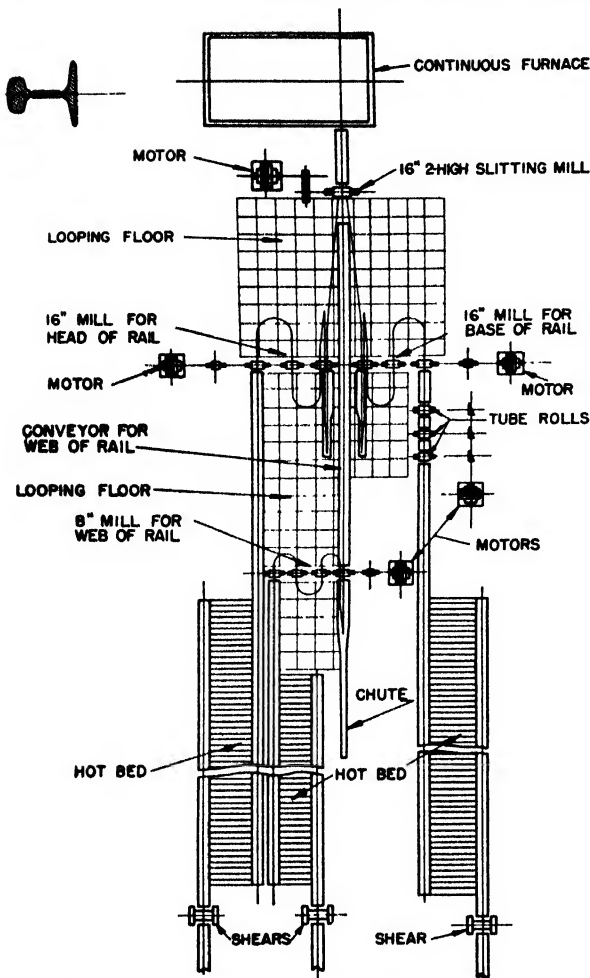


FIG. 30-10. Schematic arrangement of the slitting mill and roll stands of a rail-slitting mill. Inset at upper left shows how the head, web and base of the heated rail are separated in the slitting mill prior to rolling each, without reheating, into a bar. This particular mill has three continuous stands (marked "tube rolls") for forming a tube as described in the text.

ROLL DESIGN FOR BAR MILLS

The first roll designs for bar mills were very simple. They consisted of box passes, gothic passes, tongue-and-groove passes for flats, and so-called hand-round grooves for rolling rounds.

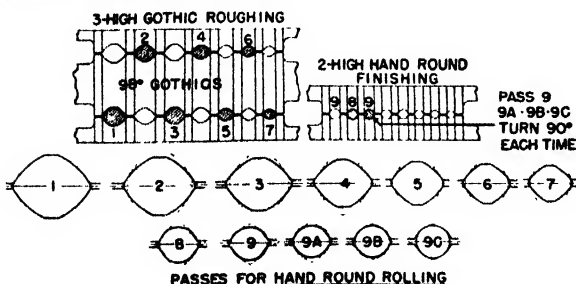


FIG. 30-11. Rolls and passes for rolling a hand round, using gothic passes in the roughing and hand-round grooves in the finishing rolls. The numbers of the sketched passes correspond to the cross-hatched and numbered pass openings between the rolls above.

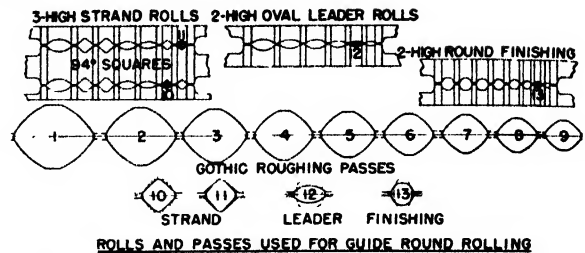


FIG. 30-12. Rolls and passes for rolling guide rounds. The roughing rolls containing the nine gothic roughing passes are not shown. Numbers on the sketched strand, leader and finishing passes correspond to the cross-hatched and numbered pass openings in the respective sets of rolls shown immediately above.

Figure 30-11 shows a design of passes and a sketch of the rolls for rolling a hand-round, using gothic passes in the roughing and hand-round grooves in the finishing rolls. Figure 30-12 shows a typical series of passes for rolling guide-rounds, using gothic roughing passes, strand open-square passes, and oval and round finishing passes. These two methods were extensively used in the older bar and guide mills.

The gothic roughing rolls are not used to any great extent today. The design has been superseded in most hand mills by the diamond roughing set. Figure 30-13 shows one of these sets, which are also used for roughing passes for hand-guide rounds. Figure 30-14 shows a line of passes for a guide round using diamond roughing grooves, oval and square strand grooves, and oval and round in the leader and finishing passes. The three-high strand rolls shown in Figure 30-12 have a combination of oval and open-square passes, also diamond passes. The open-square passes referred to are of 94-degree

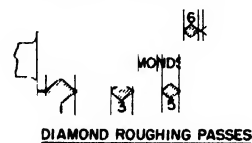


FIG. 30-13. Rolls with diamond roughing passes of the type which has largely superseded gothic roughing passes for rolling rounds by either the hand or guide method.

angularity. To make a 90-degree square using these grooves it was customary to go twice through the same size groove.

The mill with the eight-stand continuous roughing arrangement and the looping finishing mills shown in Figure 30-4 used a 4-inch by 4-inch billet of not over

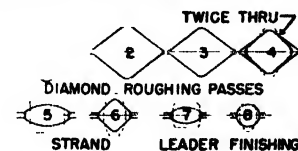


FIG. 30-14. Diamond roughing passes, oval and square strand passes, oval leader pass, and round finishing pass for guide-round rolling. Dotted lines indicate shape of piece entering pass, corresponding to shape of piece out of preceding pass turned 90 degrees.

225 pounds. The continuous stands were close together and a roll design was provided which would require the least possible twisting of the bars. The roll design on this mill provided for a $1\frac{1}{2}$ -inch round in 14 passes. The first and second passes were both flat ones in box grooves. The bar then was twisted 90 degrees into a hexagon-shaped groove (pass No. 3) and then on into pass No. 4, which was a square in the diagonal position. By using this design, only one twist of the bar was required in the first four grooves. After pass No. 4, the bar was twisted 45 degrees to enter pass No. 5, which was an oval, and from there on the passes were alternately square and oval, the bars being twisted 90 degrees in the oval form and 45 degrees in the square form, before entering the next pass. The last pass, naturally, was round. Passes Nos. 9, 10, 11, 12, 13 and 14 were looped by hand on this mill.

As mentioned earlier, mills of this type rolling short 4-inch by 4-inch billets, with their close coupled roughing stands and long loops, were not able to roll bars of close tolerance on the front and back ends of the bars. These disadvantages led to the adoption for modern continuous mills of 30-foot billets of $1\frac{1}{2}$ -inch to 3-inch squares. Rolling 30-foot billets produces bars more uniform in section throughout their lengths because the back end of such a billet is still in the furnace and retaining heat while the front end is emerging from the

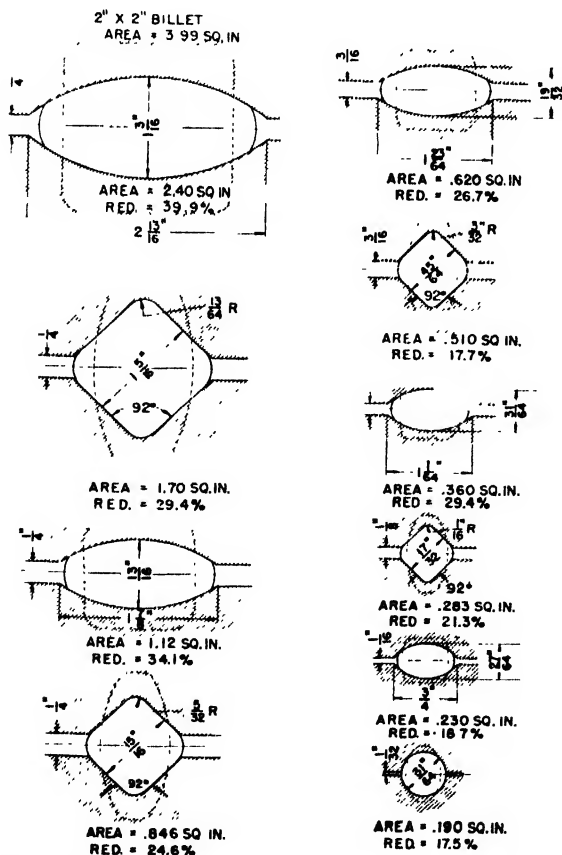


FIG. 30-15. Series of passes for rolling a 2-inch square billet into a $\frac{3}{4}$ -inch round in ten passes, using an alternate oval and square reduction down to the finishing oval and round. Dotted lines indicate shape of piece entering pass, corresponding to shape of piece out of preceding pass turned 90 degrees. Per cent reduction of the piece in each pass has been calculated from the cross-sectional areas, as shown.

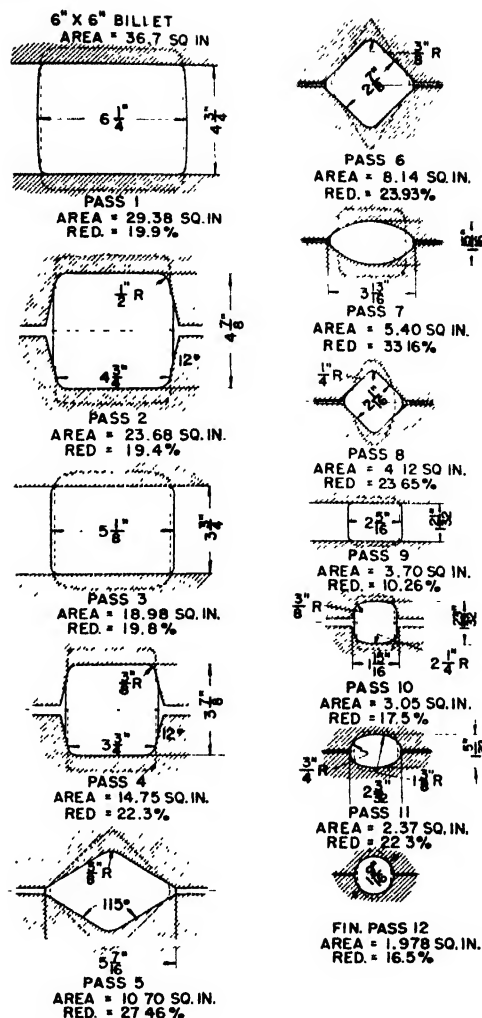


FIG. 30-16. Steps in the reduction of a 6-inch square billet to a $1\frac{1}{8}$ -inch round in twelve passes on the cross-country mill shown in Figure 30-8. Dotted lines indicate shape of piece entering pass, corresponding to piece out of preceding pass after turning the required number of degrees for proper entry.

continuous roughing mill and, as the mills generally have good speed adjustment in the finishing stands, mill loops are kept at a minimum to insure uniformity of size throughout the rolled piece.

Figure 30-15 shows the pass design for rolling a 2-inch by 2-inch billet on a modern-type mill into a $\frac{3}{4}$ -inch round in 10 passes, using alternate oval and square reduction down to the finishing oval and round.

In the cross-country mill types, somewhat different roll designs are used. Figure 30-16 shows a roll design for the mill shown in Figure 30-8, which rolls many different grades of alloy steel, including stainless steel. The design shows the steps in reducing a 6-inch square billet to a $1\frac{1}{8}$ -inch round in 12 passes. The first four passes are flat and edge, the flat passes being in plain rolls and the edging in box passes. The principal object in using these flat and edge passes is to break and free the bar of the scale that formed in the furnace during heating. In the first pass the roll pressure is on the top and bottom of the piece; the sides are free and, being compressed, crack the scale and allow it to drop off. In

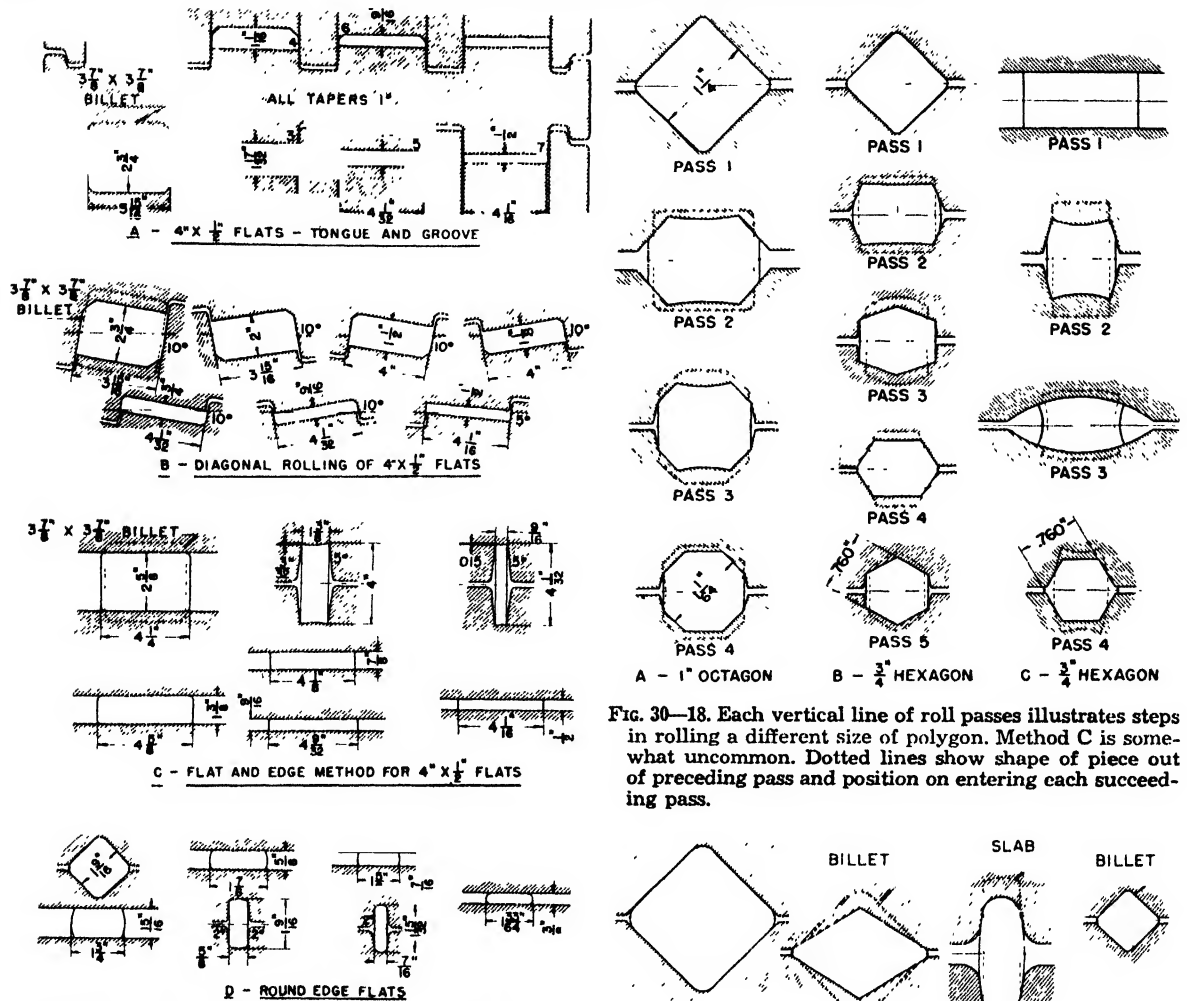


FIG. 30-17. Diagrams A, B, and C show, respectively, bar-mill roll-pass designs for rolling square-edge flats by the tongue-and-groove, diagonal, and flat-and-edge method. Diagram D shows a typical set of passes for rolling round-edge flats.

the next pass, the other two sides are free and the scale drops off here also. The next two passes repeat the same procedures. It is not practical to use this type of pass throughout the mill because box passes are not efficient in drafting in the smaller sizes, and for this reason the next two passes are diamond and square, a form of pass which effects substantial reduction in cross-sectional area and also has the merit of contacting all sides of the bar at once. In the next two passes, oval and square are used as it is desired to have a heavy reduction in order that passes Nos. 9 and 10 may be more lightly drafted. Pass No. 9 is a slabbing or flattening pass which reduces the 2 1/8-inch square to a rectangle measuring 2 1/8 by 1 1/2 inches. Pass No. 10 is called a "former" pass. The function of this pass is to round the upper and lower edges so that the following oval will be well-rounded on the sides, for when the oval is rounded at the roll parting, scale will not readily adhere to it at this point. When scale is present, it is rolled into the finished round. In some mills, a square is used to enter the round oval and, when so used on some grades of alloy steel, scale sticks to the bar at the roll parting and is rolled into the bar.

Figure 30-17 shows three roll-pass designs for rolling

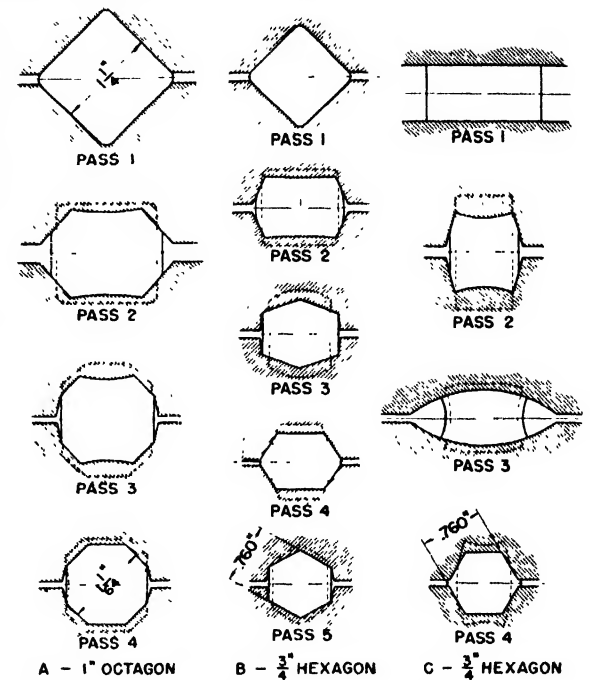


FIG. 30-18. Each vertical line of roll passes illustrates steps in rolling a different size of polygon. Method C is somewhat uncommon. Dotted lines show shape of piece out of preceding pass and position on entering each succeeding pass.

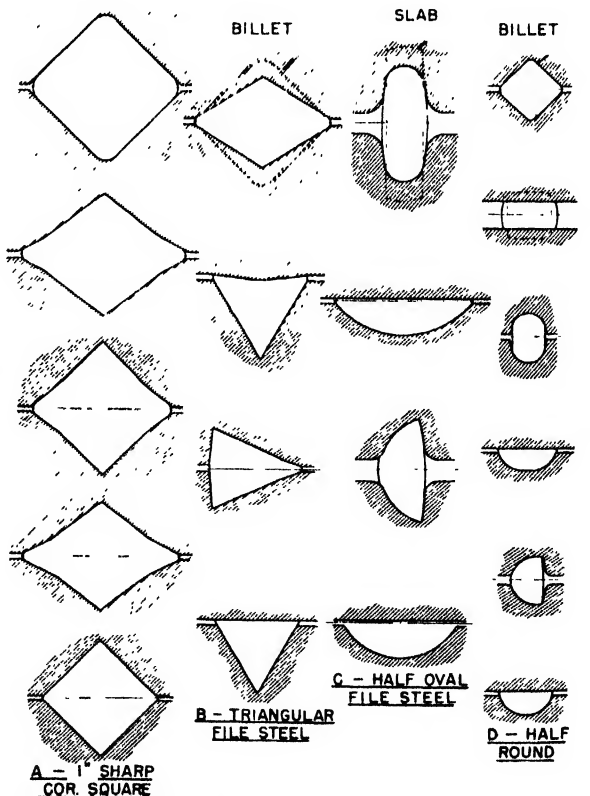


FIG. 30-19. Steps in rolling (A) one-inch sharp-cornered squares, (B) triangular file steel, (C) half-oval file steel, and (D) half-rounds on a merchant bar mill.

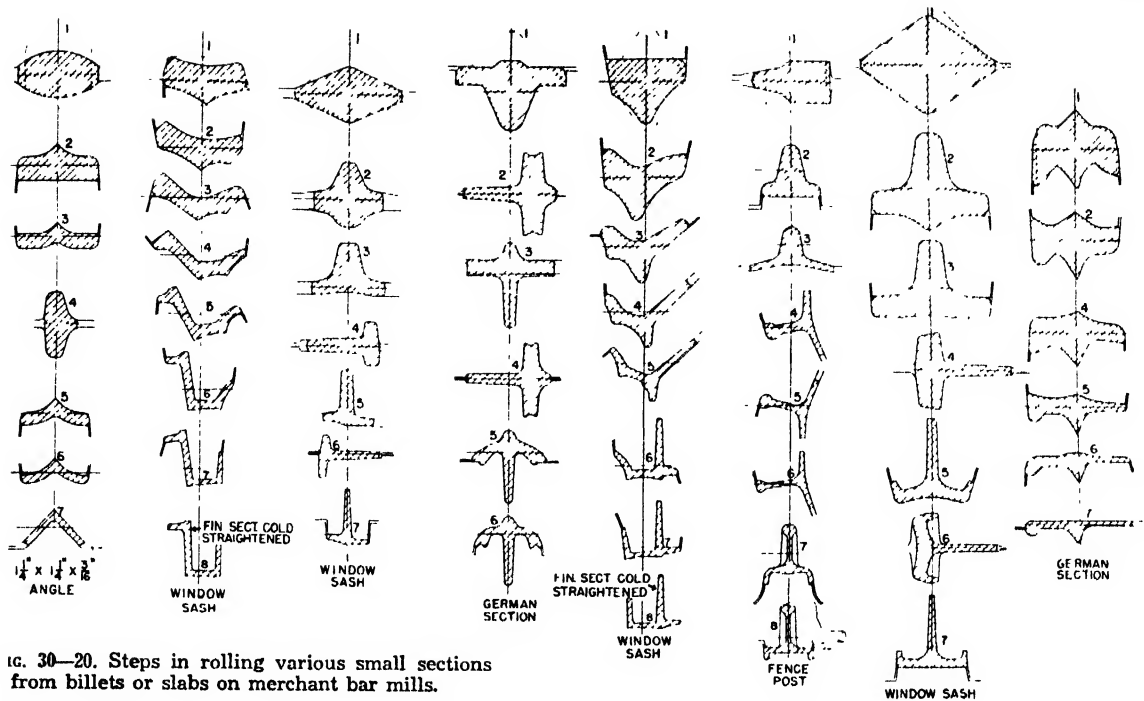


FIG. 30—20. Steps in rolling various small sections from billets or slabs on merchant bar mills.

square-edge flats, and one design for round-edge flats, as used on bar mills. Figure 30—17A illustrates a tongue-and-groove design used on a simple four-stand old-style bar mill. Figure 30—17B shows a diagonal design used on some mills, while Figure 30—17C shows the most popular design, one used on many modern bar mills, consisting of a flat and edge layout wherein the widths of the flats are regulated by edging grooves

which at the same time give the section a square edge. This design permits the rolling of several sizes in the same grooves. Round-edge flats are commonly rolled as shown in Figure 30—17D.

Octagons are often produced as illustrated in Figure 30—18A, but the most popular design for hexagons is shown in Figure 30—18B. Figure 30—18C shows an uncommon hexagon design used in one mill.

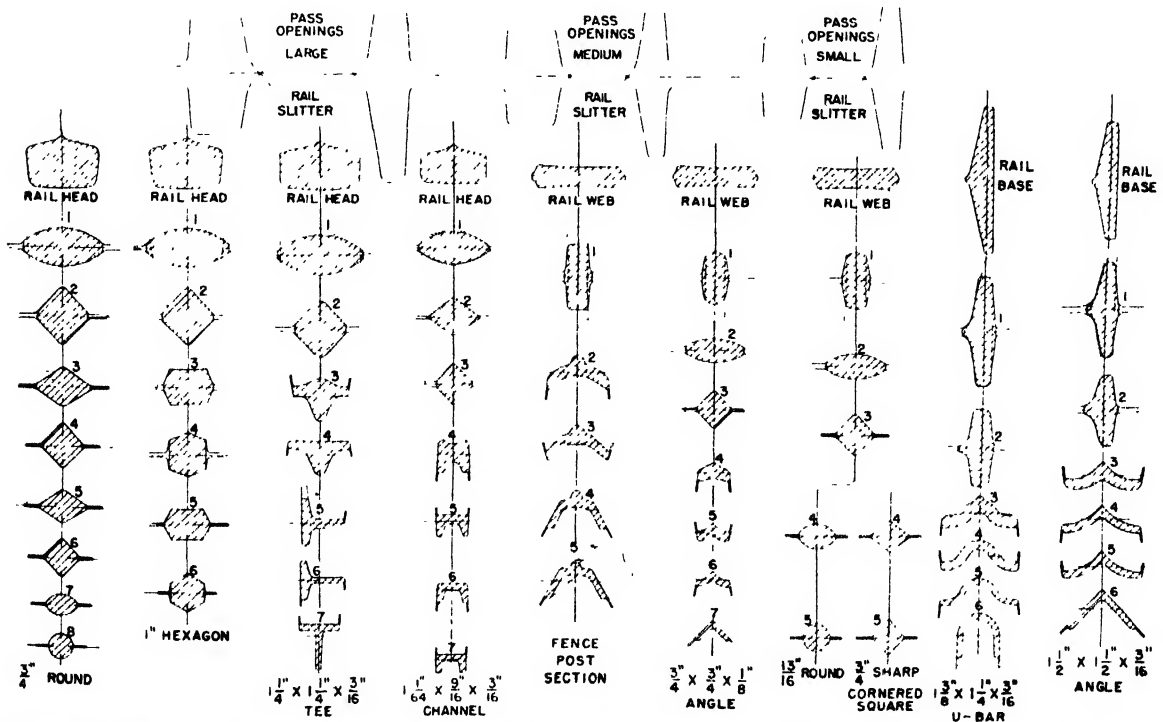


FIG. 30—21. Roll passes for producing various bars and shapes on a rail-slitting mill such as shown in Figure 30—10.

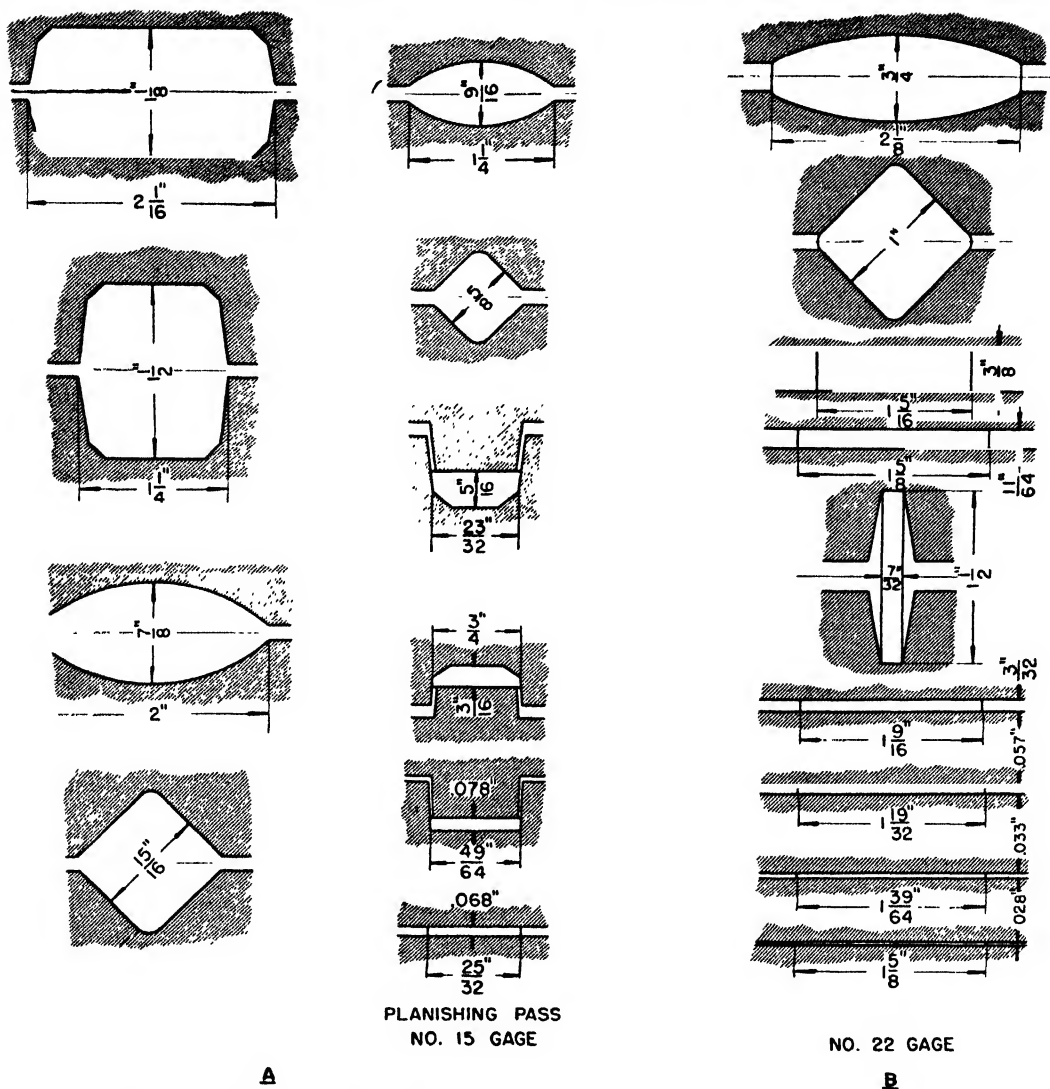


FIG. 30—22. Roll passes for producing hoop and cotton tie on a merchant bar mill.

Figure 30—19A shows a 1-inch sharp-cornered square design and Figure 30—19B a design used for rolling triangular file steel. Figure 30—19C illustrates a half-oval design also used for file steel and Figure 30—19D an arrangement for rolling half-rounds.

On many merchant bar mills, small shapes are rolled, such as angles, channels, small beams, tees, small agricultural shapes, window sash, etc. Figure 30—20 shows a number of small sections and steps in their formation from the billet or slab.

Rail-slitting mills, such as shown in Figure 30—10, roll a variety of bars and shapes. These mills, of course, must confine their product to material which can be rolled from steel having the chemical composition of

rail steel. Figure 30—21 shows a number of typical bars and shapes produced from the various parts of a rail.

Hoop and cotton-tie passes on the older mills were made by the tongue-and-groove method. The tongue-and-groove passes restricted the spread of the bar somewhat and also regulated the width. A series of passes using tongue-and-groove in the next to the last three passes is shown in Figure 30—22A. A later design is shown in Figure 30—22B. In this design the flattening is done in plain rolls and the width is regulated by a vertical edging stand. These passes represent the design used in the first continuous hoop and cotton-tie mill of 1895. The passes used in the most modern mills are of this same general design.

SECTION 2

FINISHING AND SHIPPING MERCHANT MILL BAR PRODUCTS

Coordination of Finishing and Shipping Functions
—The operations involved in finishing and shipping merchant bar products are essentially the same in all

plants, and the coordination of these functions differs only as to location of the rolling mill or mills, the type of products, arrangement of equipment, mode of trans-

Fig. 30—23. Slow-speed saw for recutting steel bars.



portation, etc. The functions of finishing and shipping merchant bar products are: (1) To provide a method of identification, cutting, bundling or coiling the product at the rolling mill as a preliminary to its delivery to the finishing departments. (2) To straighten, surface condition, heat treat, inspect or otherwise prepare the product for shipment in the finishing departments, in accordance with specifications and in compliance with transportation regulations.

BAR FINISHING PROCEDURES AND EQUIPMENT

Relation of Mill Delivery Equipment to Subsequent Finishing Operations—The type of rolling mill delivery equipment used has a very significant bearing on the straightness of the bar delivered for shearing. Most modern mills have certain features in the design of the hot beds—straight edges, kick-off equipment, roller tables and guides—to minimize bending and kinking that result in the necessity for subsequent straightening. When rolling-mill design permits, straightening units should be installed in the hot-bed delivery roll train ahead of the shears. Such an installation generally eliminates straightening as a finishing operation and necessitates fewer material handling steps prior to loading.

METHODS OF CUTTING PRODUCT TO LENGTH

The cutting of merchant bar product in most plants is performed by one of the following methods or by combinations of these methods: (1) Cutting by mill shears, directly to ordered lengths if practicable, or in multiple lengths if necessary to facilitate mill delivery, the latter requiring subsequent recutting to a variety of shorter lengths in the finishing departments. (2) Recutting by a finishing-department shear to meet ordered requirements, or for salvaging of portions of the product which may have surface or end defects.

(3) Hot sawing of large rounds or sections which are not adaptable to hot shearing. (4) High-speed friction

sawing where recutting is necessary after the straightening operation on a product previously cut by a mill shear or hot saw. (5) Machine cutting, which consists of the use of an abrasive cut-off machine, a hack saw, a cracker shear, or a slow-speed saw (Figure 30—23),

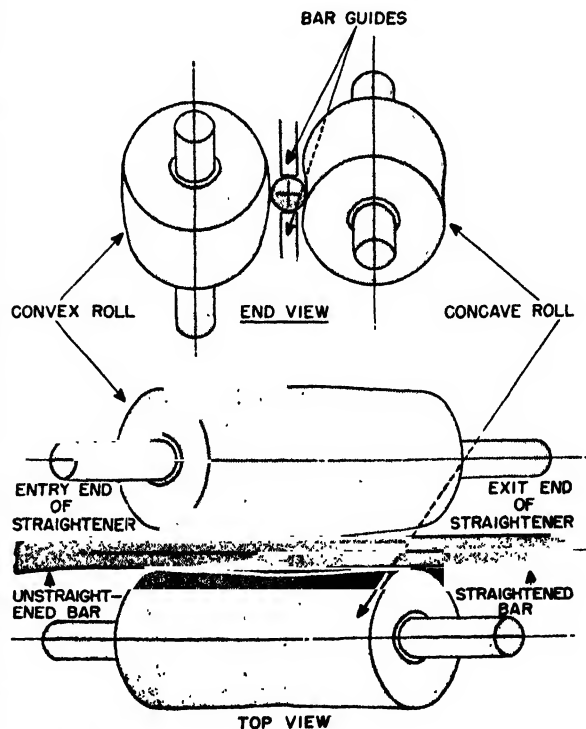


Fig. 30—24. Sketch showing end and top views of a typical two-roll unit for straightening rounds.

where the closest tolerance on end-squareness is required, or where, by the nature of the product, recutting cannot be done satisfactorily by other methods.

(6) Flame cutting, where the nature of the product and specifications permit the application of heat for rough cutting.

MACHINE STRAIGHTENING

Many merchant bar products require straightening after hot shearing at the mill. Straightening is necessary so that the product may meet either standard commercial tolerance specifications or any special specifications. In the finishing departments this is usually accomplished by roll-type machines, or by gag presses. The selection of the type of straightening equipment depends on the characteristics of the product and the end-use requirements. There are numerous commercial types of straightening machines, but they all are similar in principle and are fundamentally designed to process a specific shape within certain dimensional limitations. On this basis, straightening equipment can be divided into three main classifications:

(1) **For flat product.** This equipment includes multi-roll units for flats, squares, hexagons, etc.; and two-way roll units for straightening in planes at right angles in one pass.

(2) **For round products.** This equipment includes roll units which rotate the product through either a two-roll (cross-roll) or a multi-roll unit, and gag presses. Figure 30—24 illustrates a two-roll straightening machine.

(3) **For standard shapes and special sections.** This equipment includes both light and heavy multiple-roll units having grooves to fit each separate section, gag presses (both vertical and horizontal, with interchangeable guides and adjustable stroke), and roll units for shaping such products as small channels or U-bars which are hot rolled in an open form and then closed by cold forming before the straightening operation.

SIZING, TURNING AND CENTERLESS GRINDING

In the manufacture of merchant bar products, the design and type of rolling mill or the characteristics of the steel frequently preclude the possibility of hot-rolling a round to within precise sectional or out-of-round limitations, or to roll a round with a surface suitable for later fabrication requirements. When such requirements must be met, rounds are further processed in the finishing departments by such operations as sizing, turning, centerless grinding, or by combinations of the latter two. Bars to be processed by any one of these methods are hot rolled over-size by an amount predetermined by experience.

Sizing is applied to bars when a moderate reduction in section or an improvement in out-of-roundness is desired. It is performed by passing the bar through a two-roll or cross-roll straightening machine in one or more passes as required. Sizing can impart a cold-drawn appearance to the surface of the bar if the rolls are new and properly adjusted; however, it is not generally used for the purpose of improving the surface appearance. As sizing elongates the bar, due to reduction of cross-sectional area, re-cutting to length must follow.

Turning improves the surface of a bar by the removing of undesirable defects that may be present in the surface. It is applied on those types of steel, predominantly the alloy type and particularly the stainless steel group, where, due to the inherent characteristics of the product, it is very difficult and in some cases impossible to obtain a hot-rolled surface which can be utilized by the customer without the removal of a portion of the "skin" of the bar. Turning is accomplished by passing

the bar through a turning machine or lathe, using one or more passes depending upon the amount of material to be removed. There are several designs of turning machines used in producing merchant bar products, but they all are constructed according to a fundamental principle. They differ only in the arrangement of the revolving heads or the manner in which the bars are supported and fed to the revolving heads. In all types, a suitable cutting oil is fed to the tool or tools in the revolving head by a pressure system. In one type of machine, two cutting heads are mounted very close together, in tandem, between two guide bearings. In another type, the revolving heads are a considerable distance apart, separated by guide bearings. Due to the arrangement of the revolving cutting heads, the former type is more adaptable for bars of short length, while the latter is more suitable for bars of long length. Turning machines are made in various sizes and can handle rounds in sizes up to and including six inches and in lengths up to forty feet. The amount of metal removed per pass is dependent on the type of steel, and the diameter and the length of the bar. Removal of approximately $\frac{1}{8}$ inch on the diameter per pass is considered normal for bars of medium or large diameter. Speeds of twelve inches to fifteen inches per minute are normal when removing this amount of metal.

In finishing some grades of alloy and stainless steels, surfaces and dimensions are required that can be met only by **centerless grinding**. Centerless grinding differs from turning in that a grinding wheel is used for removal of metal instead of a cutting tool, and more accurate dimensions and better surface finish are obtained. The centerless grinding machine is constructed so that the bar is supported under the greater portion of its length as well as under the grinding wheel, rather than at the ends. This design enables the machine to operate to close tolerances by elimination of axial thrust, always present when a bar is supported at the ends. A suitable coolant is used on the grinding wheel or wheels. Most commercial centerless grinding machines are constructed to handle round bars up to four inches in diameter, in ranges of approximately one inch in diameter. When the diameter of the bar must be reduced materially, it is more economical to remove a large portion of unwanted metal by the turning machine before centerless grinding. Standard centerless grinding machine tolerances are ± 0.002 inches on bar sizes up to two inches in diameter and ± 0.003 inches on larger sizes up to four inches in diameter.

In some cases a highly polished bar is desired rather than the standard centerless ground finish. The polished finish is produced by passing centerless ground bars through a polishing or lapping machine. This machine is similar to the standard centerless grinding machine except that the grinding wheels are composed of a fine grit and the bar-supporting apparatus is constructed to operate within very narrow limits of accuracy. Polishing or lapping tolerances are one-half of those for standard centerless grinding. In precise polishing or lapping of long bars, it is very difficult to keep within these tolerance limits because of the wear of the grinding wheel and the spring of the metal. For these reasons, more precise tolerance limits than those stated above are not considered practical for commercial products.

Processing Bar Coils—There are some manufacturers, such as bolt, rivet or cold-drawn steel fabricators, who find it more convenient and economical to obtain bars (either rounds, squares, hexagons, or flats) in coil form rather than in straight lengths. As most of the smaller bar and rod mills are not equipped with flying shears to crop the front and back ends of a coil, it is necessary to

complete this operation in the finishing department of the mill. Variations in the process for cropping coils, depending upon available equipment, will be found at the various plants. The most widely used equipment is a long conveyor, sometimes over 1,000 feet in length, equipped with hooks spaced approximately ten feet apart, which pick up the coils from the mill coiler and carry them to the end of the conveyor. While the coils are moving along the conveyor, the ends are cropped either by a small shear or by flame-cutting. As the coils proceed further, they cool considerably, and then can be inspected and wired or banded securely. A shipping tag is attached, following which each coil is spark-tested as a final check of identification. By the time the coils have reached the end of the conveyor, they are cool enough to be loaded into the railroad cars for shipment. In some plants having high speed mills, where cutting to length would seriously retard output of the mills, bars which finally are to be shipped in straight lengths first are rolled in coils; subsequently, in the finishing department, they are then uncoiled, straightened, and sheared to length on a processing line designed for the purpose.

PICKLING

Stationary or Vat Pickling—Pickling is the term given the descaling process by which the hard black oxide formed on the surface of a bar during hot rolling is removed by chemical action. The removal of hot-rolled scale by pickling may be performed in order to: (1) prepare the surface of the bar for inspection, (2) prepare the product for ultimate end use. Pickling is also used, though to a much lesser degree, to remove the slow-forming red rust that develops on bar products after long exposure to air. Stationary or vat pickling in its simplest form consists of immersing the steel bars in a dilute acid bath, which is held at a predetermined temperature, and permitting them to remain stationary until the pickling action is completed. Most modern installations have improved the method by including means of keeping the bath in motion or agitation. Stationary or vat pickling is one of a number of pickling methods which are classified according to the manner in which the operations are conducted. Other methods of pickling and equipment are described elsewhere in the text.

The stationary or vat pickling method for descaling bars has been used with success and reasonable economy where the best established procedures are followed. A number of variables, both chemical and physical, determine the procedure to be followed, the design and capacities of equipment, and the lay-out to be used in the construction of a unit. Beginning about 1935, considerable data have been published concerning the pickling process, many of these being more or less empirical. Study of these data indicates that there is considerable difference of opinion as to what actually takes place as the iron oxide is removed. One theory proposes that the scale or oxide layer is relatively less soluble than the iron underneath and that the acid solution, passing through tiny cracks or fissures in the scale (formed by differences between the cooling rates of the metal and scale), reacts chemically with the bonding structure between the underside of the scale and the true metallic surface of the bar. In this reaction hydrogen gas is liberated which, it is claimed, dislodges the scale particles. A second theory proposes that the acid solution attacks the iron oxide or scale, resulting in its actual dissolution in the pickling bath. A third theory, a combination of the first and second, proposes that there is a preferential dissolution of the surface of the scale by the acid, followed by the penetration of the acid solution through tiny cracks or holes to the base metal,

where it attacks the bonding structure and liberates hydrogen gas. The hydrogen gas, as in the first theory, is said to serve as an agent in lifting the scale in small particles from the surface of the bar.

A large majority of the various types of steel bars can be pickled by the stationary or vat method, providing all factors pertaining to the physical and chemical variables of the process are taken into consideration. The efficiency of scale removal by pickling varies considerably. The scale on some types of steel bars can be removed readily, while on others, considerable difficulty is encountered. The pickling of hot-rolled low-carbon steels, for example, is done with little difficulty and the scale can be removed readily by the use of a sulphuric-acid solution. On hot-rolled high-carbon steels or alloy steels, however, many difficulties are encountered in the removal of scale. On high-carbon steels, a bad discoloration in the form of a black smudge of carbonaceous, insoluble material is the chief difficulty. The black smudge is particularly undesirable when close inspection must follow. On alloy steels, the hot-rolled scale is not removed as uniformly as in the case of the low-carbon steels. This condition can be attributed to the differences in chemical composition of the scale. By proper selection of acid or acids, by the selection of the most effective acid concentration, by adequate agitation, by use of proper bath temperature and the inclusion of an effective inhibitor, the pickling of high-carbon and alloy steel bars can be accomplished successfully. On the very high-alloy steels, such as the stainless group, pickling cannot be accomplished successfully in sulphuric-acid solutions alone, even at high acid concentration and elevated bath temperatures. Stainless steels are pickled in baths containing sulphuric plus another acid, another acid alone, or by a combination of other acids. The choice of the acid combination and concentration depends upon the characteristics of the steel to be pickled and the finish desired. The most commonly used acid for pickling is sulphuric. Other acids used in this method are hydrochloric (muriatic), nitric, and phosphoric.

Another factor affecting scale removal on hot-rolled high-carbon or alloy steels is the thermal treatment, such as normalizing or annealing, to which the bars may have been subjected. Bars so treated are difficult to pickle and, in some cases, the attendant pitting of the product is so severe that other methods of removing the scale must be employed.

The equipment for stationary or vat pickling consists of three tanks. The first tank is used for the dilute acid solution or pickling bath; the second contains only water, for washing the steel bars after immersion in the pickling bath; the third tank contains an alkaline liquid (lime water), for neutralizing any acid which remains on the bars after washing.

Tanks or vats containing the dilute acid solution or pickling bath can be of several types and are constructed of various materials. They must be water tight and capable of resisting attack by the acids used in the pickling solution. They are constructed either of wood, concrete, brick, or metal lined with wood. When wood is used for the tank or when it is used as a lining, cypress has been found to be the most durable. Recently, improvements in tank construction have been realized by using acid-resistant brick or terra-cotta tile as a lining for either wood, concrete or metal tanks. The acid-resistant brick or tile is laid in courses, with an acid-resisting mastic cement as a binder. Rubber is also used for lining tanks and has given excellent service. The practice of lining stationary or vat pickling tanks with sheets of lead is no longer looked upon with favor, because experience over

the years has indicated that the lead lining is susceptible to cracking under the rapid temperature changes which are prevalent in this method of pickling.

In addition to the three tanks required for the pickling operation, bar-mill finishing-department equipment usually includes a steel tank, containing a rust-preventative oil into which the bars are dipped in order to provide adequate protection against subsequent rusting.

As a coating of lime may be specified on bars, particularly those to be cold drawn, an additional tank for this purpose is generally part of the equipment. The lime coating serves as a lubricant in the cold-drawing process and reduces wear on the dies. It is applied after the washing and neutralizing stages of the pickling process. The tank for lime coating generally is constructed of steel and, as the lime should be kept in suspension in the water, a means of providing continuous agitation is usually a part of the installation. Pickling, washing, neutralizing, and liming tanks should be equipped with proper means of heating the bath and provision must be made for proper means of disposing of the waste solutions.

The efficiency of a stationary or vat pickling unit depends in a large measure upon the manner in which a multiplicity of similar pieces can be arranged and immersed in the pickling solution or in the washing, neutralizing, oiling or liming baths. The entire surface of each individual bar must be exposed if the best results are to be obtained. For this reason, various designs of crates and racks have been developed, one of which is shown in Figure 30—25. The steel bar product to be pickled is placed in tiers on the crates or racks. In the tiers, the bars are kept apart by separators which are called combs. These combs are designed to permit adequate spacing between each bar to allow for free circulation of the acid solution around them. The racking or placing of the steel bars on the fixtures for pickling is done manually. The materials used in the construction of the crates, racks, separators, and combs must be of acid-resisting metal and of a strength sufficient to carry the weight of the load.

When conditions and specifications permit, steel bar

product that has been coiled is pickled by passing a chain through the open center of a group of coils, and attaching the ends of the chain to the hook of an overhead crane. Such a group of coils is referred to as a "lift," and is suspended in the pickling tank by the crane. This practice is referred to as chain pickling. Chains used in this work also must be made of acid-resisting metal. Monel metal or aluminum-bronze alloys are used in many parts and fixtures, including the chains.

Steam has been the most widely used for heating pickling baths, but, the submerged gas burner method, a relatively recent development, has become popular with many operators. Where steam is used, suitable piping for conducting the steam into the bath is necessary. Adequate and convenient steam-input controls must be provided in the form of valves, either hand operated or automatic in action, in order to maintain the bath as nearly as possible at the proper temperature. When steam is introduced directly into the bath, care must be taken to avoid direct impingement against the sides or bottom as this will result in rapid erosion and damage to the tank. Also, the introduction of steam directly into the pickling bath results in an increasing dilution of the bath, which requires the addition of more acid to maintain the solution at the proper concentration.

The submerged gas burner provides for burning gas and air under automatic control in submerged burners. The hot waste gases from the burners are conducted into lead tubes which run the length of the tank on the bottom. This method heats the pickling bath quite efficiently, and the waste gases, which are forced out under pressure through holes located throughout the length of the tubes, provide the desired agitation of the bath. When this type of heating equipment is used, however, care must be taken to avoid accumulation of sludge and scale in the bottom of the tank. If the lead tubes become completely covered with sludge and scale, they are shielded from the pickling solution in the tank, become overheated, and are destroyed.

When a new pickling bath is to be prepared, the tank first is filled to approximately three-fourths capacity with water. Acid in an amount to provide the proper concentration is run into the tank, and then enough

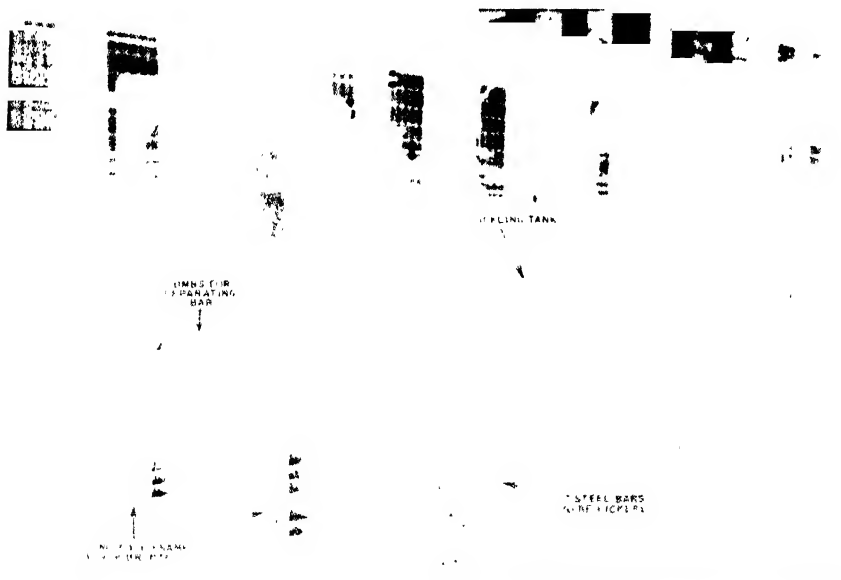


FIG. 30—25. Loaded pickling rack, showing how combs keep bars separated to allow free circulation of acid solution.

water is added to bring the solution to an operating level. The heat is then turned on and the bath is brought to the operating temperature. When placing products in the pickling bath, care must be taken to avoid any rapid or irregular movement that may splash acid solution over the sides of the tank, where it rapidly attacks the outer tank shell, concrete floors and tank foundations. It is very important that wash tanks be emptied frequently, as acid in damaging amounts builds up in the wash water and interferes with the proper function of the washing operation.

The surface of steel bars to be pickled must be free from oil or grease, as these materials protect the surface from the action of pickling solution. Any substance on the bars which may serve to contaminate or neutralize the acid solution must be removed before they are immersed in the pickling bath. After the steel bar product is washed and dipped into the neutralizing bath, it is further safeguarded against rusting by being thoroughly dried. Live steam and sometimes air, blown against the bars as they hang suspended in the rack, are used for this purpose.

Summarizing, the factors controlling the rate of pickling and the iron loss are: (1) acid concentration, (2) temperature, (3) time in bath, (4) percent of iron (ferrous) sulphate in the bath, (5) presence of inhibitors, and (6) agitation of the bath. Frequent tests of the bath should be made. Not only will the testing of the pickling solution reveal the concentration of the acid bath, but will also show the rate of iron dissolution into the bath, thus giving a good indication of the efficiency of the pickling operation. If the rate of iron increase in the solution is rapid, it indicates that the true metal is being attacked severely. Under such operation, not only is the wastage of acid high, but also the loss in weight of the steel bars becomes excessive. When the iron content of the pickling solution reaches approximately 0.5 lb. per gallon, the bath is considered to have little further use and is ready to be dumped. Testing the pickling solution for acid and iron can be done very readily by titration.

Temperature of the Pickling Bath—The significance of this factor should be understood clearly. By raising the temperature of the pickling bath the action of the acid solution upon the scale can be increased greatly, for the particular pickling job at hand. It should be remembered, however, that while increasing the temperature of the acid solution speeds up the pickling action on the scale, it also increases the tendency of the acid solution to attack the steel itself. When the metal is attacked, the iron reacts with the acid and goes into solution in the form of the salt of the particular acid or acids used. As the amount of iron salts in solution increase toward the point of saturation, the efficiency of the process decreases very rapidly. This may necessitate the addition of more acid to the bath or, possibly, the complete replacement of the bath with a new acid solution. The trend with some operators is toward the use of lower acid concentrations and lower bath temperatures. The practice requires more time per ton of steel pickled, but the quality of the work is improved. Less metal is lost from the surface of the bar and less acid is consumed in the process.

An inhibitor is an agent added to the pickling solution for the purpose of protecting the exposed surface of the metal of the bars, by inhibiting or retarding acid attack upon the metal without affecting, to any appreciable degree, the pickling action which removes the scale. The inhibiting action is not understood clearly but is generally explained by the electrolytic theory of corrosion. Inhibitors show little change and lose little of

their efficiency during the pickling operation. However, they may be broken down and their function destroyed by overheating. Many different substances are used as inhibitors in acid pickling. They range from vegetable or animal matter to complex synthetic organic chemicals. The prime requirements of an inhibitor are that it must disperse colloiddally in the bath, prevent hydrogen evolution and not leave a smudge or film on the surface of the steel. Many inhibitors contain substances that cause foaming, and the floating layer or blanket of foam on the bath prevents the escape of acid with escaping gases and vapors. As inhibitors are expensive and only a small amount is needed for desired results, less costly foaming substances are added separately by many operators when increased foaming activity is desired.

Common difficulties encountered in pickling are over-pickling, under-pickling, smudge and pitting. Over-pickling may be defined as the etched appearance of the surface of a product caused by over-activity of the acid solution. Conversely, under-pickling may be defined as incomplete removal of the scale due to limited activity of the acid solution and/or the use of too low a pickling temperature. Smudge is a carbonaceous precipitate or stain which forms on bars of high carbon content. Pitting may be defined as the appearance of crater-like indentations on the surface of the bar, which may result from over-activity of the acid solution on the metal in areas where scale has been loosened mechanically or removed prior to the pickling operation, or as the result of an electrolytic action taking place in areas where there are concentrations of dense scale on the surface.

GRIT-BLASTING

Grit-blasting or blast-cleaning is a mechanical process used for removing scale and rust on merchant bar products. It consists of eroding or abrading away the scale from the surface of the bar by impinging an abrasive substance like sand or aluminum oxide, or a metallic substance like cast-iron shot. The abrasive material may be directed against the work by air under pressure or by a mechanical apparatus utilizing centrifugal force. All types of steel bar products and shapes can be cleaned successfully by this method. As a result of the difference in economy of operation between the grit-blasting and the pickling method, the former generally is confined to the cleaning of one of the following types of products: (1) those which must have certain physical characteristics essential to subsequent processing; (2) those which cannot be cleaned satisfactorily by the pickling method after such thermal treatments as normalizing or annealing; (3) those with high alloy content whose scale cannot be removed satisfactorily by ordinary pickling methods.

The principal difference between the various types of grit-blasting equipment is in the means employed for throwing the grit or shot against the work. One type of grit-blasting machine makes use of centrifugal force generated by a rotor or impeller which rotates at a very high speed. Grit in the form of metal shot is fed from a hopper into the revolving rotor through an opening in the center of the rotor housing. The shot, on entering the revolving rotor, is picked up by the rotor vanes and is thrown by centrifugal force away from the center of the rotor toward its periphery. From the periphery, the shot is directed outward to impinge on the work at the angle desired for the best abrasive effect. Since this type of machine develops its abrasive characteristics on the basis of the velocity and the weight of the particles, lighter grit, such as sand, cannot be used satisfactorily.

Another type of machine employs compressed air or forced air from a blower for throwing the grit particles. There are several modifications of this principle, but in

each type the grit is permitted to entrain in a fast-moving air stream and is directed upon the surface of the work through a hose having a nozzle designed for the purpose. The lighter grit is used most effectively with this type of equipment. The material used for, and the size of, the grit or shot in blast cleaning are dictated by the requirements for a given job and, as previously noted, by the type of equipment in which it is to be used.

From the standpoint of density, grit can be divided into: (1) the light, and (2) the heavy. Light grit is a non-metallic inorganic material with excellent abrasive characteristics. It is purchased according to particle size. Examples of light grit are the widely used natural abrasive known as Ottawa sand and the synthetic abrasive, aluminum oxide. Heavy grit is principally of the metallic type such as cast-iron shot. Like the light grit, it is purchased according to particle size. Metallic grit is the type most generally used on merchant bar product.

The chief advantages of grit blasting are: (1) it leaves the surface of the bar with a bright metallic finish without any adhering scale; (2) it is capable of cleaning a number of types of products that cannot be cleaned successfully by pickling; (3) it does not produce such physical or chemical conditions as over-pickling, under-pickling, smudge or pitting which may attend the pickling process. The disadvantages of grit blasting are: (1) only a few bars can be processed at one time in the blast machine; (2) as the grit or shot builds up with scale the efficiency of the cleaning effect is decreased; (3) the bars are difficult to inspect for surface defects because of the peening action of the grit or shot; and (4) the cost per ton of steel cleaned is greater than the cost of pickling.

Grit blasting does produce minute indentations on the surface of the product which are more beneficial in some subsequent fabrication operations than they are detrimental. The size of these indentations and their shape serve to produce a surface sheen or finish determined somewhat by the particle shape and size of the grit. The character of this sheen or finish is very important where cold-drawing operations are to be performed. As a grit-blasted steel surface is very susceptible to rusting, bars cleaned by this method must be protected immediately from moisture and alkaline or acid vapors. Also, when a bar is grit-blasted with cast-iron shot, particles of the shot have a tendency to adhere to the product and will cause rapid rusting or corrosion on some types of steel if they are not removed immediately. In such cases the grit-blasting operation must be followed by a light pickling operation.

BAR INSPECTION AND TESTING

Inspection—In order to produce satisfactory bar products it is important that an adequate quality control system be established and maintained. This system should include mechanical and metallurgical testing as required, and the inspection for surface or other defects. The procedures for making mechanical and metallurgical tests and the inspection of steel prior to rolling in the merchant mills are covered in other chapters of this book. The inspection of merchant bar product in the majority of plants is carried out along the same general lines and can be divided into two divisions; (1) mill inspection, and (2) finished product or final inspection. The duties and responsibilities of the inspection forces in these divisions must be thoroughly co-ordinated to accomplish the best results.

Mill inspection is performed during the rolling process and is the means of minimizing or preventing discrepancies at the source. The duties of the inspector at the mill are: (1) to check the identity of the steel being

rolled, (2) to inspect the surfaces of the product to determine its suitability for further processing in the finishing departments or in its final intended use, (3) to check section and length of the product to determine its suitability for application on a particular order, and (4) to procure necessary samples for mechanical or metallurgical tests. The inspection of the finished bar products, on the basis of commercial standards or for dimensions, straightness, surface defects, and other items of form as may be required, is carried out upon completion of the finishing end operations at designated units.

The duties of the final inspector are in a measure a repetition of the duties of the mill inspector, with the additional responsibility of checking and posting heats. The checking and posting of heats is a method of reporting the amount of product accepted or rejected (per heat) on an order, with reasons for further processing or for rejections. The final inspector must lay particular emphasis on surface quality if the product has been conditioned and straightened. Various tools are required for inspecting merchant bar product. A list of these tools and a brief description of their use follows:

(1) micrometers and calipers, for measuring the section of rounds, flats, and squares; (2) snap gages, for precision rounds; (3) tape, for measuring length; (4) slide scales, for measuring width; (5) protractors, depth gages and radius gages, for shapes; and (6) the square tool for measuring off-squareness.

Surface defects, as well as other defects which cause the rejection of merchant bar products, may be the result of steelmaking practices which carry through from the ingot or may be caused by the rolling mill equipment used to produce the product. A list follows giving the general mill terminology of the most regularly occurring types of mill defects and a brief description of each:

1. **Fins and overfills** are protrusions formed when the section is too large for the pass it is entering, or when proper allowance has not been made for lateral spreading in the rolls. Overfills are broad and less sharp than fins. As a rule, overfills occur more frequently than fins and in many cases are associated on the same bar with underfills.
2. **Underfills** are the reverse of overfills, that is, they are areas in which the section is incompletely filled. They are formed by permitting the bar to be rolled scant in certain dimensions. Underfills appear most frequently on rounds and channels.
3. **Slivers** are loose or torn segments of steel rolled into the surface of the bar. They may be caused by a bar shearing against a guide or collar, incorrect entry into a closed pass, or a tear from other mechanical causes. Sometimes slivers are present in the billet and are carried through to the hot-rolled bar or shape.
4. **Laps** can be said to be a rolled-over condition caused by a bar having been given a pass through the rolls after a sharp overfill or fin has been formed, causing the protrusion to be rolled into the surface of the product.
5. **Seams** are crevices in the steel that have been closed but are not welded. They are a type of a defect very difficult to detect on certain types of steel products. Seams are caused by blow holes and cracks in the original ingot, or by faulty methods of rolling in both semi-finishing and finishing mills.
6. **Fire cracks and roll marks** are impressions in the product, of varying degree and pattern, caused by mill rolls becoming overheated, and cracking or spalling.

7. **Scratches** are long nicks or indentations in the product caused by the surface or surfaces of the bar rubbing against sharp or pointed objects such as guides on the mill, chutes, "dead" conveyor rolls, chain hoists or other mechanical equipment.
8. **Rolled-in scale** is a defect in the surface caused by scale, formed during a previous heating, which has failed to be eliminated during the rolling operations. It is one of the most prevalent surface defects.
9. **Buckle and kink** is a corrugated or wrinkled surface condition caused either by worn out pinions on a roll stand or uneven cooling beds. Buckle is an up-and-down wrinkle, whereas kink is a side wrinkle.
10. **Burned steel** appears as a rough area with checked or serrated edges. It is caused by steel being exposed to an excessive temperature and is always scrapped.
11. **Camber** is the deviation of the side edge of a bar from a straight line. It is caused by improper heating of the billet, uneven dimensions causing differential expansion or contraction, or improper alignment of the hot beds.
12. **Hook** is a short bend or curvature caused either by improperly adjusted delivery guides or by any obstruction which may halt momentarily the forward motion of the bar from one roll stand to another.
13. **Pipe** is a steel-making defect carried through from the ingot. The presence of pipe is detected as a small round cavity located in the center of an end surface.
14. **Shear distortion** is a mashed or deformed end on a bar caused by defective or improperly adjusted shearing equipment.
15. **Twist** is a condition wherein the ends of a bar have been forced to rotate in relatively opposite directions about its longitudinal axis. It may be caused by excessive draft, faulty setting of delivery guides, or lack of uniform temperature in the bar.

Testing—Numerous tests are made during the finishing operations, the purposes of which are to reveal defects otherwise impossible to detect during surface inspection. The tests most commonly employed are described as follows:

1. **The pickling test** consists of immersing short pieces of product for several minutes in dilute sulphuric acid. The acid removes the scale from the bar and exposes to view such surface defects as may be covered or hidden by the scale.
2. **The upset test** consists of heating small test pieces to a forging temperature and then subjecting them to severe compression under a hammer. The compression or upsetting action will force open any defects which could not be detected while the steel was in the as-rolled condition. A sound steel will be indicated by the absence of areas which open up.
3. **The magnaflux test** is used on certain types of steel for which there are very rigid surface requirements. It is carried out by placing the steel to be tested in a powerful magnetic field. A fine magnetic powder is then sprinkled over the surface. The particles of the powder seek out and cling to the local magnetic poles which may be developed by any sort of metal discontinuity at or slightly below the surface. Any irregularity in the metal surface will cause a distinct magnetic pattern to be set up, thereby disclosing any hidden defects.
4. **The file test** consists in removing the scale, by filing, on any surface area which may be suspected of

containing a hidden defect. The file test is a quick method employed by inspectors for determining the extent and depth of seams in bars.

5. **The bend test** is made on certain classes of material to determine the soundness of internal structure and to denote the degree of ductility. The test consists merely in bending a standard test specimen through a certain specified arc. Examination of the bend will disclose surface defects.
6. **The etch test** is one which is used repeatedly in standard manufacturing and fabricating processes to determine the soundness of internal structures. A test piece is cut from the desired location in a bar and from this test piece a specimen of the size required for etching is removed. The surface to be examined is polished and then dipped in a solution of hydrochloric, sulphuric, nitric or picric acid. For some products, solutions of iodine, copper ammonium sulphate, ferric chloride, cupric chloride or cupric sulphate are used. Directions for etching various products may be found in "Metals Handbook."¹
7. **The grit-blasting test** is used for much the same purpose as the pickling test. The grit removes the hot-rolled scale so that the surface of the test piece can be visibly inspected. It is not recommended for detecting fine seams on certain types of steel because of the peening action of the grit.

CONDITIONING METHODS AND EQUIPMENT

It has been noted that bars which are free from surface defects cannot always be produced. Some of these defects are common to all steel products, whereas others are more or less peculiar to merchant bar product. In order that the product may meet required quality standards, these defects must be removed by chipping or grinding. Their removal by either of these methods is known by the term "conditioning." Thorough inspection must precede conditioning. In a large number of cases pickling before inspection is necessary in order to reveal all the defects that may be present. The product requiring conditioning must be marked properly with a suitable chalk or crayon in the areas showing defects.

Conditioning is carried out by either chipping or grinding, depending upon the characteristics of the product. Conditioning by chipping is confined to soft and medium-hard steels and to those products on which subsequent fabrication procedure will permit the presence of grooves from chipping. Well-maintained pneumatic hammers operated by an adequate air supply, and chisels which have been properly heat-treated and dressed, are the principal equipment necessary for chipping. Conditioning by grinding is confined to those steels with a high hardness, or to bars or shapes whose contours would be changed by chipping grooves. The grinding operation is accomplished by: (1) pneumatic grinders, or (2) high cycle electric grinders. The choice of grinder depends upon local conditions such as air and power supply, or upon the suitability of the machine to perform the required task. Each type of grinder possesses certain advantages and disadvantages. The pneumatic grinder has the advantage of being lighter in weight and has a minimum of moving parts, which facilitates repairs. Its major disadvantage is its inability to maintain full speed on all loads. It also must have an adequate air supply (90 to 100 lb. per sq. in.) without a large drop in pressure from source to point of use. The high cycle electric grinder, because of its special con-

¹ Metals Handbook, 1948 Edition; American Society for Metals, Cleveland, Ohio; pages 384-399, inclusive.

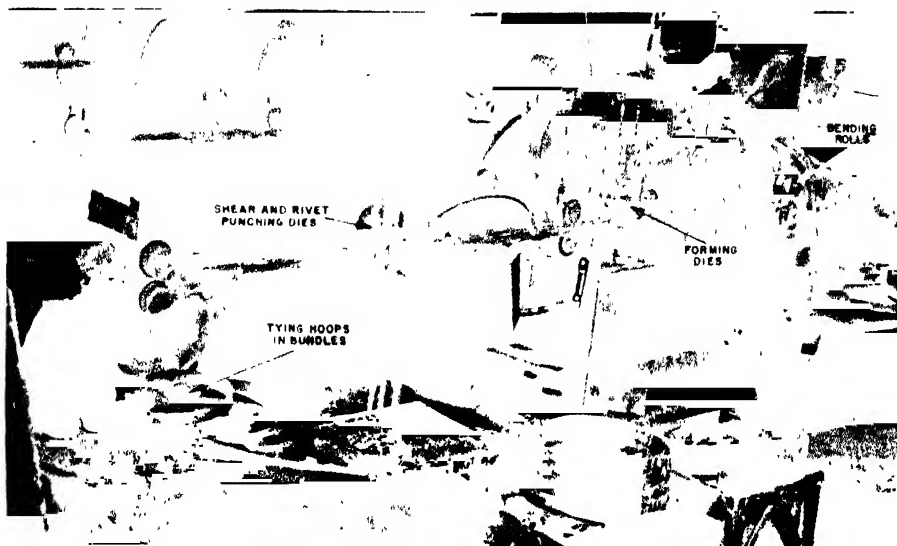


FIG. 30-26. Automatic hoop-forming machine in operation. Finished hoops are on stand in right foreground.

struction, has the advantage of being able to maintain full speed at all loads. It has the disadvantages of being somewhat heavier to handle, of having many complicated working parts, and of requiring a frequency changer adjacent to the work. The grit, bond, shape and speed of the grinding wheels are dictated by the characteristics of the steel and by the shape of the product being worked. Of the two types of conditioning, chipping is the most economical and, when conditions permit, is most generally used.

NARROW FLAT-ROLLED PRODUCTS

Band, Hoop and Cotton Tie—Band for use in commercial packaging is usually rolled in coil form in multiple width on the rolling mill, and then slit to narrow widths on multiple slitting units installed in the bar finishing departments. Edge conditioning is necessary to eliminate the sharpness of the slit edge, before final coiling and shipment. The usual sizes are $\frac{3}{4}$ by 0.028, $\frac{3}{4}$ by 0.035, $1\frac{1}{4}$ by 0.035, $1\frac{1}{4}$ by 0.050, and 2 by 0.050 inches. It is made from a steel in the 0.50 to 0.60 per cent carbon content range, with 0.80 per cent manganese and 0.15 to 0.30 per cent silicon.

Hoop—There are four general classifications of this type of product: (1) tight cooperage hoop for barrels to hold liquids, (2) slack barrel hoop for barrels to hold dry products, (3) tobacco barrel hoghead hoop, and (4) special hoop for special packages.

Hoop (except tight cooperage hoop) is made from steel in the 0.08 to 0.10 per cent carbon content range. Tight cooperage hoop is generally made from open-hearth steel having a carbon content of 0.30 to 0.35 per cent carbon, but may also be produced from Bessemer steel having a carbon content of 0.19 to 0.24 per cent.

Hoop is made either by slitting coiled strip, rolled in multiple width, into narrow coiled strip of the desired width; or, from narrow coiled strip with a hot-rolled or mill edge. The type and width of hoop being produced influences the choice of method used.

Hoop is produced in widths increasing in increments of $\frac{1}{8}$ of an inch, beginning with a minimum of $1\frac{1}{8}$ inches and extending to and including 2 inches. It is made in thicknesses between 0.027 and 0.042 inches. It is prepared in cut lengths for making hoops ranging from 2 feet $8\frac{1}{2}$ inches to 8 feet 6 inches in circumference. Automatic machines are used in the fabrication of hoop (Figure 30-26). These machines are so designed that

the strip from the coil, passing through a machine in a horizontal position, is first run through rolls in which a slight bend is made on the edge to be beaded. This is followed by a beading operation, done in forming dies that operate horizontally at approximately 400 strokes per minute. The beaded strip next moves into a combination shear and rivet-hole punching die, where it is sheared to a specified length, and where the rivet holes are punched. Hoop is produced as "curled hoop" or a "straight length." Curled hoop is made by a pinch-roll and curved guide-shoe arrangement that permits the hoop to take a circular form. A straight length hoop is produced merely by removing the curved guide shoe.

Cotton tie is a light, narrow, hot-rolled strip used, as the name implies, to bind bales of cotton, hemp, jute, etc. It is fabricated in a manner quite similar to hoop. After being finished in pinch rolls and a vibrator it is delivered onto apron conveyors. From the apron conveyors, the strip is coiled and delivered on a coil conveyor to cold shears where it is sheared to a length of approximately 12 feet. It is then bundled by hand at which time buckles are inserted. The bundles are then dipped into a tank containing an asphalt base paint if desired. Cotton tie is shipped in bundles consisting of 30 ties and 30 buckles, either painted or unpainted. Two sizes of bundles are produced; the standard, weighing about 45 lb., of $1\frac{1}{16}$ -inch wide and 0.042-inch thick (19 gage B.W.G.) cotton tie, and a special bundle, weighing approximately 60 lb., of $1\frac{1}{16}$ -inch wide and 0.049-inch thick (18 gage B.W.G.) cotton tie.

CONCRETE REINFORCING BAR

Concrete reinforcing bar is a merchant-bar product consisting of plain rounds and deformed rounds. It is used to furnish tensile strength to concrete sections subject to bending loads and to furnish additional compressive strength in sections where unreinforced concrete would prove too bulky.

All types of concrete structures are commonly reinforced with either deformed or plain bars. Concrete reinforcing bars are usually deformed. Deformed concrete reinforcing bars are bars in which the surface is provided with lugs or protrusions (called "deformations") which inhibit longitudinal movement of the bars relative to the surrounding concrete. The surface deformations (Figure 30-27) are hot formed in the final roll pass by passing the bars between rolls having



Fig. 30—27. Photograph of a deformed concrete-reinforcing bar, showing protrusions produced by rolling the bar between rolls having a pattern cut into them.

patterns cut into them so that the surfaces of the bars are forced into the depressions in the rolls to form characteristic deformations. Plain bars may be specified at the discretion of the designer and, when so specified, the standard sizes of plain bars are designated by their nominal diameter. Deformed concrete bars are produced in accordance with the specifications for minimum requirements for the deformations of deformed steel bars for concrete bars ASTM designation A305. Ten sizes of round bars have been adopted as standards as approved through the United States Department of Commerce, Simplified Practice Recommendation R26-50. The following table furnishes dimensional data on these bars:

Table 30—I. Dimensional Data, Standard A-305 Reinforcing Bars

Bar Sizes*		Nominal Dimensions— Round Sections			
Old Designation	New Numbers	Weight (Lbs. per Foot)	Diameter (Inches)	Cross-Sectional Area (Sq. Inches)	Perimeter (Inches)
1/4" Rd.	2	0.167	0.250	0.05	0.786
3/8" Rd.	3	0.376	0.375	0.11	1.178
1/2" Rd.	4	0.668	0.500	0.20	1.571
5/8" Rd.	5	1.043	0.625	0.31	1.963
3/4" Rd.	6	1.502	0.750	0.44	2.356
7/8" Rd.	7	2.044	0.875	0.60	2.749
1" Rd.	8	2.670	1.000	0.79	3.142
1" Sq.	9	3.400	1.128	1.00	3.544
1 1/8" Sq.	10	4.303	1.270	1.27	3.990
1 1/4" Sq.	11	5.313	1.410	1.56	4.430

* The new bar numbers are based on the number of 1/8 inches included in the nominal diameter of the bar. Bar Number 2 furnished in plain rounds only. Bars numbered 9, 10 and 11 are round bars and equivalent in weight and nominal cross-sectional area to the old type 1", 1 1/8", and 1 1/4" square bars. The above weights were adopted as standards by the Concrete Reinforcing Steel Institute in 1934, and supersede former practice. These weights have been approved through the United States Department of Commerce, Simplified Practice Recommendation R26-50.

Concrete reinforcing bars are produced to the Specifi-

cations for billet-steel bars for concrete reinforcement (ASTM designation A15), rail-steel bars for concrete reinforcement (ASTM designation A16), or axle-steel bars for concrete reinforcement (ASTM designation A-160). These classifications are largely self-explanatory and the concern here is with new billet steel rolled expressly for concrete reinforcing bars produced under ASTM Specification A15. Three grades of steel are produced under this specification and selection of structural, intermediate or hard grade is predicated on design specifications. The three grades differ only in physical requirements and may be produced from either open-hearth, electric-furnace or acid-Bessemer steel.

Concrete reinforcing bars are shipped from the mills in straight lengths either cut to design length in the mill shears or in long lengths to be re-cut for fabrication, as required in specific applications.

Engineers' and architects' designs and specifications are prepared in accordance with the Manual of Standard Practices of the Concrete Reinforcing Steel Institute. Bar fabricators furnish concrete reinforcing bars either straight and cut to the proper length or bent or curved in accordance with plans and specifications.

PACKAGING AND LOADING

The packaging of merchant bar products in most plants is performed according to: (1) standard practices, or (2) special practices as required by the customer. Standard practices are controlled by such factors as: (1) the weight of the bundle or lift, (2) the means of binding or fastening the bundles or lifts, (3) the means of identification, (4) the means of protecting the product in transit, and (5) the geographical location of the customer. Special practices are predicated upon the type or capacity of handling and processing equipment used by the customer.

Either wiring or banding may be applied for binding or fastening bundles or lifts, depending upon the shape, size and length of the product. Special practices for binding or fastening bundles or lifts are applied only when specified. Wiring is used generally on large rounds and heavy sections, whereas bands are used most generally for small sections or for flat products. Standards have been established which specify the spacing and the number of wires or bands to be used for various sizes of bundles. Additional wiring or banding, either for domestic or export shipment, is handled as special practice. Special practice also applies when stack piling is required, as the number of pieces and the dimensions of the bundle are limited as well as the weight. Wiring and banding is accomplished by such special tools as stretchers and sealers.

Identification of merchant bar products requires that each bundle or lift for domestic trade be identified by a standard manila tag, approximately three inches by five inches, attached to one end. Where coils are involved the same type of tag is used on each coil. The information on the identification tag will vary somewhat in different plants but it generally contains a part or all of the following information: customer's name, destination, customer's order number, mill order number, part number or special markings, number of pieces, section, length, weight, heat number, bundle or lift number, and the grade of steel. On products for export, additional identification is required on each bundle or lift in the form of duplicate manila tags on opposite ends and a metal tag attached to the center of the bundle. This is necessary because of the possible loss of a single manila identification tag when the product is rehandled at railroad terminals, shipping docks, or aboard ship. In addition to the standard manila tag, as mentioned above, the

identification of merchant bar products also requires the die stamping of heat numbers on the ends of bars of certain dimensions. These specifications are as follows: heat numbers are stamped on one end of all bars (round, squares or hexagons) of 2½ inches or above, and on all flats three inches by one inch and above. When special practices are involved, bars are stamped with codes or symbols as specified. When the product consigned for export is to be shipped loose, each piece is stenciled with the necessary information for proper identification.

Standard practice for color-marking for identification provides for painting one color (as prescribed by the customer) on one end of each bar on sizes of 1½ inches up to 2 inches and two colors (as prescribed by the customer) be painted on sizes 2 inches and over. In special practices, either for export or domestic trade, special marking is applied only when required by the customer.

Bundles or lifts are loaded in open cars without protection from the weather. When shipping carload lots, bars and bar shapes are shipped loose or in large lifts in open cars. When special practices are required, lifts may be wrapped or shrouded individually, or the entire carload may be shrouded as a unit. Loading, like packaging, is divided into (1) standard practice, or (2) special practices as required by the customer. The gondola car is the standard means of transit, either for magnet,

chain, or crane unloading. Standard practice for loading and bracing is conducted in accordance with the rules of the American Association of Railroads. Special practices as requested by the customer generally fall into one of the following categories: (1) modified methods of gondola-car loading for chain or crane unloading, or magnet unloading alone; (2) flat-car loading for chain or crane unloading, magnet unloading, or tractor unloading; (3) box-car loading for either tractor unloading or hand unloading; (4) double loads or sets for chain or crane unloading; (5) other methods of special blocking. Double loads are used principally for long lengths of structural material which exceeds the length of a single standard gondola car. Box-car loading generally is used on products which cannot be exposed to the weather and which require the best handling methods.

Material-Handling Equipment—Numerous and varied types of handling equipment are used in the finishing and shipping of merchant bar products, depending upon the type of product and the arrangement and design of the finishing department equipment. The types of handling equipment most generally used are: (1) overhead electric cranes, (2) tractors (principally the lift or peel type), (3) transfer buggies, (4) jib cranes, (5) box-car loading machines, (6) conveyors, and (7) such auxiliary apparatus as coil hooks, spreaders, package lifters, chains, and slings.

SECTION 3

HEAT TREATING CARBON AND ALLOY BAR STOCK

Heat treating may be defined as an operation or combination of operations in which a metal or alloy in the solid state is heated and cooled, under controlled conditions, according to a predetermined schedule, to obtain desired properties.

The purpose of heat treatment is to develop the full effect of the various elements in the steel as related to desired properties, through structural or phase changes. "As-rolled" bars vary in hardness and microstructure in relation to the chemical composition of the steel and, therefore, usually require some form of heat treatment to obtain a physical condition best suited for the final product. Low- and medium-carbon bars often are used in the as-rolled condition, but higher-carbon steels and most alloy steels require heat treatment. This treatment consists of some form of annealing, normalizing, or quenching and tempering, or a combination of any two or even three of these.

It has been found that the results of heat treating conform to certain definite principles, a detailed discussion of which will be found in Chapter 42. Application of these principles makes it possible to obtain transformation of a particular steel at a stipulated temperature by controlling the rate of cooling from temperatures above its critical range of temperature, and thus obtain desired mechanical properties. More detailed descriptions of how this is accomplished will be given in the following discussion of the various types of heat treating.

PROCESSES AND THEIR EFFECTS

Annealing—The term annealing is used rather loosely to describe several types of heat treatment which differ greatly in procedure yet all accomplish one or more of the following effects:

1. Remove stresses
2. "Soften," by altering mechanical properties
3. Refine the grain structure
4. Produce a definite microstructure

In most commercial operations, more than one of these effects usually are obtained simultaneously, although only one may be desired specifically. Therefore, the selection of a specific annealing process is dependent on the particular predominant or overall effect desired and the grade of steel being processed.

Full Annealing—If, for example, it is desired to refine the grain structure and produce a lamellar pearlite, a full annealing cycle should be used. This consists of heating the steel to a temperature above the transformation range, holding for one to two hours, and then cooling at a predetermined rate to obtain the desired microstructure (see Figure 30—28A). Grain refinement is accomplished in this instance by the recrystallization of the steel in passing through the critical range both in heating and in cooling, as explained in greater detail in Chapter 42. The microstructure obtained in cooling any steel from above the critical temperature range is dependent both upon the temperature range in which transformation occurs and the time required for completion of transformation in that range. Thus, it is obvious that the rate at which any steel is cooled determines the final microstructure, since the degree of transformation will depend on the amount of time allowed for it to occur. Therefore, the slower the rate of cooling and the higher the temperature at which complete transformation occurs during full annealing, the coarser the pearlite will be with correspondingly lower hardness. Such treatment is performed usually on steel of 0.30 to 0.60 per cent carbon content which is to be machined.

Isothermal annealing is a type of full annealing in which the steel first is cooled to the temperature at which it is desired to have transformation occur, at a rate sufficiently rapid to prevent any structural change above that temperature. The steel then is held at the selected temperature for the time necessary to complete such transformation (see Figure 30—28B). Thus it is possible, with this process, to obtain a more uniform

TYPES OF ANNEALING

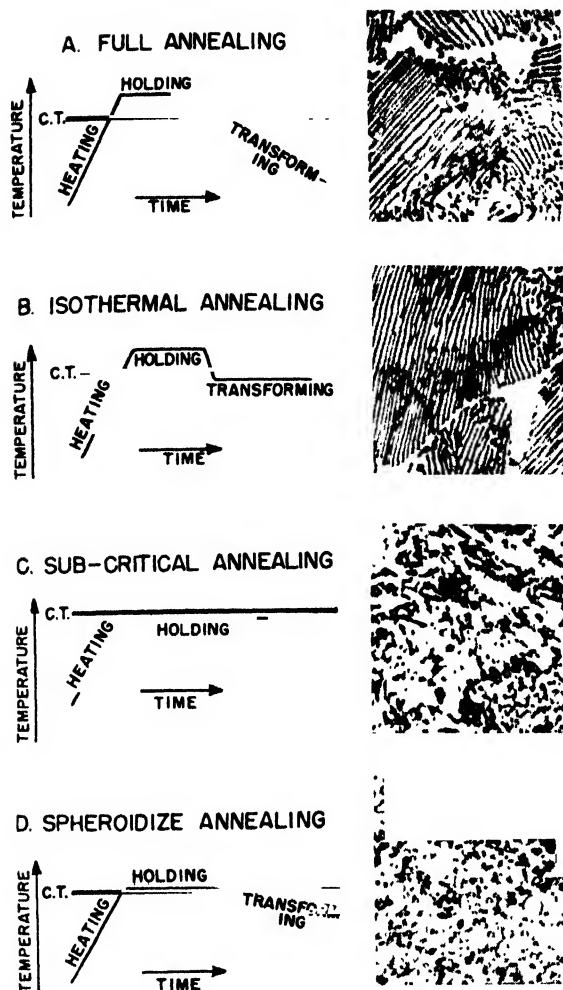


FIG. 30-28. Thermal cycles and resultant microstructures obtained from the four basically different types of production annealing.

microstructure than could be expected by continuous cooling. However, since it is necessary to drop the temperature rapidly to prevent any transformation above the desired temperature, there are definite limitations as to the mass that can be so treated. It is applicable, therefore, only to smaller sections and would not be suitable for large bars or large loads in batch-type furnaces, since it would be impossible to cool them at a rate sufficiently rapid to prevent some transformation.

A modified application of isothermal annealing is possible, however, in which the charge is heated in one furnace and transferred to another, which has been set at a temperature somewhat lower than the desired temperature of transformation, in order that the temperature of the charge will drop rapidly to that required. The selection of the temperature of the second furnace will be governed by the temperature to which the charge first is heated, the mass of the charge and the desired transformation temperature. Suitable handling equipment must be available to transfer the entire charge rapidly, since any undue delays might result in portions of the charge being cooled to too low a temperature. Continuous furnaces also are applicable to this type of cycle.

Process or Subcritical Annealing—Another type of annealing called process or subcritical annealing consists of heating the steel to a temperature just under the lower critical (A_{c1}) and holding at this temperature for the proper time (usually 2 to 4 hours) followed by air cooling (see Figure 30-28C). This type of annealing results in softening the steel due to a partial coagulation of the carbide to form spheroids or small globules of carbide. It is not suitable when a close control of hardness or structure is desired, because the prior structure of the steel determines to a marked degree the extent of spheroidization which will occur. For example, an originally coarse lamellar structure may show very little evidence of spheroidization after this treatment, whereas an originally fine lamellar or martensitic structure would show a marked degree of spheroidization. The treatment is, however, quite satisfactory for rendering bars more suitable for cold sawing or shearing, and is used to a great extent for these purposes. Since the temperature to which the bars are heated is somewhat lower than in a full anneal, there is less scaling, and warping can be controlled.

Spheroidization—Spheroidization is a type of annealing which causes practically all carbides in the steel to agglomerate in the form of small globules or spheroids. There may be a wide range of hardness with such a structure for any grade of steel since the size of the globules has a direct relation to hardness, i.e., the larger the globules the lower the hardness. Spheroidizing may be accomplished by heating to a temperature just below the lower critical and holding for a sufficient period of time. However, as pointed out in the discussion under subcritical annealing, the prior structure of the steel affects to a marked degree the final result of such a treatment. Therefore, if a well spheroidized uniform structure is desired, such a treatment is not suitable because too many uncontrollable variables are encountered. A more desirable and commonly used method for spheroidizing is to heat to a temperature just above the critical and cool very slowly (about 10°F per hour) through the critical range (see Figure 30-28D) or to heat to a temperature within the critical range but not above the upper critical and cool slowly.

This treatment is used for practically all steels containing over 0.60 per cent carbon that are to be machined or cold formed and for bearing steels such as AISI-52100 prior to machining. For the latter, the size and distribution of the spheroidal carbide particles is extremely important since they all are not dissolved in subsequent heating for hardening but remain in the condition obtained in the annealing process. Small well-defined spheroids are the most desirable and coarse elongated carbides should be avoided at all times.

Normalizing—Normalizing is a process wherein the steel is heated to a temperature about 100° to 150°F above the transformation range and cooled in air. The resulting structure will differ greatly for steels of different chemical composition, since no attempt is made to regulate the temperature at which transformation on cooling occurs. It is used for the purpose of producing a more uniform structure and removing the irregularities caused by high or low rolling or forging temperatures for large sections. It also is applicable to a wide extent in treating low-carbon steels of all sections to produce a more uniform structure, and is preferred to full annealing for these steels as the more rapid cooling minimizes banding (marked segregation of carbide and ferrite) which is very undesirable for machining.

Quenching and Tempering—The heat treating processes described so far have all been for the purpose of "softening" the steel by regulating the rate of cooling

HEAT, QUENCH AND TEMPER

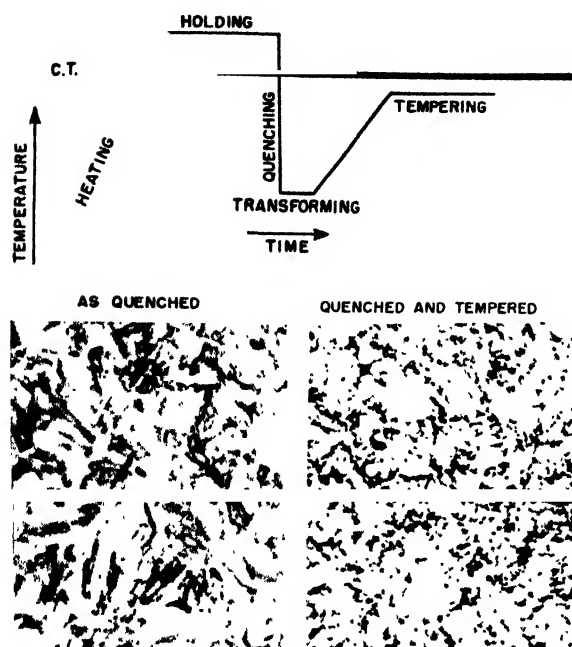


FIG. 30—29. Thermal cycle and microstructures obtained in (a) quenching and (b) quenching and tempering.

so that transformation occurs at relatively high temperatures. Such processes are necessary to render the steel suitable for further operations such as machining and cold cutting. However, this condition does not represent the optimum mechanical properties which can be developed in the steel, and a further treatment termed quenching and tempering is employed to develop these properties. When the amount of machining or cutting to be done is not great, steel sometimes is given this treatment in the bar form in preference to an annealing treatment.

The combination of quenching and tempering consists of first heating the steel above the critical range, and then cooling it rapidly by immersing it in a liquid

cooling medium such as oil or water. If the rate of cooling is sufficiently rapid, transformation does not occur until the lower temperature ranges are reached with the resultant formation of martensite or bainite. These structures are much harder than the structures obtained by transformation at higher temperatures. Martensite, however, is quite brittle and would be unsuitable for most applications where the steel may be subjected to shock. Therefore, the steel is given a tempering treatment which consists of heating it to an intermediate temperature, very seldom higher than 1200° F and usually somewhat lower. This treatment reduces the hardness by coagulation of the carbide and increases the toughness or shock resistance of the steel. A schematic illustration of quenching and tempering with the resultant microstructures after each operation is shown in Figure 30—29. It is possible to obtain any desired hardness within a wide range, with corresponding variations in strength and ductility, by selecting the proper tempering temperature. This can be seen readily by examining the chart in Figure 30—30.

Tempering is sometimes called *drawing*. Tempering is the preferred nomenclature. In ancient times and in the early days of relatively modern steelmaking practices, steel for swords, tools, etc., was said to be "tempered" after it only had been hardened. Since the hardened steel was too brittle, some of its "temper" was "drawn" by suitable heat treatment, giving rise to the term "drawing." At present, temper is considered as describing the final condition of hardened and tempered steel, and it is more logical to give the operation that regulates the final condition the name "tempering."

HEAT-TREATING PLANTS

Modern heat-treating plants vary widely with respect to type of furnaces, auxiliary equipment and general arrangement. No matter how widely they may differ in these general aspects, all must have adequate and reliable means for controlling and checking temperatures and have handling facilities suitable for transferring the heated steel from the furnace to the quenching tanks with a minimum of delay. These are factors which must be given careful consideration when planning new units or additions to existing units, and will be governed to a great extent by the type of product to be processed and the heat treatment to be given.

Another important factor to be considered is the pro-

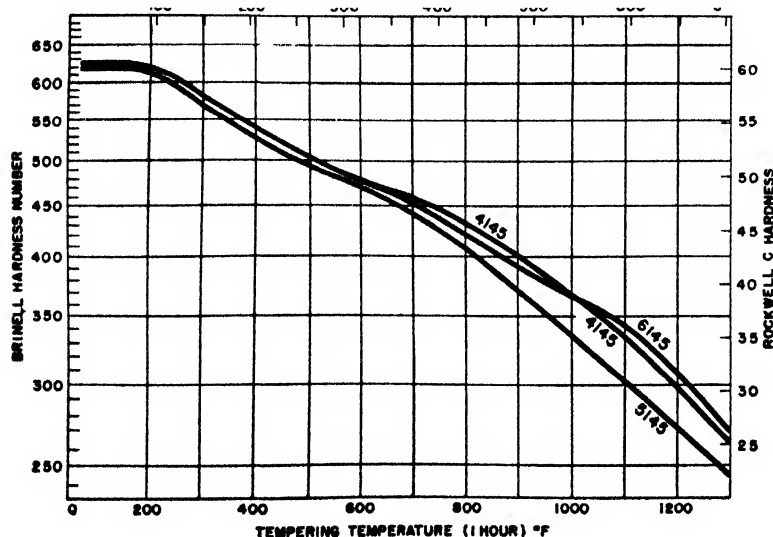


FIG. 30—30. Effect of tempering temperature on the final hardness of three grades of steel after quenching and tempering.

vision of adequate space for accumulating and building up charges so that they may be ready to load into the furnace as soon as it is available. This lessens time lost in charging the furnaces and increases the efficiency of these units.

Furnaces—In general furnaces which are suitable for annealing are not the most desirable for quenching and tempering, nor are the most desirable furnaces for quenching and tempering entirely suitable for annealing. This can be shown by analyzing the basic requirements for furnaces considered most suitable for these operations. Many of the factors discussed in Chapter 21 on "Construction and Operation of Heating Facilities" should be considered in conjunction with the following discussion.

A furnace to be suitable for quenching and tempering must be capable of maintaining uniform temperatures over a rather wide range throughout the entire charge. It can be heated by either gas or electricity, and possibly oil, although the latter fuel is not too desirable at temperatures below 1200° F.

Since uniform temperatures must be maintained in furnaces to be used for quenching or tempering, and rapid cooling of the charge in the furnace is not required, such furnaces usually are insulated heavily to minimize heat losses due to radiation. Since only single-layer loads are charged for quenching, the height of the heating chambers need not be very great. This enables a more uniform temperature to be maintained throughout the charge since the differential in temperature between the top and bottom of the furnace is much less than would be the case in a larger heating chamber.

Uniformity of temperature is also very desirable in furnaces used for annealing, but the requirements are not necessarily as exacting as for furnaces used for quenching and tempering. However, it is essential that a rather wide range of cooling rates be attained. Therefore, furnaces that cool too slowly, such as the heavily insulated type referred to in the discussion on furnaces for quenching and tempering, are not suitable for very

many annealing cycles. A furnace which will lose heat at a fairly rapid rate can be adapted to a wider range of cycles because various rates of cooling can be attained by control of the heat input. Thus, if the heat input is cut off entirely, the furnace and charge will cool rapidly; by firing at reduced rates, the heat losses from the furnace and charge can be balanced to any desired degree to control the net heat loss and, consequently, the over-all rate of cooling. Possible methods for increasing the rate of cooling in a heavily-insulated furnace would be to blow air into the furnace or to open it. This is not very desirable, however, because it is extremely difficult to regulate the temperature drop uniformly throughout the charge and erratic, non-uniform results may be obtained. Furnaces have been developed recently which are equipped with large fans that circulate the furnace gases over water-cooled pipes, thus reducing the temperature to around 800° to 900° F. The partially-cooled gases are then forced into the furnace chamber at the bottom and up through the charge, causing a rapid drop in temperature. Such equipment has many potential advantages but care must be exercised in its use to avoid non-uniform cooling.

Since annealing cycles are usually rather long and tie up a furnace for some time, it is desirable that they be large enough to accommodate the maximum size load that can be cooled uniformly. For this reason, furnaces designed for annealing have a great deal more height in the heating chamber than those designed exclusively for quenching. Consequently, there is more danger of greater differential in temperature between the top and bottom of the chamber. This affects selection of the type of furnace to be used.

Control of Temperature—As stated previously, adequate and reliable control and checking of temperature is absolutely essential for all types of heat treating. Furnaces equipped with automatic temperature-control devices are generally more reliable and can be operated more economically than manually controlled units. Thermocouples to indicate and control temperature gen-

FIG. 30-31. Partially-loaded car-bottom of a heat-treating furnace, showing use of spacer bars between layers of the load, which is supported above the refractory hearth on heat-resisting alloy castings.

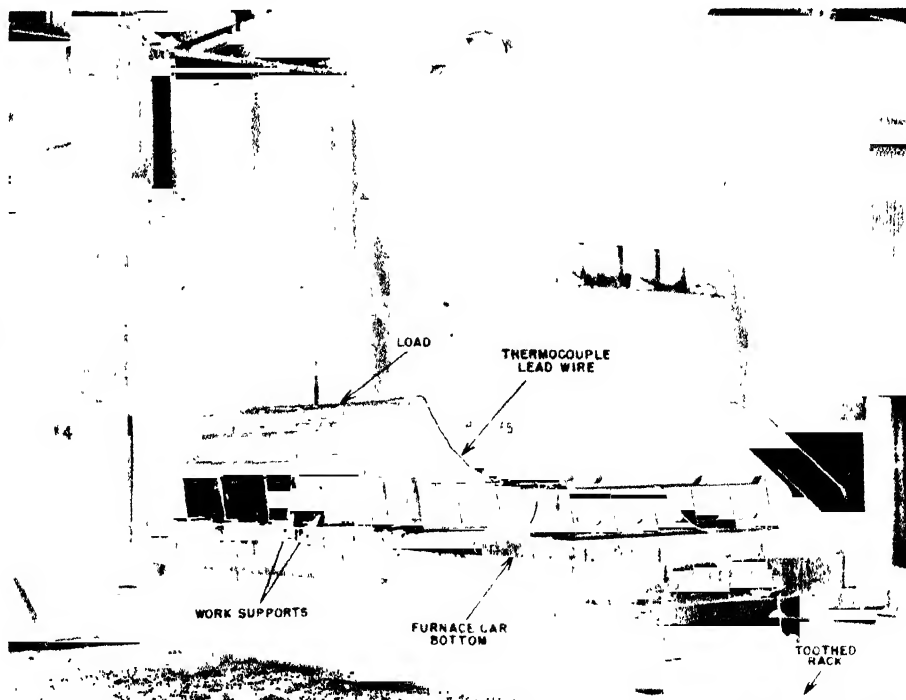


FIG. 30-32. Car-bottom batch-type heat-treating furnace with bell-type cover in operating position.



erally are located near the top of the heating chamber, as this location will be the first to reach the desired temperature on heating; placing the controlling thermocouples in this position prevents overheating. In the case of furnaces designed primarily for quenching and tempering, these couples will be close to the charge (which consists of only one layer of bars or other sections) so that additional couples should not be needed to assure uniform heating. However, in the case of furnaces designed primarily for annealing, in which the chamber height is much greater and in which there is a correspondingly greater possibility of temperature differentials, provision should be made to insert couples in the charge also, being sure that some are located so that they indicate the temperature of the bottom layer. With couples so located, the temperature throughout the

charge will be known and sufficient time can be allowed to attain uniformity.

Methods of Loading—As stated previously, loads to be quenched should consist of only one layer; if there is more than one layer the load would not be uniformly quenched because the efficiency of this operation is dependent on the entire surface of the bar being in contact with the quenching medium. The same principles apply to normalizing, since uniform properties are dependent on attaining a uniform rate of cooling along the entire length of the bar, which can only be obtained by having the entire surface exposed to the air and not in contact with other hot bars.

Loads for annealing are generally more than one layer high but care should be taken to avoid having a load that is too compact, as this would prevent uniform

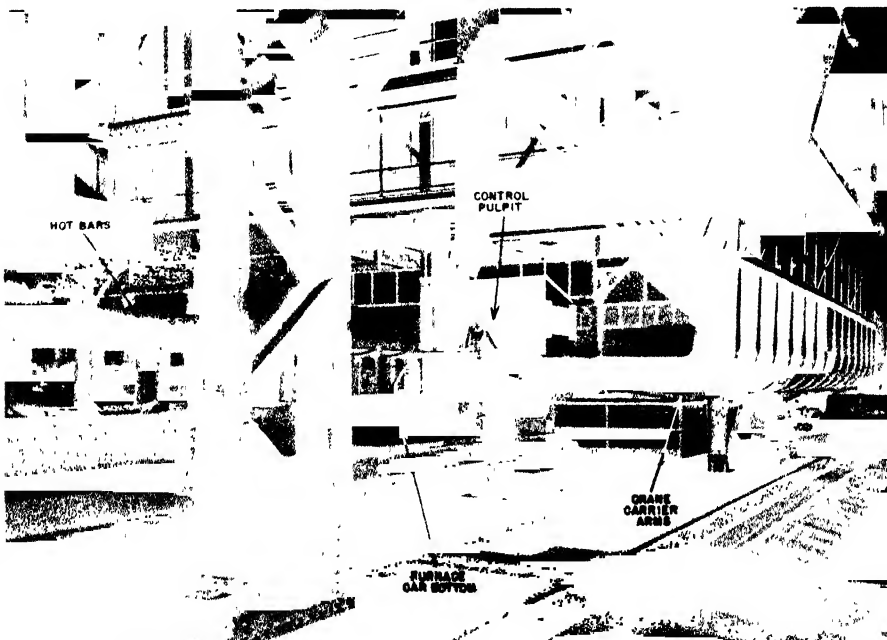
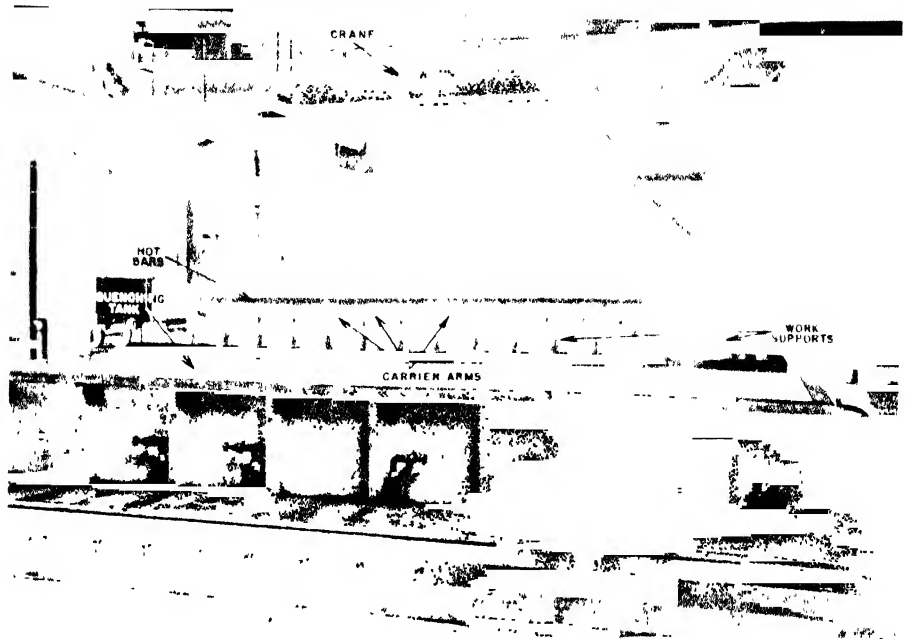


FIG. 30-33. The bell-type cover (upper left corner of illustration) of the heat-treating furnace of Figure 30-32 has been raised and hot load is being moved on furnace car bottom toward crane carrier arms.

FIG. 30—34. Hot steel bars, supported on crane carrier arms above the quenching tank in the foreground (see also Figure 30—35).



cooling throughout the entire charge. A common practice is to separate the load into layers by the use of spacer bars, as shown in Figure 30—31. The load also may be separated lengthwise down the middle, leaving an open space approximately six inches wide. These methods of loading provide for better circulation of gases throughout the charge, thus enabling the operator to cool the entire charge at a reasonably uniform rate.

Auxiliary Equipment—Effective quenching is dependent on immersion of the steel in the quenching medium while the temperature is still above the critical range. If there is too great a delay in moving the load from the furnace to the quench tank, it may air cool below the

critical range, especially at the ends of the bars. Thus, there is the danger of obtaining some products of transformation in the higher temperature ranges. To offset this hazard the quench tanks should be located close to the furnace and cranes or lifting devices should be provided which are capable of moving the entire load to the tanks at a rapid rate. By moving the entire load at one time, all bars will receive the same treatment. If only one bar or any portion of the load is quenched at a time, there is always danger that the last bars quenched will have cooled below the critical temperature before reaching the quenching tank.

Figure 30—32 shows a car-bottom batch-type furnace

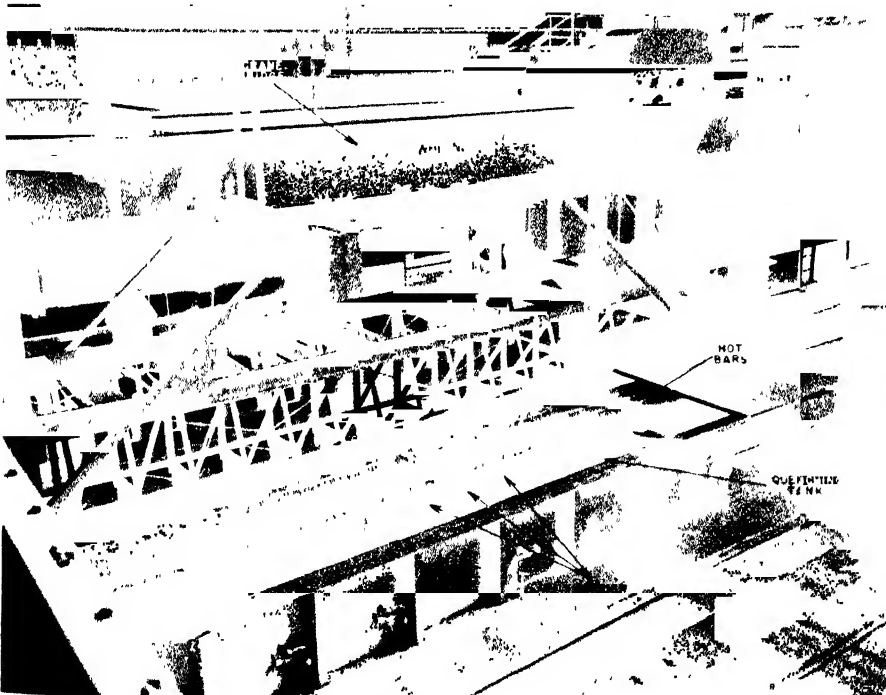


FIG. 30—35. Looking down on a load of hot steel bars supported on carrier arms above quenching tank.

FIG. 30—36. Hot steel bars in position on elevator roll-table section above quenching tank, ready to be lowered into the quenchant. This is furnace "C" in Figure 30—37.

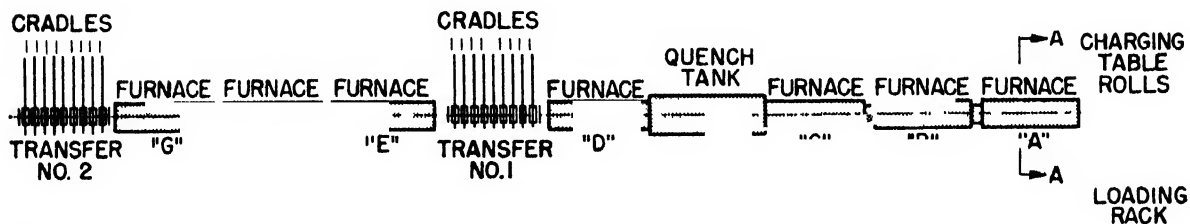
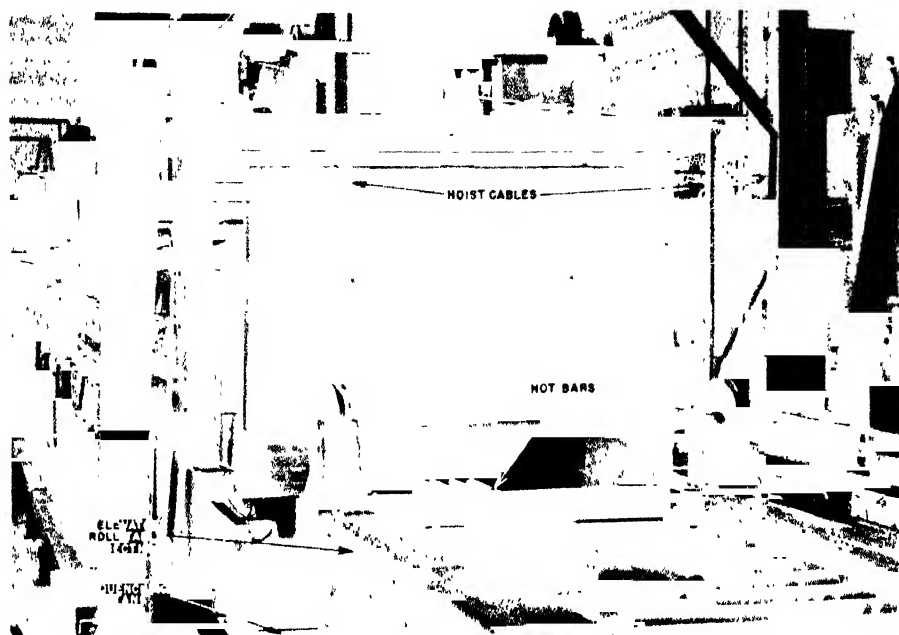


FIG. 30—37. Layout of the various units comprising a typical continuous heat-treating line for bar products. Work travels from right to left. See also Figures 30—38 and 30—39.



FIG. 30—38. Perspective view of most of the actual continuous heat-treating line shown diagrammatically in Figure 30—37.

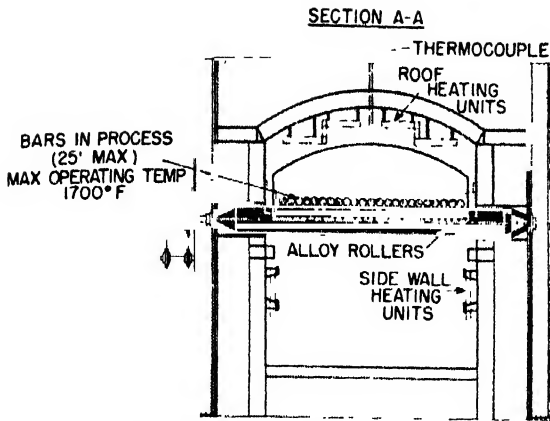


FIG. 30-39. Schematic cross-section of one of the high-temperature furnaces in the line shown in Figure 30-37.

equipped to handle loads rapidly from the quench tank to the furnace. This is a rectangular bell-type furnace with a car bottom. To discharge the load, the furnace bell is raised (there are no doors on this furnace) and the car is moved out from under it, alongside the quenching tank which is in front and to one side (see Figure 30-33). The entire load then is picked up by a crane, equipped with carrier arms or bales as shown in Figures 30-34 and 30-35, which moves it quickly into the quenching medium. Agitation is obtained by alternate raising and lowering of the load in the tank through a travel distance of approximately four feet. This particular unit is equipped with automatic controls so that the entire sequence of movements of the equipment is initiated by closing a single control switch. This eliminates delays which might be caused by improper timing in closing successive switches by the operator. The time required from the instant the bell of the furnace is lifted until the load is immersed in the quenching medium is one and one-half minutes.

A roller-hearth furnace with similar handling equipment is shown in Figures 30-36, 30-37, 30-38 and 30-39. In this furnace, the bars in single layers are placed on the charging table indicated in Figures 30-37 and 30-38. From here they progress through the three separate chambers of the furnace, where they are pre-

heated to about 1200 ° F in the first chamber, then progressively heated above the critical temperature range and soaked in the other two. Progress is not continuous through these chambers, the bars being held for a sufficient period of time in each chamber to be heated throughout to the prescribed temperature. The time required in each chamber, of course, is determined by the size and number of bars being treated. After the bars have been soaked properly in the last chamber, they are run out on the delivery table, which is suspended over the quenching tank (Figure 30-36), and immediately are immersed in the quenching medium. When bar temperature has fallen sufficiently, they are raised from the tank and run into the succeeding furnace for stress relieving, after which they may be removed from the transfer table or continue through the remaining three furnaces for tempering.

Facilities should be available for quenching in either oil or water. The choice of quenching medium is to be determined by the grade of steel and the section, as explained in greater detail in Chapter 42. Facilities also should be provided for cooling the oil between the quench loads to maintain a uniform temperature of the quenching medium, because variations of an appreciable magnitude in this temperature will have a marked effect on results. Storage facilities for the oil also should be provided since a reserve supply is necessary to replenish that lost through dragout. One quenching tank can be used for either oil or water, the oil being pumped back to the storage tank when it is necessary to use water in the quenching tank. However, this is not the most desirable arrangement since there is always danger of contaminating the oil with water. It is more desirable to have separate tanks for oil and water when the size of the unit warrants such an installation.

Chapter 31

PRODUCTION OF WROUGHT - STEEL WHEELS

Introduction—Classified according to methods of manufacture, steel wheels are either fabricated from several pieces joined by riveting or welding, or they are made in one piece and designated as solid iron or steel wheels. Very light wheels, such as automobile wheels, and extremely large wheels generally are fabricated. Solid steel wheels are either cast from liquid metal or wrought from a solid block of metal. According to modern practices, therefore, the term wrought-steel wheel is applied to wheels formed from a suitable block of metal by forging alone or by forging and rolling operations. This chapter describes the production of wrought-steel wheels by both forging and rolling.

Parts and Classification of Wrought-Steel Wheels—Wrought-steel wheels generally are used in heavy traction service. Such wheels are designed with contours which are considered as being composed of five parts, namely, the hub, the web or plate, the rim, the tread and the flange. These parts, for the case of a railroad wheel, are illustrated in Figure 31—1. The flange may be single or double, and the hole in the hub into which the axle is fitted is called the bore. The tread is the outer surface of the rim on which the wheel rolls or makes contact with the rail. The flange keeps it from leaving the track. The dishing, or coning, refers to the curvature of the web section. All traction wheels have an inside contour and an outside contour. The flange, the rim, the service to which they are applied, the methods of forming, the finish, and the finishing treatments are all bases for classifying wheels. Thus, they may have a single flange (a flange on the inside only) or a double flange (a flange on both sides) and a rim thickness that will permit worn treads and flanges to be machined one or more times to

their original contours. **One-wear wheels** have the rim thickness designed so that it is unusual for the rim to be recontoured after one period of wear. **Multiple-wear wheels** have a rim thickness of $2\frac{1}{2}$ inches or more, and may be machined two or more times to the original tread and flange contour.

Classifications based upon the service to which the wrought steel wheels are to be applied are defined as:

(1) **Industrial car wheels** are single-flange wheels designed for use under cars used for industrial purposes, such as mine cars, railroad hand cars, and cars or buggies used in steel mills.

(2) **Industrial locomotive wheels** are single-flange wheels designed for use under electric and other types of locomotives used in mines, and in industrial plants.

(3) **Crane track wheels** may have either a single or a double flange, and are used under such equipment as traveling, gantry and bridge cranes, transfer and turn tables, and floor-type charging and drawing machines.

(4) **Railroad freight-car wheels** are commonly rolled to finish on treads and flanges, and have hub faces as forged.

(5) **High-duty wheels** are manufactured to closer tolerances than freight-car wheels and are used under railroad and electric-railway passenger cars, and under tenders, and engine and trailer trucks of locomotives.

Wrought-steel wheels are completely covered by standard specifications including the type and kind of steel used, the design of the wheels, the dimensional tolerances, and the inspection and tests. While alloy steels are used in some circular shapes, wrought-steel wheels, heretofore, have generally been made of carbon

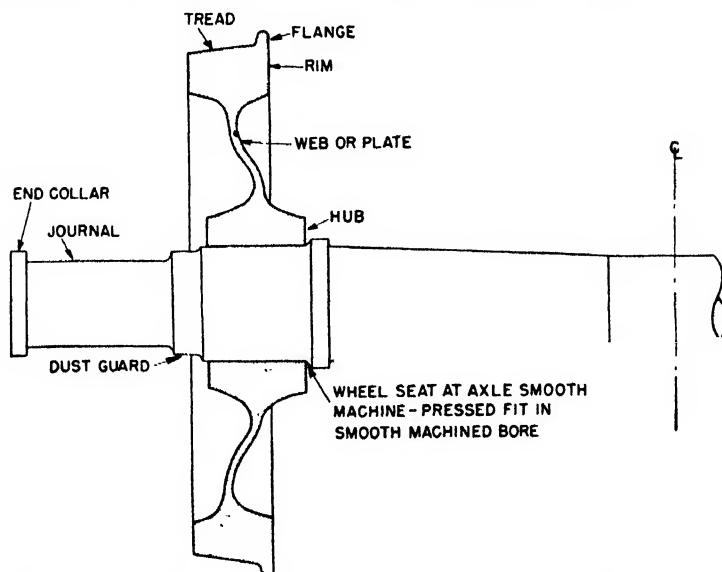


FIG. 31—1. Sketch showing cross-section of a wrought-steel railroad wheel mounted on an axle, giving nomenclature of the principal parts of each.

steel. The remainder of this chapter will be devoted to the manufacture of freight-car and high-duty wrought-steel wheels.

Classified according to methods of manufacture, wrought-steel wheels may be forged; forged and rolled; rolled to finish; machined on tread, flange, and hub faces; or machined all over. With respect to heat treatment, wheels are no longer air cooled from forging or rolling temperatures, but are always controlled cooled, rim toughened, or oil-quenched and tempered by processes to be described later.

Classes of Wrought-Steel Wheels—Wrought-steel wheels are manufactured to Association of American Railroads and American Society for Testing Materials specifications. These specifications call for four classes of wrought-steel wheels. The four classes differ in chemical composition and hardness and were developed after extensive research followed by many studies of wheels in different types of railroad service. Each class is processed and treated for a definite service and it is important that wheels of the proper class be used in the service for which they are intended.

American Association of Railroads specification M-107 and American Society for Testing Materials specification A-57 cover one class of untreated wheels (Class U) and three classes of heat-treated wheels (Classes A, B and C) grouped according to carbon content, as follows:

Element	Per Cent
Carbon	Class U..... 0.65-0.77
	Class A, not over..... 0.57
	Class B..... 0.57-0.67
	Class C..... 0.67-0.77
Manganese.....	0.60-0.85
Phosphorus, not over.....	0.05
Sulphur, not over.....	0.05
Silicon, not less than.....	0.15

Classes A, B and C are heat treated to the following hardness requirements:

Class	Brinell Hardness Number	
	Minimum	Maximum
A	255	321
B	277	342
C	321	363

These specifications state:

"The Brinell measurement shall be made on the front face of the rim with the edge of the impression not less than $\frac{3}{16}$ inch from the radius joining face and tread. Before making the impression, any decarburized metal shall be removed from the front face of the rim at the point chosen for measurement. The surface of the wheel rim shall be properly prepared to permit accurate determination of hardness."

These specifications require that heat-treated wheels be stamped with the class letter A, B or C, together with the letter E for wheels entirely quenched, and the letter R for wheels that have the rim quenched, and that both letters be enclosed in angular brackets as follows:

Mark	Significance	
	Class	Treatment
[AR]	A	Rim Treated
[BR]	B	Rim Treated
[CR]	C	Rim Treated
[AE]	A	Entire wheel treated
[BE]	B	Entire wheel treated
[CE]	C	Entire wheel treated

Outline of Methods for Forming Solid Wrought-Steel Wheels—All methods for forging and rolling solid wrought-steel wheels require a cylindrical, or nearly cylindrical, block of steel for the initial forming operations. Originally, the blanks were sheared from slabs, but this method located the central portion of the ingot diametrically across the wheel, so that the line of segregation terminated at opposite points in the tread. The objections to this method are obvious. Therefore, the procedure was changed, as shown in Figure 31-2, so that

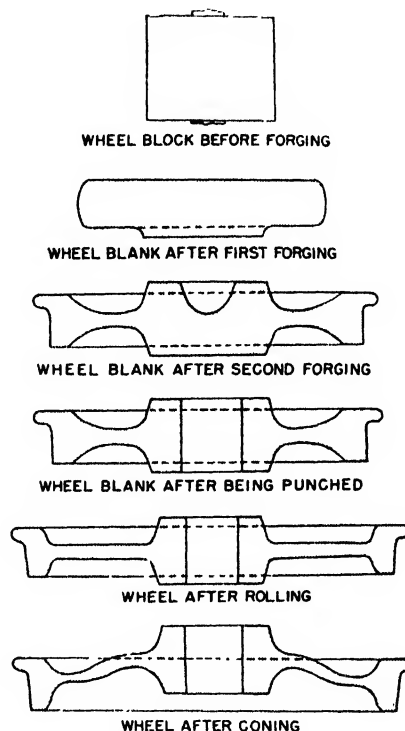


FIG. 31-2. Cross-section showing steps in the procedure of making a wrought-steel wheel by a combination of forging, rolling and pressing operations.

metal in the line of segregation, or central portion of the ingot, would be punched out in forming the bore.

To accomplish this, the steel blocks may be prepared in any one of three ways, namely, (1) by casting the metal for each wheel as an individual ingot, (2) by cutting the blanks from ingots, or (3) either by hot shearing or cold cutting the blanks from a round bloom. The United States Steel Corporation, one of the producers of solid steel wheels, uses the third method. After the blocks have been press forged to the general form of a wheel, called the **wheel blank**, the shaping is completed by rolling the blank in the wheel mill which may be one of two types, designated from the position in which the blank is held in the mill during the rolling, as a vertical mill (Figure 31-3) and a horizontal mill (Figure 31-4). The practice will be described in detail as an example of the manufacture of wheels.

PREPARATION OF BLOCKS

Wheels are produced by the United States Steel Corporation at Gary Steel Works and at the Wheel and Axle Division of Homestead District Works. While the practices of these two plants differ somewhat as to equipment and details of manipulation, this difference is not

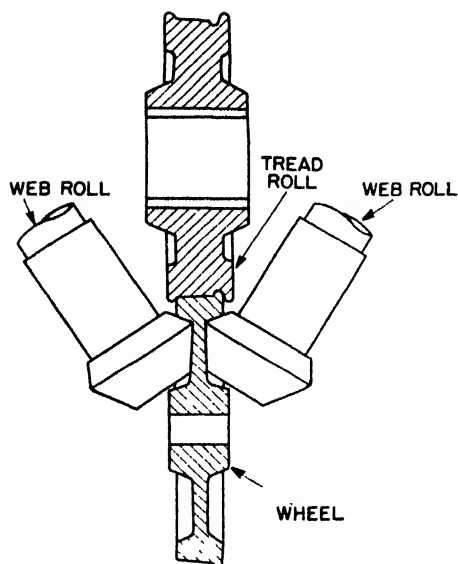


Fig. 31-3. Schematic arrangement of the rolls in a vertical mill for rolling wheels.

of a fundamental nature. Both plants are equipped for controlled cooling of wheels and for heat treating by all the approved methods. At Homestead the steel to be rolled into round blooms for wheels is cast in 25-inch by 25-inch hot-top molds with rippled or corrugated walls, and rolled on the 52-inch mill into round blooms either 15 inches or 17½ inches in diameter, according to the weight of the wheels to be made. With proper allowance for discard, these rounds are cut, while still hot, into blocks to form a wheel of the desired weight and dimensions, by a special shear which cuts the bloom almost in two but leaves a central core about 5 inches in diameter to be fractured when the bloom has cooled to atmospheric temperature. Instead of air cooling, however, the partially sheared blooms from the shears are put into special, insulated containers and allowed to cool slowly to 300° F. The cooling is thus controlled as a

precaution against the possible formation of minute internal cracks in the blocks. The blooms are then unloaded at the shipping yard, inspected, surface conditioned, and fractured. At Gary, the steel is cast in 25-inch by 25-inch big-end-up hot-top molds and rolled into 18-inch round blooms on the 42-inch mill. While still hot, these blooms are sliced into blocks by a special shear which cuts to the center of the bloom, and the blocks are individually inspected and surface conditioned as may be necessary. As each block is separated from the bloom, it is hammer stamped with numbers to identify it. At both plants the smallest wheel that can be rolled is about 26 inches in diameter, and the maximum sizes are 40 inches in diameter at Gary and 46 inches at Homestead.

FORGING OF WHEEL BLANKS

Heating the Blocks for Forging—This operation is carried out in two stages, referred to as the "preheat" and the "forge heat." Hence, at the wheel plant the blocks are first placed in a gas-fired preheating furnace where they are slowly and uniformly heated to a temperature of 800° F in a period of 6 hours or longer. The object of this slow heating is to avoid abrupt temperature gradients and resulting high internal stresses that occur in a block when its surfaces are suddenly exposed to a high temperature while the central portion is cold. The preheating furnaces are preferably of the continuous type in which the blocks may move from the cold end to the hot end, but it is possible to obtain similar heating effects in sand-bottom furnaces, in which the blocks remain stationary, by charging a number into the furnace when it is relatively cool and gradually raising the temperature.

After preheating, the blocks are transferred by charging machine (Figure 31-5) to the forge furnace, which is of the sand-bottom type. This furnace is held at a uniform heating temperature so that the blocks reach a forging temperature in about 2½ hours, and the blocks are charged and drawn in a definite order, so that none will be overheated or underheated. Full control of the heating combines automatic pyrometric control with skill and judgment of the heaters, who become expert in estimating both steel and furnace temperatures.

First Forging of the Block—As indicated in Figure 31-2, the forming of the wheel blank preliminary to

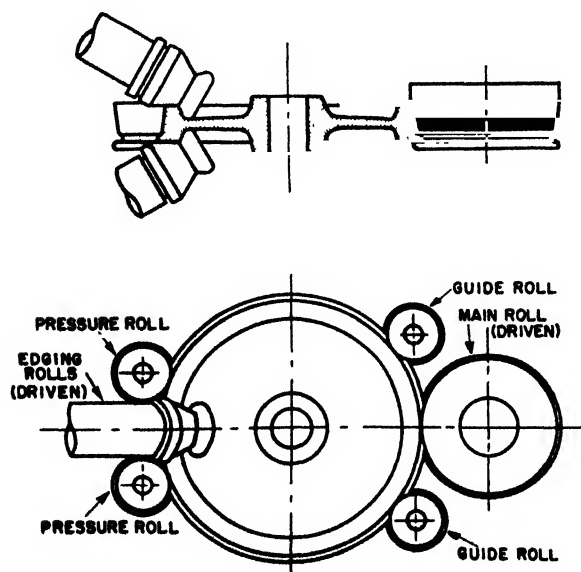


Fig. 31-4. Elevation (above) and plan (below), showing disposition of the rolls in a horizontal mill for rolling wheels.

FIG. 31-5. Operator of a charging machine drawing a heated wheel block from furnace preparatory to transfer to the forging press.

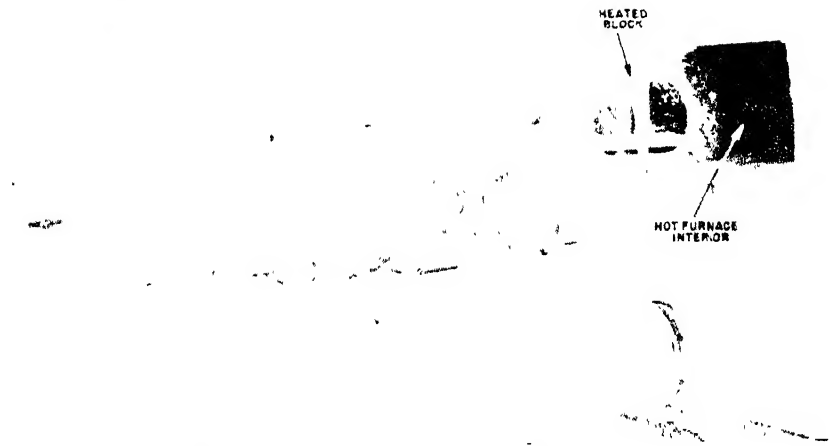


FIG. 31-6. Hot wheel block in position between the dies of a forging press at the start of the first operation in forging a wheel.

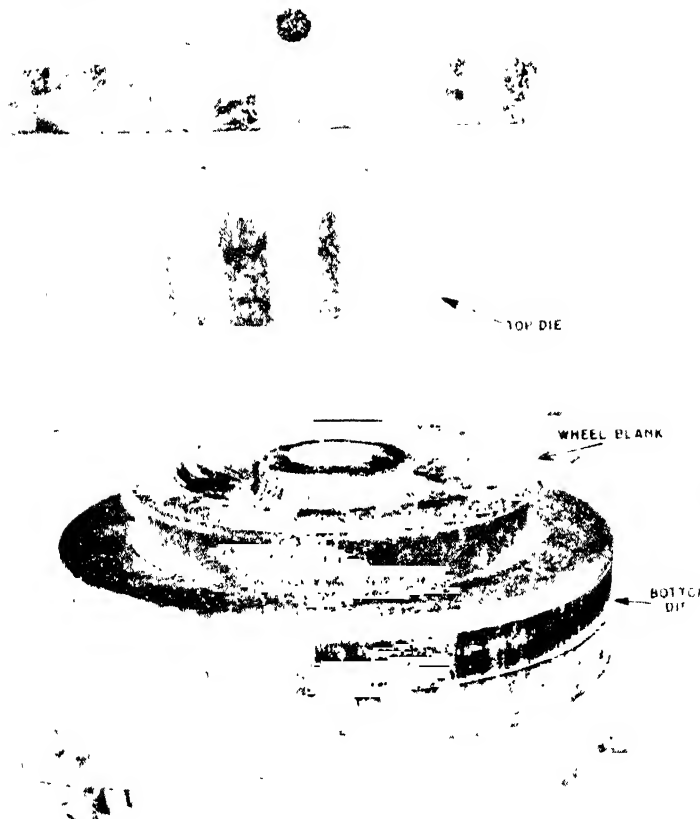


FIG. 31—7. This shows the wheel blank after the second forging operation. Note indentation formed in center of hub to facilitate punching of the bore.

the final forming in the rolling mill is carried out in three steps spoken of as the first forging, the second forging, and punching the bore. However, in the forging operations it is essential that the block or bloom be free of scale formed during the reheating. In some plants this scale is removed by mechanically-operated chain beaters; in others various procedures are used. An effective method is carried out as follows: When the block is removed from the forge furnace and delivered by the charging machine, the scale on the ends is knocked off with long handled steel scrapers. The block is then placed in the bottom die of the press and held in a vertical position, while the press is started slowly. When the top die contacts the block, the first deformation cracks off the scale, which is immediately blown out of the die by jets of compressed air. The press then continues the stroke to complete the first forging. From this point, the practices at different plants vary somewhat. Thus, the first forging may be merely an upsetting of the block and punching of the bore to form a circular bloom. At present, the practice at plants of the United States Steel Corporation is to do some forming in the first forging, and punch out the bore after the second forging (Figure 31—2). However, this procedure is subject to some variation, for the forging may be done in one heat, or in two

heats and on one press or on two different presses.

When two heats are used, two presses are usually employed for the forging. In this method, the scale is removed from the two ends of the block and it is then placed vertically in the first press, where it is first centered by two arms which engage it from opposite sides of the press and so support it until the top die has descended upon it (Figure 31—6). In this press, the bottom die corresponds to the outside face of the wheel, while the top die is not contoured, but may be slightly convex to cause a radial flow of the metal in taking the shape. The pressure, applied in successive steps through accumulators and intensifier, starts with about 700 pounds per sq. in., then increases to 2500 pounds per sq. in., and finally, to as much as 5000 pounds per sq. in., if needed. In general, the presses used for forging wheel blooms and blanks vary from 5000 to 12,000 tons capacity. This pressing is, in itself, a severe test upon the metal, and any occasional flaws, such as seams or cracks, are sure to be exposed. When such flaws do develop, the blank may be reconditioned if the flaws are of a minor character, but if they are of an injurious nature the blank is scrapped. The perfect blanks are now placed in a second reheating furnace, where their temperature is equalized and they are again brought to forging temperature.

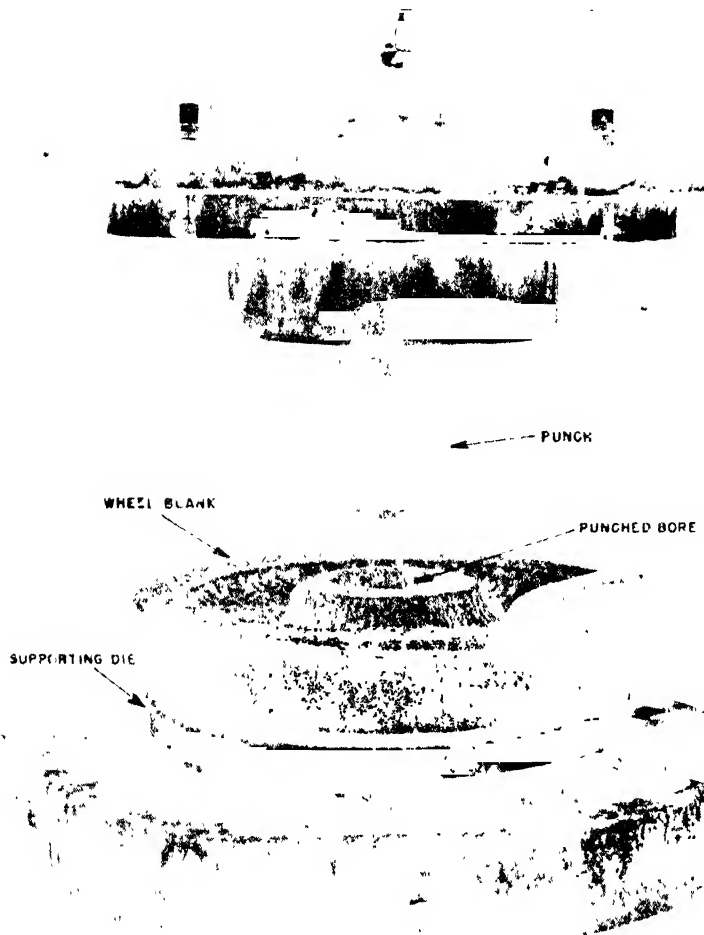


FIG. 31—8. Forged steel wheel after the operation of punching the bore.

Second Forging—If both the first and second forgings are done on one heat, sometimes designated as double forging, the press is provided with two top dies, one plain or slightly convex and the other conforming to the inside of the wheel. These dies are mounted upon two crossheads in such a manner that either may be brought beneath the ram of the press, thereby dispensing with one forging press and permitting the forging to be accomplished in one operation when the conditions are favorable. Generally, however, the plain die is used first, then the inside die is moved into position and the pressure applied, thus forming the blank in two stages, but with a single heating. With this press, the procedure can be varied to permit reheating if necessary, but the object is to accomplish the forging of the blank in one heat. When the forging is done in two steps with intermediate reheating, the procedure is about as follows: When the temperature of the blank formed in the first forging has been restored and equalized, the blank is brought to the press, cleaned of scale, placed in the bottom die which is lubricated to prevent sticking, and formed by a stroke of the press, the top die of which conforms to the inside of the wheel (Figure 31—7).

Punching the Hub Bore—By the process described, the next step consists of punching the bore in the hub. This operation is carried out with a small press such as that shown in Figure 31—8. During punching, the outside of the hub of the wheel is supported in neatly fitting dies in order to avoid forcing this part of the wheel out of shape.

ROLLING OPERATIONS ON WHEELS

Reheating the Blanks for Rolling—The forging now resembles a wheel and is called a wheel blank. However, a great deal of work must still be done on the blank to produce the finished wheel. While in general the blank may lack tread and flange, these parts have been partly formed in the forging operations. As to dimensions, the forged blank is 3 to 5 inches undersize in diameter, some three-fourths of an inch oversize in that part of the web near the rim, and considerably oversize in the rim. These oversize parts contain metal for the working and shaping to be done in the rolling mill, and dimensions will vary with different sizes of wheels and the method of rolling. The hub and a small annular section of the web next to the hub are forged to correct size for the finished wheel. In some processes the forged blank is not reheated for the rolling, but when forged as described above, certain parts, such as the web, are liable to become cooled to a greater extent than other parts, and temperature equalization becomes desirable. This reheating is done in flat-bottomed gas-fired furnaces, similar to the reheating furnaces mentioned in connection with the forging. The reheating furnace has other advantages, also, as it provides means of holding the blanks during interruptions in mill operations from any cause and of reheating blanks in which slight flaws have developed. To eliminate these flaws the blanks are cooled under controlled conditions, the flaws are removed by chipping or grinding, and the blanks are re-

heated in two stages as described for the heating of blocks. These temperature equalizing furnaces, as well as the reheating furnaces, are equipped with recording pyrometers as an additional precaution against overheating or too rapid a heating of the blanks.

The Rolling Mill—Mills for rolling wheels are of two types, vertical and horizontal, already shown diagrammatically in Figures 31—3 and 31—4. The following description, however, is confined to mills of the vertical type used at both the Homestead and Gary plants of the United States Steel Corporation. Even these two mills vary somewhat as to details of construction and operation, and each represents a somewhat complicated piece of machinery in which 5 or 7 rolls are made to bear on the wheel during the rolling. These rolls consist of one tread, or back roll, two web rolls, and two or four (2 sets) rim rolls, which are supported, together with all of their bearings, pinions or gears, adjusting screws, levers, etc., in a single pair of horizontal housings. These housings are large steel castings placed one above the other. The bottom housing lies directly upon the mill foundation and forms the support for the rolls and for the top housing some four feet above it.

The housings, between which the rolls are located, are held apart by suitable pillars or posts and are bound firmly together by large bolts. The rolls may be described as follows: The largest roll is the back roll, which resembles a wheel in form, approximately 33 inches in diameter, and is located in the same vertical plane as the car wheel blank, to the rear of the central axes of the housings. This roll is either driven or revolves by friction, bearing on the rim of the wheel, forming the tread

and flange. The Homestead mill is equipped with a variable-speed motor, while the Gary mill uses a constant-speed motor.

On the opposite sides of the tread roll are located the two web rolls. They are about three feet in length, lie in a horizontal position and extend inward, so that their center lines form nearly 30 degree angles with the mill center line and intersect at a point near the center of the wheel that is being rolled. On their front ends they carry the rolling heads, or surfaces, which conform to the shape of the wheel beneath the rim, while their rear ends are anchored in rotating coupling boxes. Light steel spindles, some five feet in length and provided with suitable wobblers, connect these couplings to the two bevel gears, one of which stands on each side of the mill at the rear. These gears mesh into similar gears mounted on the driving shaft of a 750-horsepower, direct-current motor (130 r.p.m.), used to drive these rolls. This motor is located at the rear and on the center line of the mill. Just back of the rolling heads, these rolls are supported in sliding bearings which permit them to be spread as desired. The pressure for rolling is transmitted to these bearings through radial levers, the long arms of which are each attached to the same screw above the housings (see Figure 31—9). By this arrangement, with the same motion, but in opposite directions, equal pressures are imparted to the two rolls at the same time. This screw, which corresponds to the adjusting screws on ordinary mills, is actuated by a 15-horsepower, direct-current motor (550 r.p.m.). By this means, the power of the motor is multiplied many times and is capable, at its maximum, of exerting sufficient pressure

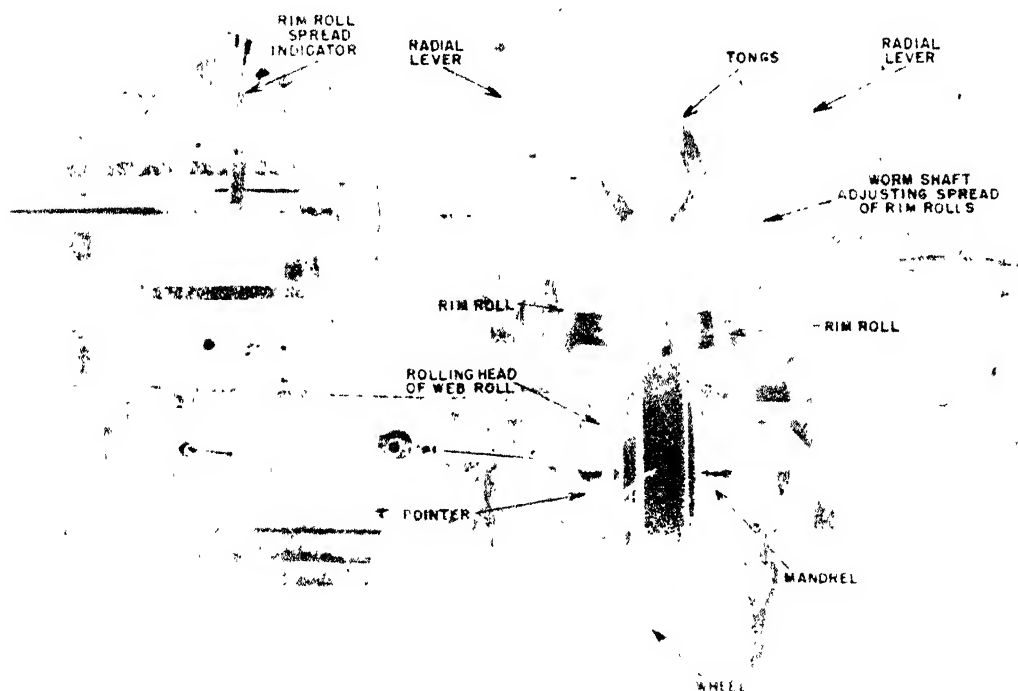


FIG. 31—9. Steel wheel being rolled in a vertical-type wheel mill.

on the web rolls to stall the mill. As to the orientation of these three rolls, they are so placed that their lines of contact with the wheel in rolling, and the axis of the wheel itself, all lie in the same horizontal plane. The friction driven rim rolls are located below the web rolls, so that all the rolls lie within an arc of 180 degrees of the circumference of the wheel being rolled. In mills using four rim rolls, one set is located above the web rolls. These rolls are approximately twelve inches in length and nine inches in diameter and are so placed that the projected axis of rotation of the roll on either side of the wheel intersects the axis of rotation of the wheel. They are mounted upon sliding frames attached to the front of the mill housings. These frames are moved by horizontal screws connected by vertical worm shafts and gears to a common shaft, which extends in front of and beneath the housings, and is operated by an electric motor set eight or ten feet to the right of the housings. In this way the spread of these rolls is made uniform. However, the bottom set of rim rolls, due to the manner of rolling, do nearly all the work. An indicator, mounted on the upper horizontal screw attached to the sliding frame on the right side of the mill, is in plain view of the operator, who is able, by this means, to read the spread of the rolls and thus control the width of the rim. These rolls may be so formed that they will roll the sides of the rim at a slight angle to the vertical, so that these surfaces will lie in parallel planes after the dishing, or coning process. Two shelves attached to the housings in front of the tread and web rolls and separated by a space a little greater than the thickness of the wheel at the hub, give support for the wheel, which is mounted on a loosely fitting mandrel during the rolling. This mandrel is provided with removable bearings, which rest, unattached, upon the shelves, thus leaving the wheel free to move forward after the rolls have gripped it.

The Rolling Process—After the forged blank has reached the proper temperature for rolling, it is removed from the furnace and carried to the mill by a charging and drawing machine, where it is gripped beneath the rim by tongs suspended from a small jib crane standing on the housing above the rolls, or by a floor crane especially designed for the purpose. The wheel, held vertically, is guided between the two supporting shelves, and the mandrel is inserted through the punched bore. Then the wheel, resting on the mandrel, is pushed back against the tread roll into position for rolling. The web and rim rolls are then brought to bear on the wheel, the latter rather lightly at first. The large driving motor is started, and the wheel is made to revolve by the action of the roll or rolls upon it. The web rolls, working upon both sides of the web and the under side of the rim, force the metal back into the grooves of the tread roll with considerable pressure, until this part of the wheel has reached the dimensions for which the mill is set, or the diameter desired, while the spread of the metal and the width of the rim is controlled by pressure applied to the rim rolls. The diameter of the wheel is ascertained by a gage, or caliper, one end of which is attached to the tread-roll housing, so that it is moved simultaneously with this roll, in the same direction and through the same space. The other end of the caliper projects in front of the mill, and is provided with a hinged arm or pointer, so that it may be raised out of the way for inserting the blank or removing the wheel. The end of this pointer is curved toward the mill at right angles to its length. At the beginning of the rolling, the roller lowers this pointer to rest on the left hand shelf, in which position its curved end extends toward the tread roll and is

opposite its line of contact with the wheel, the pointer having been adjusted so that the distance between its point and the tread roll is equal to the diameter of the wheel desired. The wheel increases in diameter during the rolling and moves forward on the loose sliding mandrel until a circle on the center of its tread comes in contact with this pointer, when the roller stops the mill and spreads the rolls for the release of the wheel. During the rolling, jets of water are directed against the surfaces being rolled to remove the scale and give a smooth finish to the wheel. In addition to the water, a salt jet is also directed against the tread. The actual rolling process requires about one minute, so that the maximum capacity of the mill is high.

Effect of the Rolling—All the work of the rolling is concentrated upon the outer part of the web, the rim, the flange and the tread, where the additional grain refinement due to rolling is most needed. This refinement is marked as is shown by the visible difference in the structure of the metal in the hub and in the tread. As machining the tread removes much of this fine-grained metal, it would appear that the wheel rolled to a finish should be superior in wearing properties to the machined wheel, on first run at least. However, where wheels must be perfectly circular, machining is necessary. Furthermore, although the most careful mill practice is employed in producing rolled-to-finish wheels, some of the wheels thus produced require machining in order to eliminate slight surface blemishes.

FINISHING WROUGHT-STEEL WHEELS

Stamping—After each wheel has been rolled, it is taken on a small buggy to a stamping press, where markings fully identifying it are stamped on the inside rim. Stamping practice, however, varies at different plants. For example, at Gary the stamping is done in the coning press. These markings include the brand, the month and year of manufacture, the serial and heat numbers, and occasionally other markings such as type of heat treatment.

Punching Web Holes—Although standard specifications generally omit this requirement, a few wheels are still ordered with holes punched in the plate, or web, and the mills are equipped to punch these holes when the wheels are so ordered. After the rolling and stamping, the wheel is taken on the buggy to a small press, where the web holes are punched when required. This press is fitted to punch either two or four holes, 1¼ inches in diameter, equally spaced on radii of 9, 10, 11, or 12 inches, the standard radii for all the different sizes of wheels. Later these holes are reamed to a diameter of 1¾ inches and have the edges rounded to a given radius.

Coning—If web holes are not required, the wheel is taken from the punching press, or from the rolling mill, to the coning press for dishing. In this operation the form of the web, or plate, is changed from that of a flat ring to that of a truncated cone sloping from the hub to the rim. This form of the plate is important for it serves to prevent the development of the high internal stresses that would be set up by the cooling of the wheel if the plate were left flat. Coning presses are of various designs, but essentially they consist of a set of two dies, the top one of which is attached to the ram of the press. In the generally preferred method of coning, the hub is held rigidly and coning accomplished by the use of top and bottom dies shaped to contact the outer diameter of the hub and the underside of the rim. The top die descends a fixed distance, exerting pressure on the underside of the rim until



FIG. 31—10. Wrought-steel wheel undergoing finish machining on a lathe of special design.

contact is made with the bottom die. It has been found that, with this method of coning, improved control of geometry of the wheel is obtained.

Controlled Cooling—After coning, the wheel is allowed to cool to a temperature below the critical range. It is then control cooled under proper conditions to 300° F. Various methods of controlling the cooling are in use and have been found to give satisfactory results. The requirement is that every part of the wheel must

be kept at the same temperature until the heaviest portion has cooled to 300° F or lower.

Inspection of wheels is very rigid and may be said to begin with the making of the steel and to continue through every step of manufacture until the wheel is ready for shipment. For example, the wheels are inspected after rolling and, when cold, each wheel is rolled to the inspection platform for the initial finished wheel inspection. This inspection covers surface defects,

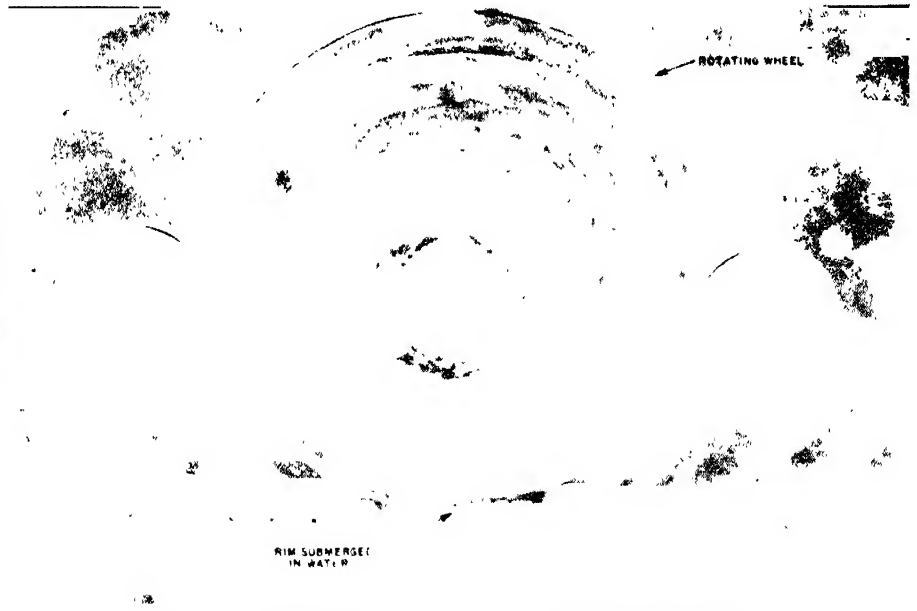


FIG. 31—11. Wheel undergoing quenching of the rim by being rotated in a special machine designed so that only the rim of the wheel contacts the quenching medium.

location of the hub, rotundity of tread, and the size, which is measured in standard tape sizes. These tapes are graduated in eighths of an inch, beginning with seven feet for a zero mark for wheels 28 inches and over in diameter, and with four feet for wheels under 28 inches in diameter. The surface defects consist principally of overfills, underfills, slivers, scale pits, and block marks, and as they are seldom deep, they may be removed by machining. The tape size is plainly marked on each wheel.

Machining and Final Inspection—After this preliminary inspection, the wheels are machined as required to meet the specifications or to remove the defects. On rolled-to-finish wheels, the machining consists of rough boring, while the outer part of the plate, the rim, tread and flange remaining as rolled and the hub as forged. The lathes used for machining wheels are massive and include many ingenious features of design, as indicated by Figure 31—10. The wheels are then rolled back to the platforms for final inspection, which is even more rigid than the first. In this inspection the wheels are tested for size, eccentricity and size of bore, position and size of hub, thickness and height of flange, radius of throat, thickness of rim, coning, rotundity, and soundness. After being re-stenciled with tape size and marks requested by the customer, such wheels as come within the allowable tolerances are mated and sent to the shipping platform.

Machining includes either rough machining or rough and finished machining operations. Extra machining not covered by standard specifications includes:

1. **Finish boring** covers machining of the hub bore to a smooth finish and to permissible variations in dimensions, so that the wheel may be mounted on the axle.

2. **Machining rim and outside of hub** consists of machining the back and face of the rim, the tapered surfaces on the diameter of the hub and on the inside diameter of the rim and the fillets joining these surfaces to the web, or plate, of the wheel.

3. **Finish machining hub faces** refers to finish machine work on a hub face when a hub projection or depression is machined to closer tolerances than required by standard specifications.

4. **Machined all over** is applied to a wheel the dimensions of which must be close to the nominal dimensions and the weight of which must be distributed symmetrically about the axis. All surfaces of the hub, the plate and the rim are machined.

5. **Hub recessing** consists of counterboring or turning a recess in the face of the hub for installation of a bearing plate to prevent hub wear from lateral pressure.

6. **Turning outside of hub** is performed on wheels for electric railways, and consists of turning the end portion of the outside hub to a cylindrical section for a gear casting seat.

HEAT TREATMENTS FOR WROUGHT-STEEL WHEELS

In addition to the controlled cooling mentioned above, three methods of heat treating wheels have been developed since 1925, designated as **normalizing**, **rim toughening**, and **oil quenching and tempering**.

In **normalizing**, the rolled wheels are permitted to cool to a temperature well below the critical range, and then are placed in a continuous reheating furnace and heated to a temperature above the critical range to refine the grain structure. Upon being withdrawn from this furnace, the wheels are cooled in air to a temperature below the critical range, approximately 950° F, and are then control cooled to 300° F or below. This treatment develops a uniform grain structure in all parts of the wheel, prevents formation of internal defects, and raises the ductility markedly. In this case, they are heated very gradually to above the upper critical temperature and then cooled as described.

For **rim toughening**, the wheels are cooled and heated as for normalizing, but upon removal from the furnace they are immediately placed in a special quenching machine (Figure 31—11) and rotated at a uniform rate with part of the rim submerged in water. This method prevents the water from coming in contact with either the plate or the hub of the wheel. After the rim has been quenched in this manner for a given period, the optimum duration of which has been predetermined, the wheel is removed from the quenching machine, transferred to a furnace and heated to a proper tempering or drawing temperature to remove internal stresses and obtain the hardness desired. The wheel is then placed in pits and control cooled to 300° F or below. This treatment leaves the hub and the plate in a normalized condition, and increases the hardness of the wearing zone of the rim from the normal Brinell hardness number of about 260 to approximately 340.

To **oil quench and temper** the wheels, they are heated as described above, withdrawn at a temperature slightly above the critical range of the steel, and totally immersed in oil contained in a tank constructed to permit the wheels to be rotated for the period of the quench. The wheels are then placed in a reheating furnace, heated to the required tempering temperature, and control cooled to 300° F or below. This treatment refines the grain structure and toughens equally all parts of the wheel, developing the hardness to about the same extent as in the rim-toughening treatment.

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Specifications M-103 and M-107; Association of American Railroads.

Chapter 32

PRODUCTION OF RAILROAD AXLES

Axles are important because of their functions and the number in service. They form vital parts of every kind of vehicle and vary in size from a few ounces to several thousand pounds. The larger sizes manufactured by the United States Steel Corporation include axles for railroad freight and passenger cars, Diesel locomotives, locomotive tenders, engine trucks, trailers, electric motor trucks, electric street cars, and driving axles. The Homestead District Works is cited as an example of a plant manufacturing these larger axles, although they are also produced at Gary Steel Works of United States Steel.

Composition and Heat Treatments—As to composition, axles are generally made of carbon steels of various grades, ranging from 0.40 to 0.55 per cent carbon and 0.60 to 0.90 per cent manganese, with phosphorus and sulphur each under 0.05 per cent.

They may be ordered "as forged;" annealed; normalized; normalized and tempered; quenched and tempered; or normalized, quenched and tempered, the quenching being in either oil or water according to the size of axle, the kind of steel, and the mechanical properties it is desired to develop.

THE AXLE WORKS

The essential equipment of the plant includes continuous gas-fired furnaces for heating the blooms; double-acting steam forging hammers; manipulator; gag press straighteners; double cutting-off, facing and centering machines; as well as rough-turning and finishing lathes, boring lathe, hollow drill machine, and a complete heat-treating plant. Maximum weight of forgings that can be produced is 6000 pounds; maximum length, twelve feet; maximum diameter, sixteen inches; minimum diameter, three inches.

As to arrangement, the layout of the plant provides for the most economical handling of the materials. The blooms start at one end of the plant and continue in one direction, progressing step by step through the various operations, until arrival at the other end of the plant where they are in a form ready for shipment.

METHODS OF FORMING AXLES

In general, axles are formed by a combination of rolling and forging operations, and the practices of different plants vary somewhat. Thus, with a suitable mill properly equipped for the work and located near the forge shop, it is possible, by starting with an ingot of proper size, to roll the bloom and complete the forging in one heating. The axle bloom is rolled either round or square and the forming is completed by forging. In other plants, where it is necessary to start with cold steel after it has been rolled into blooms (which must correspond in dimensions and weight to the size of the axles for which it is intended), the various steps in the process of manufacture are as follows:

Inspection of the Blooms—The blooms are subjected to a rigid inspection before the steel is shipped to the axle works. Those blooms that show any signs of pipe

or insufficient discard at the blooming mill shears are rejected. Surface defects, such as seams, slivers, and surface cracks, are eliminated by surface-conditioning treatments. Blooms passing this inspection are shipped to the axle works, where they are stored under cover until needed.

Heating the Blooms—Proper heating of the blooms for forging requires that they be uniformly heated throughout and brought gradually to the forging temperature, which should be kept as low as possible and yet permit the work to be done. The advantages of a low finishing temperature in obtaining maximum grain refinement were discussed in Chapter 19, under "Hot Working." The importance of heating slowly is obvious; rapid heating may cause the outside of the bloom to become somewhat hotter than the core, which condition would prevent uniformity in grain structure and in the flow of the metal during the forging operations. In addition, high internal stresses would be developed. Slow heating gives the heat a chance to "soak" into the bloom and thus produce that uniformity in temperature from center to surface so necessary to secure a finished forging of the best quality. Reheating furnaces of the continuous type are used, and these are equipped with recording pyrometers. In addition, the temperature of the steel is periodically checked with an optical pyrometer as the bloom is drawn from the furnace. With this type of furnace, the bloom is placed in the furnace at the cold end and is slowly pushed or rolled toward the hot end. The bloom reaches a full forging temperature only a short time before it is drawn from the furnace.

The Forging Operation—Having been brought to the proper temperature for forging, the blooms, within a certain range of sizes, are pushed out of the hot end of the heating furnaces upon a conveyor, which serves one furnace, and are carried by it to a roll table that distributes the blooms to five of the plant's seven hammers arranged in a row on one side of the table. Square blooms pushed out of the second reheating furnace are delivered to the other two hammers by the manipulator, as shown in Figure 32—1, and forming begun as described below with the exception that all manipulation of the forging is performed on one side of the hammer only. Adjustable deflecting rails built in the side guards of the roll table serve to divert the blooms to small receiving tables, of which there is one for each hammer. This leaves the blooms in positions to be most conveniently grasped by the hammer tongs, which are suspended from cranes.

The tongs having been quickly clamped on, the bloom is swung around between the forging dies of the hammer (Figure 32—2). These dies are provided, when desired, with one or more grooves; one, the plain groove used to do the greater part of the forging, is located directly under the piston rod, while the other grooves, used to form special sections, such as the journals, are placed beside the plain groove. The forging is begun at the middle of the bloom, which is rapidly reduced by heavy blows of the hammer, the forging progressing toward

FIG. 32-1. Axle being forged from a heated square bloom on a hammer equipped with a manipulator. One end of the forging (that gripped by the manipulator tongs) has been completed, the bloom has been turned, and the opposite end will now be worked.



the free end of the bloom. Here, the journal or other special section is formed by the special grooves in the die in a few blows. The piece is again placed in the plain groove, and the forging is smoothed up and brought, by light blows of the hammer, to correct diameter, which is determined by caliper. The tup is then brought to rest upon the axle, which is held between the dies while the

tongs are released, and tongs on the opposite side of the hammer are made fast to the finished end. The other end of the axle is then forged down like the first, except that, in addition to diameter, the length is also fixed. The crane is then swung around, and the axle is placed on the hot bed, which is supported about three feet above the floor by two rails.

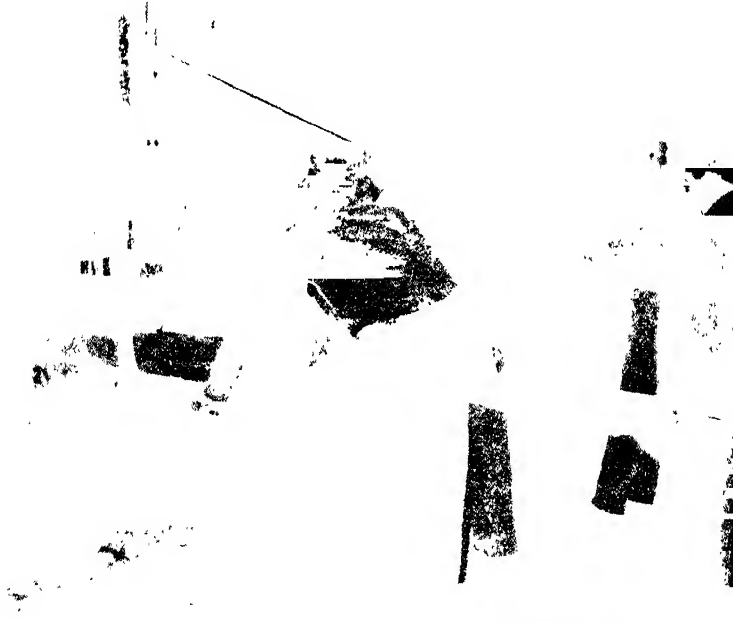


FIG. 32-2. Forging an axle on a hammer where the manipulation of the bloom is performed manually. When one end has been forged, the bloom is gripped by tongs on the opposite side of the hammer and forging is completed without reversing the bloom.

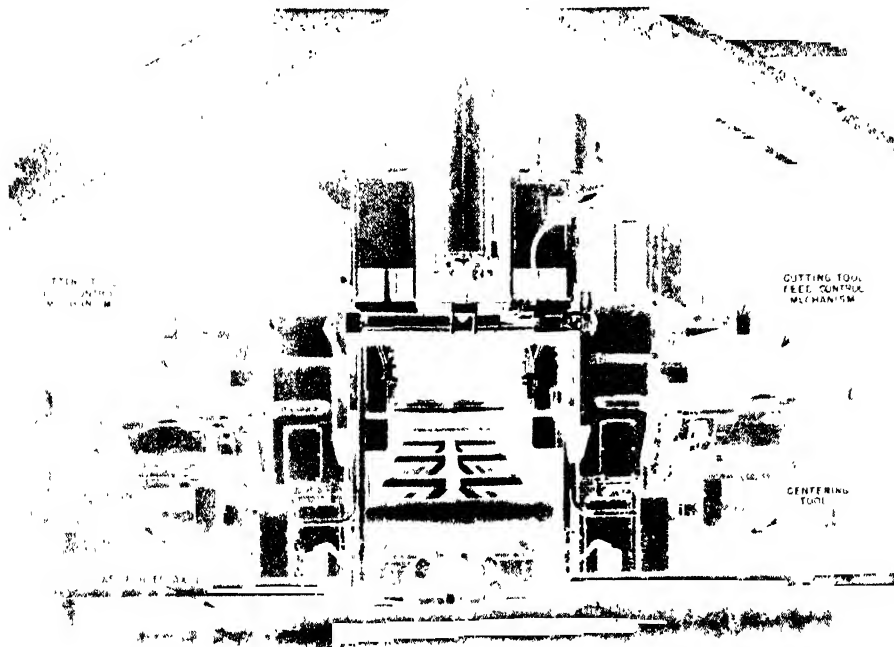


FIG. 32-3. General view of a special lathe, known as a double combination cutting-off, facing and centering machine, for machining forged axles.

FINISHING PROCESSES FOR AXLES

Straightening—Except in the case of heat-treated axles and driving and trailing axles, the next step after the forging is straightening, which is accomplished in gag presses while the axles are still hot. From the hot beds, the forgings are carried by overhead cranes to similar beds in front of the presses. Here each axle is inspected for straightness, and those which require it are straightened before their temperature drops below 950° F. At this temperature, the axles are placed on cradles and put into pits for controlled cooling down to approximately 300° F. Heat-treated axles are straightened after being treated, but driving and trailing axles which are too large to be straightened by the gag press, are straightened under the hammer when necessary. Bored axles are straightened before they are bored, which operation precedes the heat treatment.

Cutting-off and Centering—After removal from the controlled-cooling pits, the forgings are moved forward by overhead cranes and distributed to the cutting-off lathes. These lathes are double combination cutting-off, facing and centering machines (Figure 32-3) and are designed to work on both ends of the forging at the same time. Upon being inserted in this machine, the forging is grasped at the wheel seats by adjustable revolving centering clamps, which hold it firmly to the central axis of rotation. Cutting tools adjusted to the correct length then are brought to bear and cut off the excess metal at the ends. Ends are faced in this operation, a tolerance of one-eighth inch over length and nothing under being permitted. When these tools have cut to within about $\frac{1}{16}$ inch of the center the crops drop off and, with the forging still held by the centering clamps, the revolving centering tools are brought to bear at each end. These tools are shaped to cut a 60-degree cone-shaped centering hole (to the dimensions shown in Figure 32-4) with a five-sixteenths inch in diameter clearance hole for points at the bottom of the centering hole. When axles are ordered to be smooth-forged only, the operation of cutting-off and centering completes the work done by the mill.

Rough turning before shipment is of decided advantage to both the shop and the customer, because certain flaws can be detected only after rough turning, and a considerable saving can be effected in handling and transportation by holding excess weight to a minimum. Lathes of the type shown in Figures 32-5 and 32-6 are used for this work. Rough turned material falls into two classes, known as "rough turned on journals and wheel seats," and "rough turned all over." Axles of the first class are put into service with the center portions between the wheel seats as forged. In the case of axles rough turned all over, the center portions are forged slightly over size to provide for the metal removed in

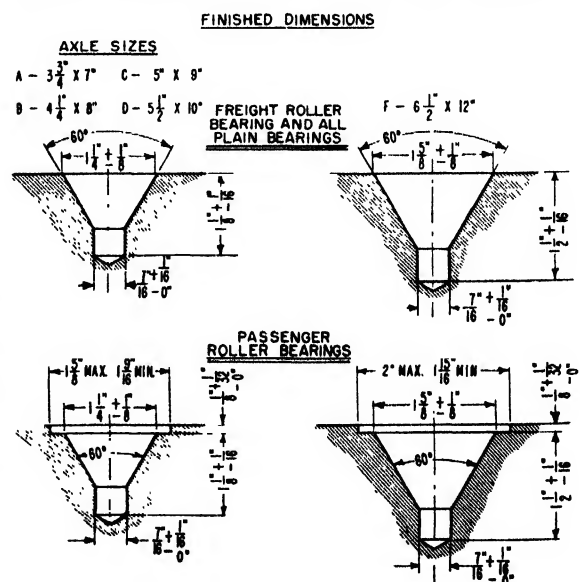


FIG. 32-4. Standard dimensions of centering holes cut in the ends of axles in United States Steel Corporation plants.



FIG. 32-5. Lathe of the type used for rough turning only journals and wheel seats of forged axles, after cutting-off, facing and centering (Figure 32-3).

turning to size; in such case, the center portion is furnished with body finish when no further machine work is to be done by the customer. In the case of car axles or other axles with a tapered body, this metal is removed at the same time (or after) the journals and wheel seats are rough turned in a lathe (Figure 32-6). In finishing rough turned axles, the wheel seats are finish-turned only, while the dust guards, journals and collars are finish-turned and burnished, and in order to provide the excess metal required for this work, these parts are rough turned one-eighth inch over size on their diameters.

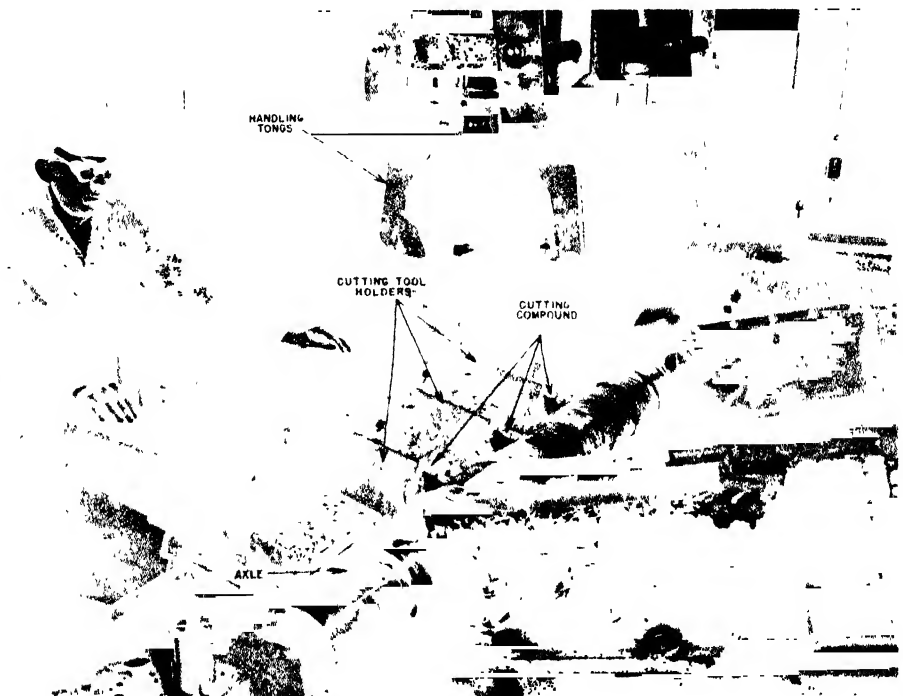
Boring—A hollow forging permits a more efficient transfer of heat in both the heating and cooling cycles

of heat treatment than is possible with a solid forging, and this results in a more uniform grain structure. The practice of boring forgings longitudinally through the center—adopted particularly for those of large diameter to be subjected to a quenching operation—is also effective to a degree in other heat-treatment procedures.

Boring has other advantages. It eliminates any uncertainty about traces of piping or segregation that may have escaped detection by discard or close inspection; and a considerable reduction in weight may be made with no appreciable loss in effective strength of the forging.

Boring is desirable for all sizes larger than 7 inches in diameter. The diameter of the bore should be equal to

FIG. 32-6. Lathe of the type used for rough machining of the entire surface of forged axles after cutting-off, facing and centering in the machine shown in Figure 32-3.



at least 20 per cent of the minimum diameter of the forging.

Boring consists in a single cutting operation which may leave tool marks or slight ridges on the wall. When the boring is from both ends of the forging, the junction of the bores need not be concentric. Standard tolerance for boring is minus $\frac{1}{16}$ inch minimum, and plus $\frac{1}{8}$ inch maximum. Bored axles often are reamed and the ends plugged and centered to permit finish machining.

AXLE HEAT-TREATING PLANT

The heat-treating plant is housed in the same building with the hammers and lathes and consists of furnaces for heating forgings, a quenching tank containing water, a tank holding quenching oil, and all the necessary supplemental equipment for handling and testing the material.

Heat-Treating Furnaces—The inside working space of each of the furnaces is twenty-one feet in length and eight and one-half feet in width, and is designed to heat forgings uniformly. The furnaces are provided with removable bottoms of the car type, which facilitate the charging and drawing operations. This bottom is moved into and out of the furnace by a toothed rack attached to the bottom of the car and a stationary pinion actuated by an electric motor, the car itself resting on rollers that move over a double track. The doors of the furnace are of the vertically lifting type, and are hydraulically operated. These features, together with the close proximity of the quenching tanks, permit the drawing and quenching of a charge in the least possible time, less than a minute being required to transfer a charge from the closed furnace to either of the quenching tanks. The measures taken to obtain uniform heating are particularly noticeable in this furnace. The furnace is of the bottom cross-fired type, in which gas is employed as fuel, and is heated by burners firing into passageways at spaced intervals in the car bottom, thus permitting the temperature at any point in the furnace to be controlled closely. The furnaces have sprung-arch roofs, arched from side to side. In order that the entire surface of the material may be exposed to heat of the same intensity, the charge is supported at a height of about eighteen inches above the floor of the car bottom by two stools that extend the entire length of the car. These stools are spaced about four feet apart. The floor of the car bottom is constructed of brick laid upon a bottom of

steel plates, and is of a thickness that will give the plates ample insulation from the heat of the furnace. The bottom is made to fit the furnace neatly, and the escape of hot gases from the heating chamber is prevented by sand seals. Means are provided for taking the temperature of the charge at regular intervals during the heating. Recording pyrometers are used throughout the heating cycle.

Advantages of Heat Treating Axles—The proper heat treating of axles, because of their size, is accomplished with some difficulty. However, with proper equipment, care, and judgment, decided advantages result therefrom. It is the only way in which the grain structure of the as-forged axle can be refined and made uniform and, in doing this, variations in the as-forged grain structure which result from the heating and working of the bloom are overcome. Stresses remaining in the piece after forging also are eliminated. The principle advantage is the control of and improvement in mechanical properties effected through correct heat treatment. For example, annealing refines the grain and develops the maximum ductility of the steel; normalizing refines the grain of the as-forged axle and increases both its strength and ductility; and quenching and tempering refine the grain to give maximum control over the final mechanical properties. The latter treatment offers the only positive means of markedly increasing the strength as well as the torsional and shock-resisting properties that are very desirable under modern conditions of road traffic.

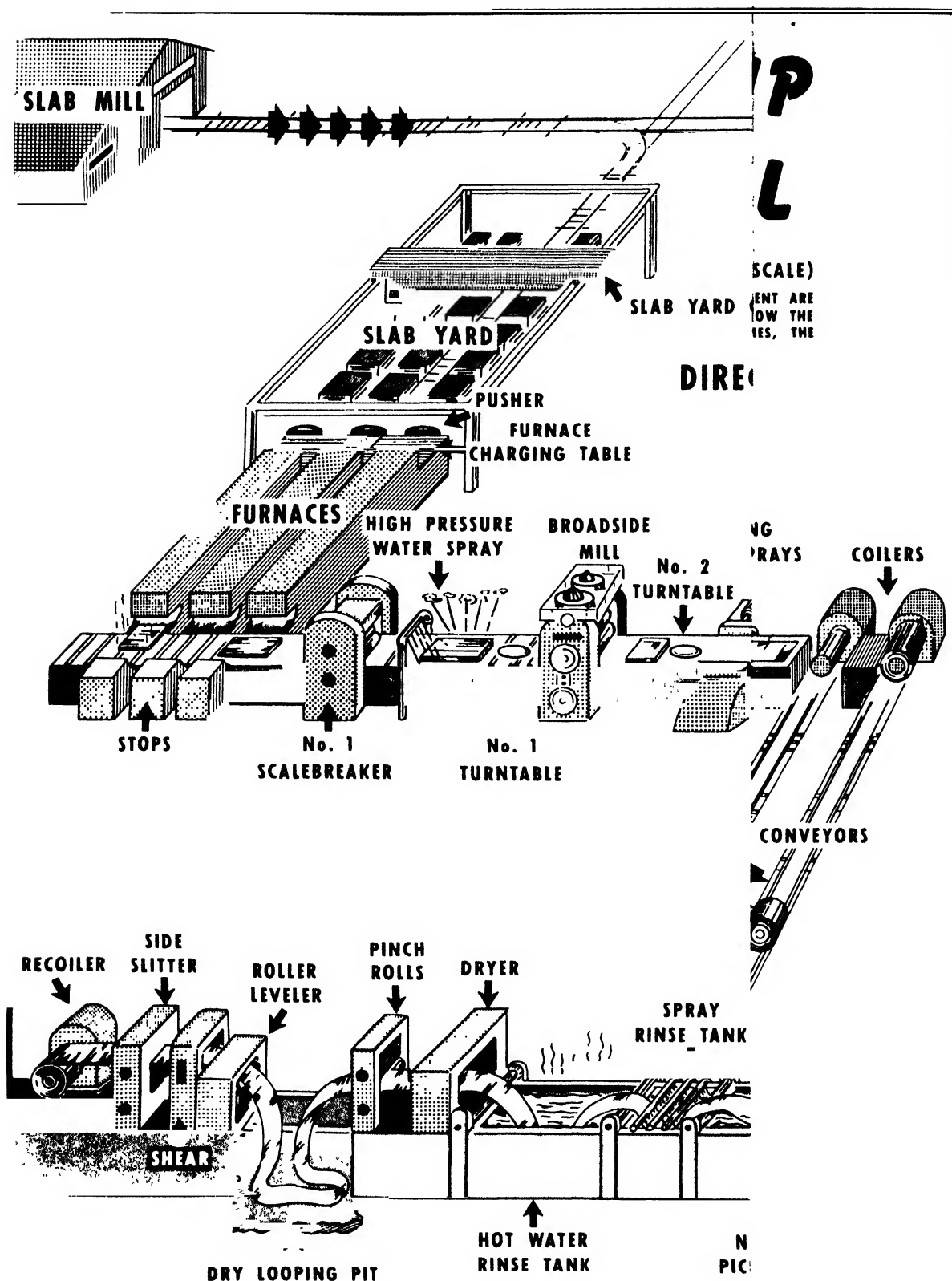
Testing equipment includes all the latest devices for testing materials. In the shop, hollow drill machines are provided for cutting out tests. A drop-testing machine, adapted for testing axles in accordance with standard specifications, is provided. Other mechanical tests are made in the physical laboratory, which is equipped with all the necessary appliances for accurate testing in accordance with standardized methods.

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Chapter 33

THE MANUFACTURE OF HOT-STRIP MILL PRODUCTS

SECTION 1

HISTORICAL DEVELOPMENT

The mills used for rolling flat-rolled steel products include the following, named in the order of their development:

- (1) The two-high mill for hot rolling sheets in packs.
- (2) The two-high mill for rolling sheared plates.
- (3) The universal mill for rolling plates.
- (4) The three-high or Lauth mill for rolling sheared plates.
- (5) The continuous or tandem hot-strip mill for rolling sheets, strip and hot-rolled breakdowns for cold reduction in coils.
- (6) The cold-reduction (cold-rolling) mill for sheets and strip.
- (7) The continuous sheared plate mill.

The plate mills mentioned above are described in Chapter 27.

The method for rolling sheets on two-high, single-stand mills (described in detail in a later part of this chapter) originated between 1720 and 1728 and antedates all other methods for rolling iron and steel. Up to about 1890, the finished flat-rolled steel products could be classed as sheets, plates and bars, although many thin products, such as pipe skelp, were rolled on the merchant-bar mills. The bar mills also rolled thinner sections, and about this time there developed such a demand for very thin flats that special mills were built to supply the material. The differentiation first took place in the narrower widths, ranging from $\frac{3}{8}$ inch to 3 inches, and from 0.065 to 0.028 inch in thickness. The higher grades of this material were designated as hoop; a common grade was known as cotton tie. In 1890, bands 14 $\frac{1}{2}$ inches by 0.14 inch were rolled at Warren, Ohio and, in 1892, a mill designed to roll sheets in tandem rolls was built in Austria. In 1893, a mill at Bridgeport, Connecticut produced thin sections up to 7 inches in width and, in 1895, a semi-continuous mill was built

there to produce thin hot-rolled products up to 10 inches wide. In rolling on these first mills it was found that the limits as to width and thickness bore a certain relation to each other. In the Bridgeport mill, the ratios of width to thickness of product varied between 100 and 160. From these beginnings, the widths within the same range of thicknesses were increased at intervals by various producing mills until, in 1920, steel 22 inches wide and 0.105 inches thick was rolled successfully at Weirton, West Virginia. Some of the product of the successful strip mills subsequently was pickled and cold rolled, when it was designated as **cold-rolled strip steel**, but much of it was used as rolled and was known as **bands**, **band steel**, or **hot-rolled strip**. In 1920, therefore, flat-rolled products were classed commercially as **sheets**, **plates**, **bars**, **bands** and **hoop**, although the use of the term "strip" commonly was applied to the light products, which were somewhat narrower than the heavier sheets rolled on pack mills.

Up to 1920, it was customary to observe a certain width-to-thickness ratio in the hot rolling of strip, and the maximum ratio for successful hot rolling was considered at that time to be 250. In 1923, however, a continuous hot-rolling mill at Ashland, Kentucky, began rolling much wider strip, the widest and thinnest being 36 inches by 0.065 inch. This mill can be considered the forerunner of the modern continuous wide hot-strip mills described later in this chapter.

In 1926, at Butler, Pennsylvania, the first mill was built to combine successfully use of the following principles: (a) four-high finishing stands; (b) control of the direction of travel of steel through the pass line of the tandem finishing mills by progressively decreasing the product crown in successive mill passes, and (c) hot-coiling equipment at the discharge end of the mill. This installation was the first of the modern wide continuous hot-strip mills as known today.

SECTION 2

CLASSIFICATION OF FLAT-ROLLED STEEL PRODUCTS

About half of the rolled steel products now made in the United States may be classed as flat-rolled material. Flat-rolled steel products (including sheets, strip, tin plate, black plate, flat bars, slabs, plates, skelp and hoop) may be distinguished from other forms of rolled steel in two general ways. First, flat-rolled steel is produced on rolls with smooth faces in contrast with the cut or grooved roll faces employed in the manufacture of shapes and, second, in flat-rolled products the ratio of

width to thickness is generally high as distinguished from other rolled products. The ranges of dimensions are wide, varying in thickness from 0.005 inch in light strip to 15 inches in heavy plates, and in width from $\frac{3}{16}$ inch in narrow strip to 204 inches in wide plates.

Sheets, strip and tin plate comprise about three-fourths of the total tonnage of all flat-rolled steel products. Total shipments of these commodities from mills in the United States in 1953 was as follows:

Hot-Rolled Sheet and Strip.....	9,964,071 net tons
Cold-Rolled Sheets and Strip, Galvanized Sheets, Long Ternes and Enameling Sheets.....	16,214,232 net tons
Tin, Terne and Black Plate.....	5,410,427 net tons
Total	31,588,730 net tons

Flat-rolled steel products fall into two major categories: hot rolled, and cold rolled. Hot-rolled products are reduced to final thickness by heating and rolling at elevated temperature. Hot rolling usually is conducted at temperatures between 2200° and 1200° F, except in the case of pack-rolled sheets for some purposes where the finishing temperatures may be as low as 900° F. In ordinary practice, virtually all hot rolling is conducted well above 1300° F, the "lower critical temperature" of plain carbon steel. Cold-rolled products are really "cold finished" products, since much of the reduction from ingot to final thickness is, of course, done while the product is hot, in a manner similar to that employed for hot-rolled products. Cold rolling is carried out on products which have not been heated immediately prior to the cold-rolling operation in which they are reduced to final thickness. However, the temperature of the steel is raised due to frictional effects of rolling. The temperature of the steel in coils immediately after cold reduction has been measured, and coil temperatures ranging from 250° to 450° F have been recorded: the temperature of the steel in the actual nip of the rolls is probably higher than this, but is quickly lowered by the coolants used in rolling. The important distinction is that cold-rolled products receive enough cold reduction in the final rolling operation to affect the surface and mechanical properties of the finished product.

The list of flat-rolled steel products includes both semi-finished and finished materials. Among the semi-finished products which have already been discussed are slabs (Chapter 25) and flat bars (Chapter 30). An important semi-finished flat-rolled product of the hot-strip mill, which will be discussed in this chapter, is hot-rolled breakdowns for cold reduction, in coils.

Flat Hot-Rolled Finished Products Classified—The chief hot-rolled finished flat products are divided into four major groups, namely, bars, plate, hot-rolled strip, and hot-rolled sheets. Dimensions, particularly thickness and width, are the principal bases of classification, as shown in Table 33—I.

Table 33—I. Product Classification by Size of Flat Hot-Rolled Carbon-Steel Products.

Width (Inches)	Thickness (Inches)				
	0.2300 and thicker	0.2299 to 0.2031	0.2030 to 0.1800	0.1799 to 0.0568	0.0567 to 0.0449
To 3½, incl.	Bar	Bar	Strip	Strip	Strip
Over 3½ to 6, incl.	Bar	Bar	Strip	Strip	Strip
Over 6 to 12, incl.	Plate	Strip	Strip	Strip	Strip
Over 12 to 48, incl.	Plate	Sheet	Sheet	Sheet	Sheet
Over 48	Plate	Plate	Plate	Sheet	Sheet

Table 33—II. Product Classification By Size of Flat Cold-Rolled Carbon Steel

Width (Inches)	Thickness, Inch		
	0.2500 and thicker	0.2499 to 0.0142	0.0141 and thinner
To 12, incl.	Bar	Strip ^{1, 2}	Strip ¹
2 to 12, incl.	Bar	Sheet ³	Strip
Over 12 to 23-15/16, incl.	Strip ³	Strip ³	Strip
Over 12 to 23-15/16, incl.	Sheet ⁴	Sheet ⁴	Black Plate ⁴
Over 23-15/16	Sheet	Sheet	Black Plate

¹ When the width is greater than the thickness with a maximum width of ½ inch and a cross-sectional area not exceeding 0.05 sq. in., and the material has rolled or prepared edges, it is classified as flat wire.

² When a particular temper as defined in A.S.T.M. specification A109, or a special edge, or special finish is specified, or when single-strand rolling is specified in widths under 24 inches.

³ Cold-rolled sheet coils and cut lengths, slit from wider coils with No. 3 edge (only) and in thicknesses 0.0142 inch to 0.0821 inch incl., carbon 0.20 per cent maximum.

⁴ When no special temper, edge or finish (other than Dull or Luster) is specified, or when single-strand rolling widths under 24 inches is not specified or required.

Flat Cold-Rolled Products Classified—The chief cold-rolled products, all classified as "finished," are divided into four major groups, namely, bars, strip, sheets and black plate. The dimensional bases for differentiating between these commodities are shown in Table 33—II.

Plates are flat, hot-rolled, finished steel products, rolled either direct from ingots or from slabs, and are defined further in terms of width and thickness as shown in Table 33—I. Sheared plates are rolled in widths up to 204 inches and are sheared on all edges to the widths and lengths desired. Universal-mill plates are rolled to all widths up to 60 inches, and sheared on the ends to the lengths desired. Plates vary widely in composition, and are ordered to specifications based on chemical composition, mechanical properties, or both.

Bars are hot-rolled, finished steel products having various cross-sectional forms, including small shapes. Flat bars are rectangular in section, and are 1¾ (0.2031) inch and over in thickness and 6 inches and under in width. The larger sizes may be sheared hot or sawed, but the smaller sizes generally are sheared cold to length. The edges may be finished practically square or rounded, and the narrower sides of the thinner sections may be flat, convex, or slightly concave. As to chemical composition, practically all grades and kinds of steel are rolled into bars.

Hoop, as already indicated, was formerly an important commodity and was made in various widths and thicknesses ranging from about ⅝ inch to 6 inches in width, and from about 0.109 inch to about 0.035 inch in thickness. The narrower widths, made of low-carbon plain steel and used to hold the staves of barrels, casks, etc., together, were subjected to special forming operations after rolling, but the material applied to other purposes varied greatly in composition, and much of it required special care in finishing and handling.

Strip, Sheets, and Black Plate are classified according to dimensions, as shown in Tables 33—I and 33—II. Continuous mills roll product in very long lengths, ranging in general from 100 feet to 2000 feet or longer according to the width and thickness of the strip and the size and equipment of the mill. The products are either cut to specified lengths after rolling, or utilized in coiled form.

The distinction between the hot-rolled and the cold-rolled classes of these commodities lies, as mentioned above, in the methods used to attain finished thickness. The cold-reduction process, however, applied to the hot-rolled and pickled steel imparts, after proper heat treating and finishing operations, greatly superior surface and mechanical properties to the hot-rolled coun-

terpart of each commodity. The starting material for the cold-reduction process consists of the semi-finished product of the hot-strip mill designated as hot-rolled breakdowns in coil form.

Further sub-classification of sheets, strip and black plate is necessary to approach an understanding of the diversity of characteristics which enables steel in these forms to be applied to so many important uses. As one example, most black plate actually is not used as such, but is coated with tin to produce tin plate of many varieties for many uses, including the common "tin" can. Such group subdivisions are based on steel type, product treatment, characteristic properties, and final use, and will be discussed after description of the manufacturing methods used.

SECTION 3

SOURCES AND TYPES OF STEEL FOR SHEETS, STRIP AND TIN PLATE

Chemical Compositions—Steel compositions used for the manufacture of thin, flat steel products range from so-called "pure iron," in which the sum of all elements other than iron in the product is less than one-third of one per cent of the total weight, to the high-alloy stainless and heat-resisting steels composed of as much as 50 per cent alloying additions. About four-fifths of the sheet, strip and tin plate tonnage rolled, however, is made from steel compositions within the following ranges (based on ladle analyses):

Element	Per Cent
Carbon	0.03 to 0.12
Manganese	0.20 to 0.60
Phosphorus	0.04 maximum
Other elements	Low as possible

This general range of compositions provides the best combination of rollability during manufacture and formability in most of the applications for which these products are used. Such compositions, too, are well suited for the production of rimmed steel, which is preferred for flat products because of the superiority of its ingot surface. Deviations from this basic composition range are deliberately employed to obtain specific desired properties in the steel, according to principles discussed later. Within the basic composition range, however, most steel plants further subdivide the indicated ranges for individual elements to fit particular production conditions or consumers' special needs. The end result of such adjustments of composition will differ from plant to plant; accordingly, most consumers' requirements are expressed best in terms of suitability for particular applications or of desired properties, with composition restricted only where a direct relationship between composition and performance is known.

Sulphur, silicon, copper, nickel and chromium generally are considered as the "other elements" of the basic composition given above. Except in the steels where they are added deliberately to produce alloy steels with definite properties, these elements offer no advantages and, when present in greater than certain amounts may even be detrimental to the rolling or fabricating properties of steels for sheets, strip and tin plate. An effort is made to keep sulphur and chromium contents each below 0.05 per cent, and copper and nickel contents below 0.15 per cent. Silicon content naturally falls under 0.02 per cent in the rimmed and capped steels popularly used for sheets, strip and tin plate, but is present in amounts up to 0.15 per cent when this element alone is

used as the deoxidizing agent in the manufacture of steel in the range of the basic low-carbon composition. Other elements seldom are found in undesirable amounts, although unusual local conditions affecting the scrap or ore supply may result in the presence of enough molybdenum or tin or both, in the steel to cause it to be somewhat harder in the finished condition than would be the case if these elements were absent or present in only very small amounts.

Steelmaking Processes—The steel for sheets, strip and tin plate is made in this country by the open-hearth, Bessemer or electric-furnace process, each being used where it is best suited to produce steel having the desired composition and properties. Bessemer steel, for example, inherently contains larger amounts of phosphorus (totaling about 0.10 per cent) and nitrogen (about 0.015 per cent total) than open-hearth steel of otherwise similar composition; the additional stiffness imparted to certain products by the presence of these elements in the steel causes Bessemer steel to be selected for such limited applications. Stainless and some other alloy steels are melted in the electric arc or induction furnace for conversion to sheet and strip, the processes in these cases being chosen for their ability to produce alloyed grades of steel with minimum loss of valuable alloying elements in melting and finishing the steel in the furnace. However, about 95 per cent of the steel for sheet, strip and tin plate is produced in the basic open-hearth furnace, for the following reasons.

The basic open-hearth process provides the most economical means for the utilization of scrap, pig iron and Bessemer blown metal, and for the positive control of phosphorus content of the steel by permitting its removal to the desired degree, while producing steels of suitable composition and properties for most flat-rolled products. Furnace charges vary from the duplex and "liquid metal" processes in which no scrap is used, to the opposite extreme of charging up to about 75 per cent cold scrap. Melting practices vary widely to produce steels which, although they may be made by different practices, possess closely similar characteristics when rolled into end products.

Rimmed, capped, semi-killed and killed steels all are used for conversion to thin, flat steel products. Rimmed steels comprise more than half of the sheet, strip and tin plate tonnage made, since steel of the basic low-carbon composition given above, when properly refined, tapped and teemed, provides a naturally rimming steel that can be cast into ingots with sound surfaces and possesses a high degree of cleanliness and ductility.

Mechanically-capped steel retains most of the surface qualities of rimmed steel and provides more uniformity of hardness throughout the cross-sections of rolled products, while increasing the yield of sound steel obtained from each ingot. This modification of rimmed steel by mechanical capping is, therefore, of importance in producing steel for such an application as tin plate where a controlled degree of uniform stiffness is desirable in the end product. Aluminum capping or top killing also is employed; in this practice, the rimming action is stopped after having progressed to the desired point by adding aluminum to the molten steel in the top of the mold. The aluminum killed ("special killed" or "fine-grained") extra-deep-drawing steel is a highly specialized modification of the low-carbon type, having virtual freedom from age-hardening and, hence, unique suitability for some types of drawing operations.

Rimmed, capped and even the "special killed" low-carbon steels are cast into ingot molds without hot tops. "Semi-killed" grades, usually made to possess somewhat higher carbon and manganese contents, still allow sufficient deoxidation control (with aluminum or silicon) of gas evolution and shrinkage to provide a sound ingot from an open-topped mold. Steels having high contents of carbon or alloying elements, those fully killed to attain certain desired end properties, and occasionally the "special killed" grade, are hot-topped to eliminate the formation of "pipe" resulting from the shrinkage characteristic of the solidification of such grades. The stainless steels are the best known hot-topped steels converted to thin flat-rolled products.

Slabs—Slabs are the raw material for the modern continuous hot-strip mill. A slab is defined as a rectangular steel section having a minimum thickness of 1½ inches and minimum width not less than twice the thickness. Slabs are generally provided in thicknesses of 2 to 7 inches, widths of 12 to 64 inches, and lengths of 60 to 240 inches, depending on strip-mill requirements. They must be accurate enough in dimensions and sound enough in structure to permit conversion in subsequent rolling operations with a minimum of difficulty, and their edges and surfaces must be free of injurious defects which would carry through to the finished product.

Two methods are practiced in converting steel from ingot form to slab and then to hot-rolled sheets, strip or breakdowns for cold reduction. By one procedure, the ingots may be heated in soaking pits and rolled on a blooming or slabbing mill to slabs of the required width and thickness, then sheared to length and immediately passed along to the hot-strip mill for final reduction to desired thickness, without reheating. The second and most generally used method is similar to the foregoing, except that the slabs are allowed to cool after being sheared to length at the slab-producing mill. The cooled slabs then are laid out for inspection to locate visible surface and edge defects which are marked for conditioning by the procedures described in Chapter 26. The conditioned slabs then are charged into reheating furnaces at the continuous hot-strip mill.

The first or single-heating method described above results in substantially lower fuel costs and minimizes handling and conditioning expenses, but does not provide sufficient flexibility in scheduling hot-strip mills

rolling a widely varying product mix or a substantial proportion of small orders. The reheating method (the second described above) has proven more advantageous as it provides full flexibility of hot-strip mill scheduling, permits closer metallurgical control of steel-rolling temperatures and minimizes injurious steel-surface defects resulting from defective slab areas.

In accordance with the latter practice, conversion of ingots to slabs is effected by the following typical steps: After stripping, the ingots are charged in the soaking pits of a blooming or slabbing mill and brought to a uniform temperature, approximately 2400° F for the low-carbon steel grades which comprise the bulk of this tonnage. They then are removed by crane-borne tongs to ingot buggies that convey them to the entry roll table of the slabbing or blooming mill. Some mills are equipped with turntables at this point which automatically record ingot weight and place the ingots butt first with respect to the rolling stands and shears; this practice is advantageous as it provides a close check on ingot weight and efficient control of end scrap. Ingots then are passed along a roller table to the reducing stand, consisting of a slabbing mill or a blooming mill of the single stand, reversing type. The slabbing mill is equipped with both horizontal and vertical rolls that work on all four sides of the ingot simultaneously, while the blooming mill operates with horizontal rolls only (see Chapter 25). Greater tonnages and wider slab sections can be produced on the slabbing mills as a result of this difference in design. After reduction to the prescribed slab thickness and width, the elongated slab is advanced toward a shear at the end of the mill roller table. Some mills, at a point between the reducing stand and the shears, are equipped with automatic flame scarfing equipment for the purpose of removing all but the worst edge or surface defects from the slab. The shears cut the slab product to the designated lengths, cropping sufficient scrap from the two ends of the slab, corresponding to what originally was the top and bottom of the ingot, respectively, to insure elimination of pipe, porosity, mechanical end laps, slag deposits, and so on.

Immediately after hot shearing, the slabs are hot stamped with identification markings, such as heat number, ingot number and cut number. After shearing and stamping, the slabs are piled on cooling beds and permitted to cool to a workable temperature, then laid out individually for inspection and marking of surface and edge defects such as scabs, ingot cracks, spongy surface, breaks, tears, and so on which, if not removed, would result in surface slivers, scabs, skin laminations or cracked edges on the finished strip. The defects are removed by scarfing with an oxy-acetylene torch or, on stainless steels, by grinding with abrasive wheels or powder scarfing. The slabs then are repiled, and each is painted with identifying information normally including heat, ingot and cut numbers, thickness, width, weight and code letters or numerals representing chemical composition and steelmaking process.

The finished slabs finally are transported to the storage yard of the hot-strip mill, where they are stacked in orderly fashion according to size and steel grade to facilitate their selection and charging into magazines feeding the reheating furnaces of the hot-strip mill.

SECTION 4

CONTINUOUS HOT-STRIP MILLS

Development and Output—The terms "strip" and "sheet," as applied to the finished products, have definite

reference to width and gage limitations, as shown in Section 1 of this chapter, and the term "hot-rolled

breakdowns for cold reduction" defines a semi-finished product for subsequent rolling by another process. These distinctions do not exist when the terms are used in connection with a continuous mill rolling such products; thus, a mill rolling continuous lengths of strip, sheet, or breakdowns commonly is known as a "strip mill." The history of development of continuous hot-strip mills was summarized in Section 1 of this chapter.

At the end of 1954, about forty wide continuous hot-strip mills were in operation in the United States, their combined annual capacity being approximately 40,000,000 net tons. As a group, the wide strip mills roll flat steel in thicknesses of 0.04 inch to 1.25 inches, widths of 24 to 96 inches, and lengths up to 2,000 feet. Each mill has its own limitations as to sizes of finished product though, as a general rule, no mill exceeds a product width-to-thickness ratio of 1000:1.

General Arrangement of Modern Mills—The modern wide hot-strip mill has become quite standardized in its general layout. Slabs are heated in two or more continuous reheating furnaces. A typical rolling train will consist of a roughing scalebreaker, four four-high roughing stands, a finishing scalebreaker, and six four-high finishing stands. Driven table rolls convey the steel from furnace to mill and also from stand to stand. If the mill is to produce strip, sheets or breakdowns of greater width than the maximum width of slab available, the first rougher or roughing stand is a broadside mill in which the width of the slab is increased in a single pass by cross rolling. In this case, turntables for manipulating the slab must precede and follow this stand. A slab squeezer also follows the broadside mill. The next three roughing stands usually are provided with integral vertical edgers in front of each stand. Separating the roughing train from the finishing train is a holding table, while the finishing end is a closely grouped tandem train composed of the finishing scalebreaker and six finishing stands.

High-pressure hydraulic sprays to remove scale from the hot slab are located after the two scalebreakers and perhaps at several roughing stands. Water is supplied to the spray nozzles by suitable high-pressure pumps.

Following the last finishing stand there is usually a flying shear for cutting the rolled product into lengths, if so desired. As the steel proceeds from the mill, it is carried over a long table called the runout table, consisting of individually driven rollers. Two or more coilers are located in this table; they operate to coil the material when continuous long lengths are required. If short lengths are cut at the flying shears, the coilers are inoperative and the steel passes over them and onto a piler at the end of the table. Additional tables may be installed parallel to the central runout table, with suitable transfers for moving material to them; this equipment is used principally when the heavier gages are being rolled.

The most commonly used hot-mill arrangement just described, employing continuous roughing and continuous finishing trains, provides high rolling capacity and rapid steel travel with little loss of heat, but entails a high installation cost and a fixed number of passes, with some loss of flexibility in making rapid changes in the mill set-up when the size of product to be rolled is changed. An alternate arrangement, used in several instances, employs a reversing roughing mill and a continuous finishing train; this arrangement has a lower original installation cost, requires less floor space, and is flexible with regard to the number of passes available, but at a sacrifice in capacity and operating cost. Another modification of the conventional mill (the single-heating practice already described) provides for

rolling the slab direct from the blooming operation, utilizing retained heat and by-passing the slab-reheating furnace; this practice saves fuel but sacrifices flexibility of scheduling rolling operations on the continuous hot mill.

Still another arrangement for continuous hot rolling of steel calls for the use of a reversing hot mill for finishing. This mill may have a conventional roughing train or single reversing stand for roughing, followed by the single reversing finishing mill. The reversing finishing mill has a pair of pinch rolls and a paddle-type coiler located on both sides (entry and exit) of the mill. The coilers operate inside of small heating furnaces which keep the steel hot and permit the finishing rolling operation to be carried out by repeated reversing of the finishing stand. This type of mill entails low initial cost and is highly satisfactory for the production of small orders or the rolling of alloy steels.

Control of Finished Product Quality—When an order is accepted for production by a particular mill, the first step taken is to determine the proper grade of steel and the size and surface quality of the slab necessary to make the order. The order is then grouped with others and scheduled in its proper rolling sequence. The principal factors taken into consideration at this important stage are rolling width, gage, and steel composition.

The next step in production requiring control is the operation of heating the slabs to rolling temperature. The slabs must be heated uniformly throughout, and also must have a uniform "scale jacket" that will "clean up" readily in rolling. Many rolling delays and mechanical difficulties are a direct result of poor heating practice; the steel may not be "soaked" sufficiently, may be too hot, too cold, or unduly hard to clean.

The third step is to rough down the slab to a predetermined intermediate thickness. As the slab leaves the last roughing stand, it should be flat, straight, free of furnace scale, true to width and of a cross-section suitable for further reduction on the finishing stands. The first rolling pass on the slab is done on a scalebreaker which is followed immediately with a high-pressure hydraulic spray to facilitate removal of the furnace scale. In addition, there are usually one or two more descaling sprays following the second or third roughing stands and numerous steam and air sprays available to remove any further scale that may be loosened during rolling or edging. Proper use of the broadside mill, slab squeezer and the three vertical edgers normally will guarantee the uniform width essential for all subsequent operations.

Next, the finishing train must be operated with careful regulation to obtain a finished hot-rolled product of prime quality. Surface, gage, width, finishing temperature and cross-sectional contour of the product, all are required to meet given standards depending upon the subsequent treatment or ultimate use of the material in question. As an example, metallurgical requirements may dictate a definite finishing temperature for a particular gage and width to be rolled. Time on the holding table prior to coiling, number of descaling sprays used during rolling in the finishing stands, speed of the finishing train, and method of drafting, all affect the finishing temperature and may be varied at the discretion of the operator to help meet requirements. Defects in the surface of the rolled steel, if not evident in the rough slab, usually can be traced to defects in the surfaces of the work rolls on the finishing stands and are corrected readily by substituting newly-surfaced work rolls. The principal factors affecting the overall dimensional accuracy of the finished hot rolled product include contour of the work rolls and back-up rolls as installed, changes

HOT STRIP ROLLING MILL

TYPICAL REDUCTIONS PER PASS IN FINISHING STANDS

(THIS DRAWING IS ENTIRELY SCHEMATIC AND NOT TO SCALE)

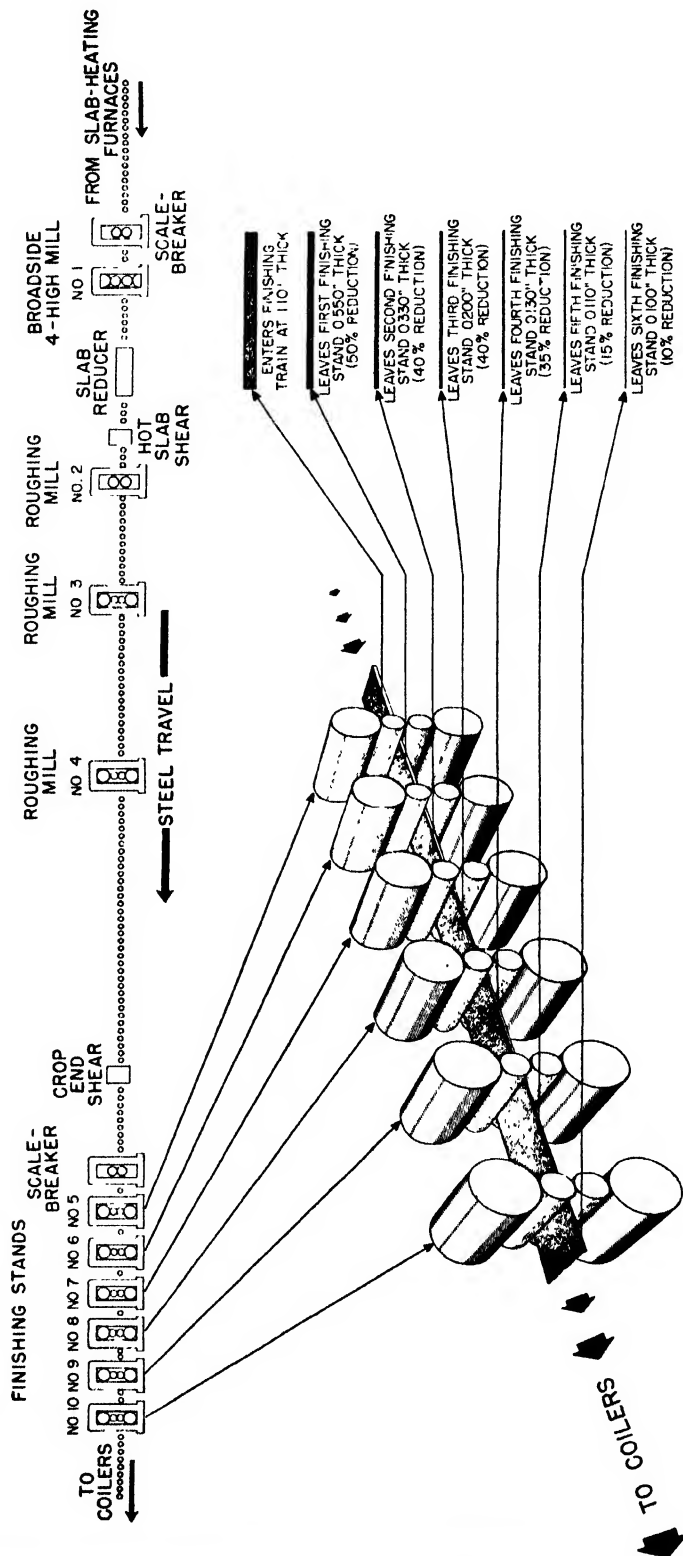


FIG. 33-1. Typical reductions per pass in the finishing stands of a hot-strip rolling mill.

in the contour of the work rolls and back-up rolls due to intermittent heating and cooling, method of drafting (i.e., amount of reduction in successive passes), and rolling sequence of various gages and widths.

Figure 33—1 shows typical reductions per pass in the finishing train of a continuous wide hot-strip mill.

The final step in rolling an order on a strip mill is disposition of the hot-rolled product. On some mills, the product may be cut into shorter lengths on a flying shear located at the exit end of the mill, the sheared pieces progressing along the runout table to a hot piler. The greater portion of hot-rolled flat material, however, is handled by the hot-coiling method; this includes the semi-finished product designated as hot-rolled break-downs in coils for subsequent cold reduction, as well as hot-rolled sheets in coils which may be shipped as such or transferred to the finishing department where they are uncoiled and processed into the form of flat cut sheets. The essential requirements of the coiler are to receive the material at mill speeds and coil it tightly without excessive tension, telescoping, scratching or marking and, finally, discharge the finished coil quickly without damage.

A Modern Wide Continuous Mill—The 80-inch hot-strip mill at the Gary Sheet and Tin Mill of United States Steel Corporation serves as a modern example of the wide continuous mill. This mill was built originally in 1935 and was modernized completely in 1948.

Slabs for the mill may range in size from 4 to 7½ inches in thickness, 18 to 64 inches in width and 80 to 216 inches in length. The conditioned slabs, as needed, are carried by overhead crane to the individual storage machine located in front of the charging end of each of the five slab-reheating furnaces, which extend at right angles into the hot-strip mill building. The slab pushers of each furnace travel under the corresponding storage magazine and push the bottom slab of the pile in the magazine onto the charging table. This table is equipped with driven rollers to position the slabs properly before the furnace doors. When the slabs are in position on the

charging table, they are shoved over skids into the furnace by pushers (Figure 33—2) operated by 150-horsepower, 500 r.p.m., DC motors through worm reduction drives. These are controlled from an operating platform behind the charging end of the furnaces. The doors of the furnaces also are controlled from there and are arranged so that, when the charging door is opened to push a slab into the furnace, the door on the delivery end simultaneously opens, permitting a slab to be discharged over an apron onto a roller table (Figure 33—3).

The five furnaces all are of the zone-controlled, triple-fired, recuperative, continuous type. Each furnace is 80 feet long by 20 feet wide and is capable of heating approximately 105 tons of steel slabs per hour to a maximum of 2400° F. The furnaces normally are fired with coke-oven gas or fuel oil, and are equipped with automatic combustion and temperature controls. Natural gas also may be used or a mixture of natural and coke-oven gases. Approximately 340,000 cu. ft. of coke-oven gas per hour are required to heat each furnace. Preheated air at 700° F is obtained for combustion by passing the hot products of combustion from the furnaces through recuperators. Air is pushed through the recuperators for each furnace by a motor-driven fan, rated at 50,000 cu. ft. per minute. Flue gas is conducted through underground flues to individual stacks, 150 feet high, or can be diverted to be used on two waste-heat boilers.

The slabs move through the first 60 feet of the furnaces on water-cooled skids which permit heating on both tops and bottoms. These skids use about 60,000 gallons of cooling water per hour for each furnace. The last 17 feet of the furnaces, known as the "soaking zone," have a flat bottom (combination steel and brick) where the slabs are brought to a uniform temperature throughout. Accurate control of temperatures, furnace atmosphere composition, and pressure in all parts of the furnaces is facilitated by a battery of recording and indicating instruments. Doors are provided along the sides of the furnaces for observation purposes and to permit manual positioning of slabs if desired.

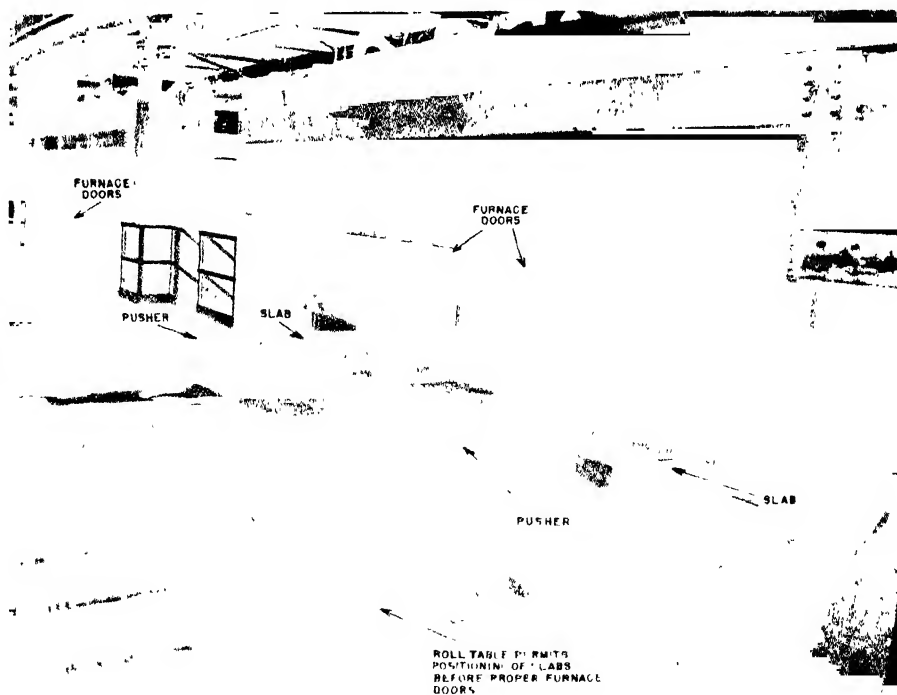


FIG. 33—2. Charging end of heating furnaces for an 80-inch mill, showing slabs in position before the furnace doors. In the foreground are the pushers which force the slabs through the nearest furnace.



FIG. 33—3. Discharge ends of a group of three heating furnaces for an 80-inch mill.

FIG. 33—4. Broadside stand of an 80-inch mill. Here slabs receive their first heavy reduction in thickness. The slab shown here has received a broadside (sideways) pass to extend it to the proper width. The turntable (slab turn-around) next will turn the slab 90 degrees before it goes on to the following roll stands.

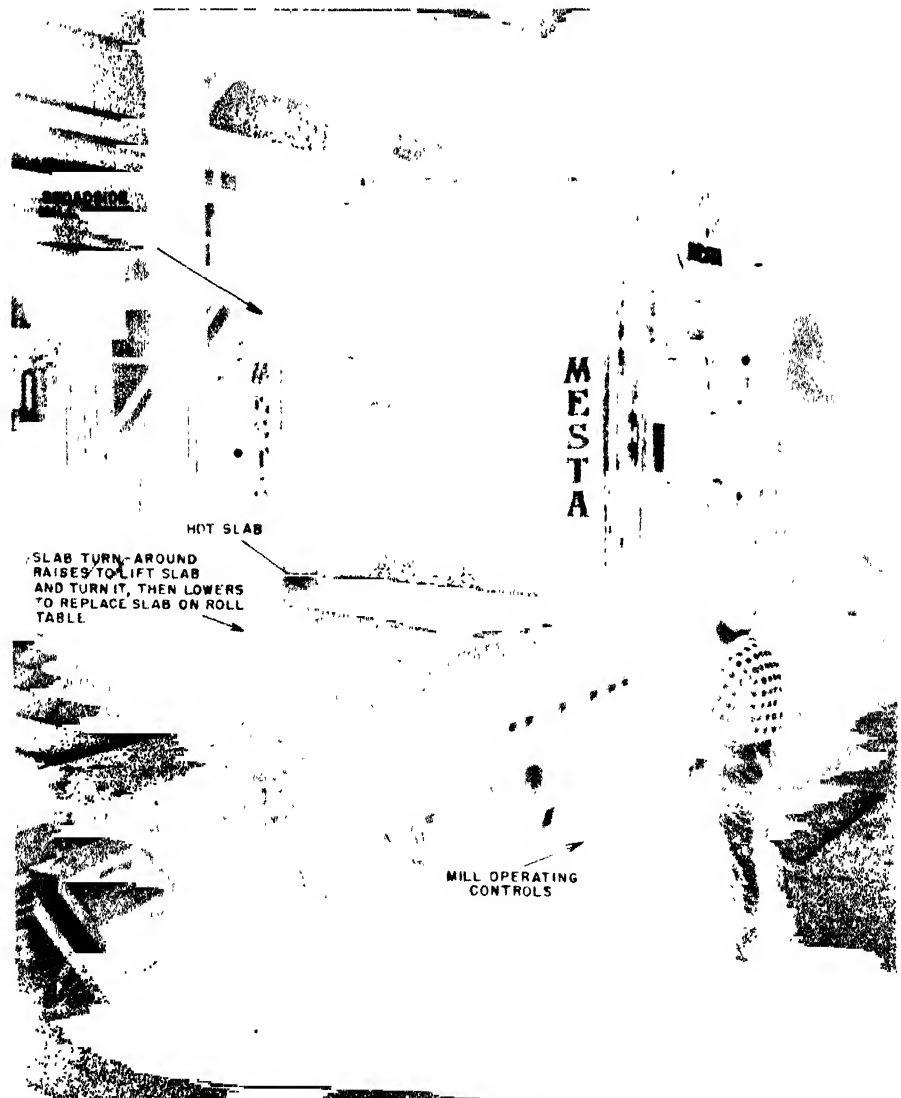
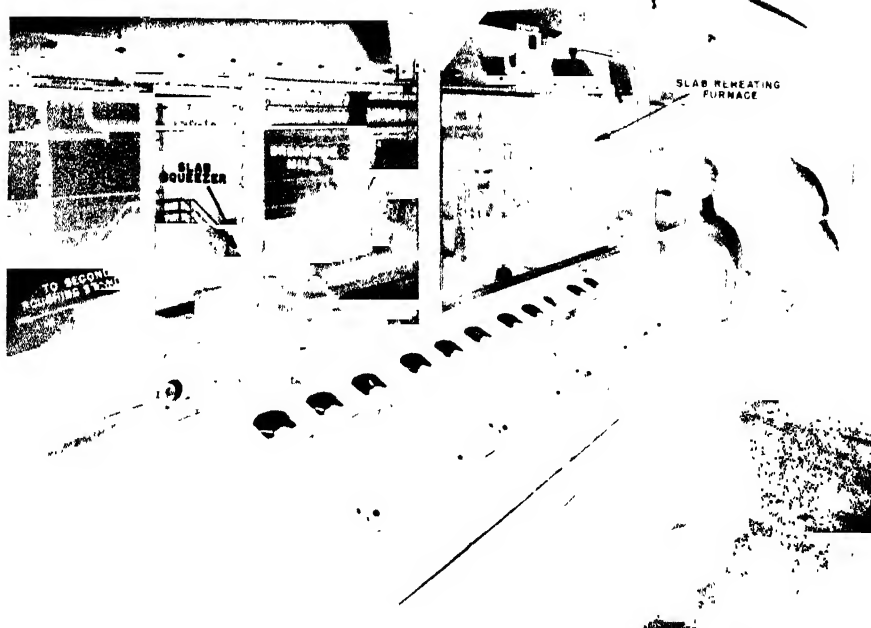


FIG. 33-5. View of part of an 80-inch continuous mill from the operator's pulpit from which the speed of the roughing stands is controlled.



Slabs are discharged from the heating furnaces at temperatures ranging from 2000 to 2400° F, depending upon the grade of steel and thickness of finished product. They are carried to the mill by a roller table 245 feet long. This table is made up of six units, each driven by a 50-horsepower motor. The table is reversible, so that, when necessary, slabs can be returned to slab storage by a transfer located parallel to the furnaces.

The first mill stand in the 80-inch hot strip mill is the No. 1 scale-breaker. This mill is a two-high, 36-inch by 80-inch stand, operated at 212 feet per minute, and driven by a 1250-horsepower, 490 r.p.m., AC motor through a double reduction-gear drive equipped with two 12½-foot flywheels. The descaling water spray following this stand operates at 1300 lb. per sq. in. pressure. Slabs travel 100 to 200 feet per minute over a 26-foot reversible table between this scale-breaker and the first roughing stand.

The first roughing stand is a four-high 40-inch and 54-inch by 130-inch mill operated at about 182 feet per minute, driven by a 3500-horsepower, 6600-volt, 340 r.p.m., AC motor through a gear set and two 13½-foot flywheels. This stand is a broadside mill in which slabs up to 127 inches long can be spread to a maximum of 77 inches wide in a single pass, when it is necessary to produce widths greater than the original slab (Figures 33-4 and 33-5). A rack-type slab pusher on the entry side of this stand assures square and proper entry of the slab between the rolls. Slab turn-arounds are provided before and after this stand to rotate the slab through 90 degrees when the stand is used for broadsiding. Following the broadside mill, a slab squeezer serves to true up slab edges and widths and to flatten them for subsequent rolling. This squeezer has a 24-foot head with a maximum stroke of 6½ inches on a 6-inch thick slab. Slabs reach the squeezer and pass to the second roughing stand over a reversible roller table 100 feet long, in three independent sections, each driven by a 50-horsepower, 550 r.p.m., reversible DC motor. Table speed is 120 to 240 feet per minute. A slab shear is located between the squeezer and the second roughing stand.

The second roughing stand is a 44-inch by 80-inch universal two-high mill with vertical edging rolls mounted on the entry side of the stand. This mill is driven by a 3500-horsepower, 490 r.p.m., AC motor. The gear drive is equipped with two 13½-foot flywheels. The edging rolls are driven through twin worm reduction sets by a 350-horsepower, 400 to 1000 r.p.m., DC motor. After leaving this stand, the slab passes over a reversible roller table 51 feet long, driven by a 50-horsepower, 550 r.p.m., DC motor at speeds of 178 to 356 feet per minute.

No. 3 rougher is a four-high 32-inch and 53-inch by 80-inch universal stand, as is No. 4 rougher, located 94 feet beyond No. 3. Both of these stands have vertical edging rolls mounted on the entry side, duplicates of those on the second roughing stand. Each mill is driven, through a gear set with two 13½-foot flywheels, by a 3500-horsepower, 490 r.p.m., AC motor. A holding table, 200 feet long, lies between the roughing and finishing trains. This table is divided into two sections, each driven by two 50-horsepower, 550 to 1100 r.p.m., DC motors, providing table speeds from 100 to 400 feet per minute. The rolled product is cooled by holding on this table, if necessary, to attain proper finishing temperatures to meet requirements. A rotary crop shear is installed at the finishing end of the table, so that both the front and back ends of the material can be squared off before finishing.

A second scalebreaker, two-high, 36-inch by 80-inch, followed by high-pressure descaling sprays and driven by an 800-horsepower, 150 to 450 r.p.m., DC motor, precedes the finishing train. The finishing train is six four-high, 26-inch and 53-inch by 80-inch stands spaced on 22-foot centers (Figure 33-6). Each stand of the finishing train is driven by a 600-volt, DC motor. Numbers 5, 6 and 7 stand motors are of 4500-horsepower capacity, operate at 125 to 282 r.p.m., and drive through gear sets. No. 8 (4500-horsepower, 85 to 190 r.p.m.), No. 9 (4500-horsepower, 100 to 230 r.p.m.) and No. 10 (3000-horsepower, 110 to 255 r.p.m.) motors drive direct.

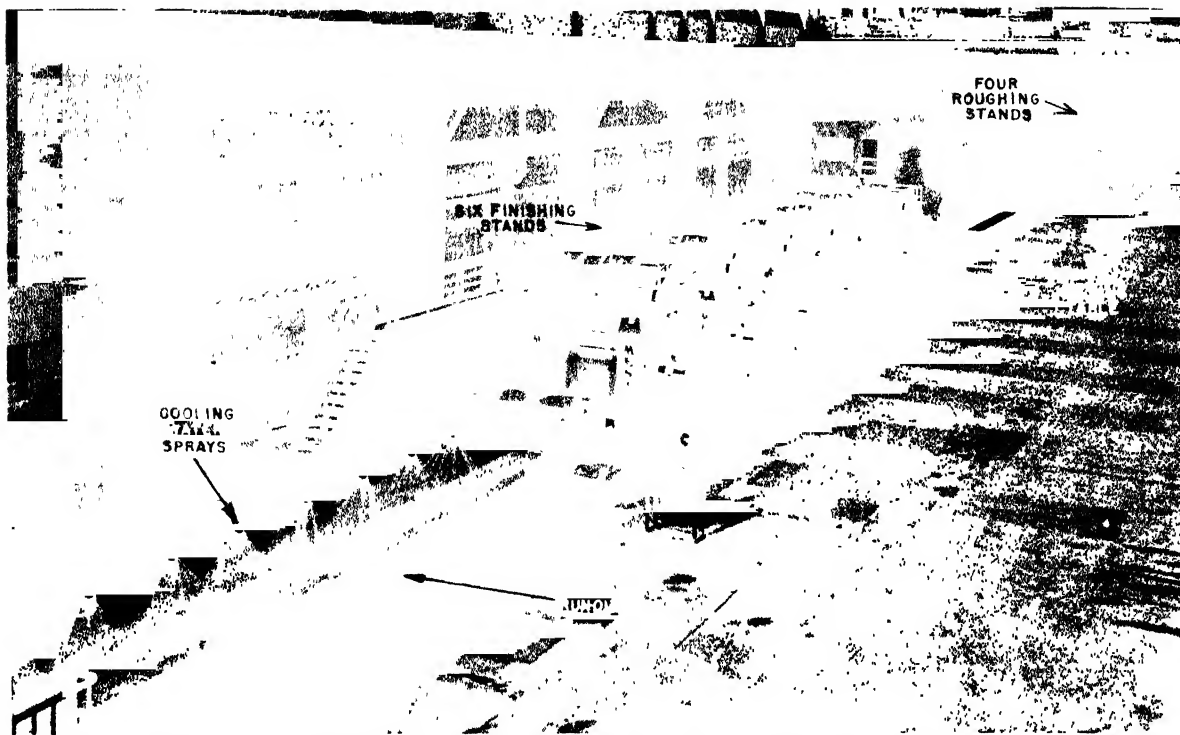


FIG. 33-6. General view of an 80-inch continuous hot strip mill. In the background are the roughing stands. In the center are the six finishing mills. The cooling sprays over the runout table are in the left foreground.

Product speed leaving No. 10 stand reaches a maximum of 1960 feet per minute. Loopers are provided between the finishing stands, each driven through a worm reduction unit by a 200 pound-foot torque DC motor. Steam sprays are provided on the delivery side of each stand and tachometers are installed in control stations to indicate mill speeds.

The screwdown on each finishing stand in the mill is operated by two 75-horsepower, 575 r.p.m., DC motors equipped with magnetic clutches and dynamic brakes. Single 35-horsepower, 575 r.p.m., DC motors drive the screwdowns on the two scalebreakers and the first two roughing stands through cutout clutches; two similar motors with magnetic clutches and dynamic brakes are used on No. 3 and on No. 4 roughing stands. The top roll on all stands, except No. 2 scalebreaker, has a hydraulic roll balance. No. 2 scalebreaker has a spring balance.

Oil-film bearings are used on all back-up rolls in the four-high finishing stands. Grease-lubricated roller

thrust bearings are used on Nos. 1 and 2 scalebreakers and 3 and 4 roughing mills, while fabric bearings are used on the back-up rolls of No. 1 roughing mill and the work rolls of No. 2 roughing mill. The mill is lubricated automatically by five separate recirculating oil systems servicing the oil-type back-up roll bearings, the pinion-stand gears and bearings of the large mill-drive motors and motor-generator sets. Grease requirements are maintained automatically by several central-station greasing systems strategically located throughout the mill.

Hydraulic requirements for the descaling systems are met by three units made up of two 1000-gallon per minute, 1300 lb. per sq. in. pressure pumps driven by a 1250-horsepower, 6600-volt, 1800 r.p.m., AC motor. Temperature of the product is watched carefully throughout its passage in the mill by use of one recording pyrometer and three recording-indicating pyrometers placed at the entry to No. 1 roughing stand, on the

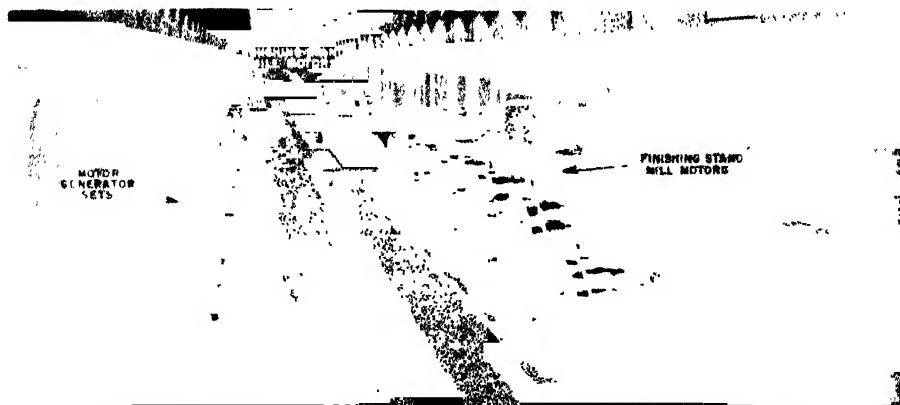


FIG. 33-7. Motor room of an 80-inch hot-strip mill. At the left are the motor-generator sets which supply direct current to the finishing-stand mill motors in the right foreground.

holding table, at the entry to No. 10 stand and at the entry to the coiler.

The hot-strip mill building is 95 feet wide and 2352 feet long; it is served by three 60-ton cranes with 10-ton auxiliary hoists, one 60-ton crane with 15-ton auxiliary and one 15-ton crane, all of 90-foot span. A building 92 by 624 feet, conveniently located alongside the hot mill, houses service and roll shops. The latter include roll lathes, roll grinders and special degreasing equipment for cleaning bearings. The hot-strip mill motor room, 65 by 644 feet, is served by one 50-ton crane having a 60-foot span. The twelve hot mill drives, with their reduction gears, control equipment, switchgear, etc., are located in this room (Figure 33-7). The motor room is equipped with a recirculating-type ventilating system forcing air through coolers located in the motor-room basement into the motor room proper. All make-up air is filtered before distribution. The ventilating system is designed to handle 525,000 cubic feet of air per minute at 95° F.

Power is supplied from the outdoor substation via a metal-clad bus system at 6600 volts to the switchboard and is distributed to equipment through conduit laid in the concrete floor. Variable-voltage direct current is supplied to the main motors driving the finishing train from three 3-unit, 6000-kw, 600-volt synchronous motor-generator sets, each consisting of two 3000-kw, 600-volt, DC generators driven at 360 r.p.m. by an 8600-horsepower, 6600-volt synchronous motor.

Product Disposition—After leaving the last finishing stand of a continuous hot-strip mill, the rolled product is delivered onto a long runout table where it is carried either in long lengths to coilers, or to a sheet piler in cut lengths. The 80-inch hot-strip mill being described produces only coiled product. It has a flying shear, spaced 11 feet 9 inches from the last finishing stand,

which is used to crop either or both ends of product that is to be coiled as a single piece. The shear is driven by twin 150-horsepower 450 to 900 r.p.m., DC motors.

After leaving the flying shear, the steel passes over a roller table 364 feet long, having 243 rollers, each driven by a 4-horsepower, 0 to 800 r.p.m., DC motor through rubber-ball couplings. The table is equipped with cooling sprays both above and below the rollers. Three mandrel-type coilers are installed at the end of this table, each driven by a 350-horsepower, 1200 r.p.m., DC motor. Coils with a maximum width of 77 inches, an inside diameter of 32 inches and an outside diameter up to 69 inches can be accommodated. Air-operated pushers remove the coils from the mandrels onto a coil tilter, which places coils on end onto a chain conveyor which carries them to the coil-storage building (Figure 33-8). Heavy-duty scales are installed in the hot-coil conveyor, giving an accurate check on production and yield figures.

Two 900-kw, DC generating sets and several smaller units furnish power for all of the product-disposition equipment. Two generator sets normally are allotted to flying-shear service, two to the three runout-table sections between the flying shear and coilers and one to the three coilers. One generator is a spare which can be switched into any of the other services.

Metallurgy of Hot Strip—Wide hot-rolled strip from a modern continuous mill may be used in the as-rolled condition, in which case it is referred to as "hot-rolled sheets," with or without the application of such auxiliary treatments as pickling, shearing and flattening. When produced as hot-rolled breakdowns for cold reduction, it is pickled in coil form, cold reduced as much as 90 per cent of its original thickness, heat treated and further processed to cold-rolled sheets, strip, black plate, or the various coated sheet-mill and tin-mill products. In any case the metallurgical requirements of the great bulk

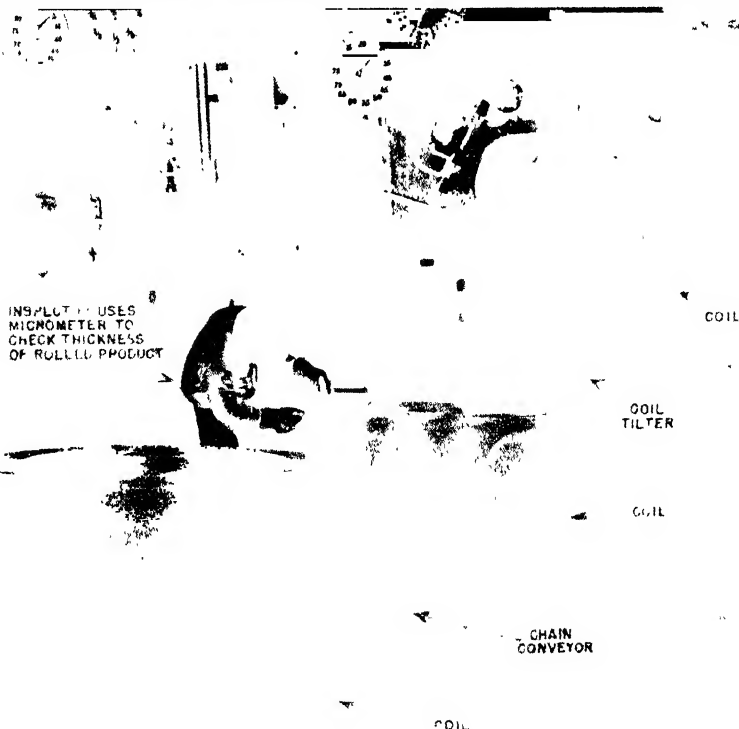


FIG. 33-8. Coiled product leaving the coiler of an 80-inch continuous hot-strip mill.

of the product are relatively simple and lend themselves to best operating conditions on the hot-strip mill and subsequent processing units.

The last hot-rolling operation (in the last finishing stand) should be conducted above the upper critical temperature on virtually all continuous hot-mill flat-rolled products. Such a practice permits the rolled steel to pass through a phase transformation after all hot work is finished and produces a uniformly fine, equiaxed ferritic grain throughout all portions of the steel. For the low-carbon steels generally used, proper finishing temperatures will have been attained when the apparent product temperature emerging from the last rolling stand is over 1550° F. This finishing temperature is practical over most thicknesses rolled on most modern mills at normal maximum rolling speeds.

If part of the hot rolling is conducted on steel which already has transformed partially to ferrite, the deformed ferrite grains usually will recrystallize and form patches or layers of abnormally coarse grains during the self-anneal induced by coiling or piling at the usual temperatures of 1200 to 1350° F. Such a structure is more likely to occur at the surface of the product, which is colder than the interior during rolling. Very thin hot-rolled material, inadvertently finished far below the upper critical and coiled or piled too cold to self-anneal, may retain microstructural evidence of hot-working. Neither condition is suitable for some types of severe drawing applications; both may be corrected by normalizing the sheet.

A special case occurs in the steels of the so-called "pure iron" or "enamel sheet" compositions, in which the sum of the carbon and manganese contents may be well under 0.10 per cent. Such compositions often exhibit a hot-short temperature range between 1650° to 1900° F, and normal hot rolling in that range may produce deep cracks on the edge of the product. Accordingly, it is the practice on many mills to complete the roughing operations above the hot-short range, to allow the steel to cool through the range by holding it on the conveyor table between the last roughing stand and the finishing train, and to resume rolling by passing the product into the finishing train below the hot-short range. By this practice it is impossible to finish above the upper critical temperature of these steels.

The runout table following the last rolling stand of most hot-strip mills is long enough and equipped with

enough quenching sprays to cool the single thickness of rolled product 200 to 500° F below the finishing temperature before the continuous length or the hot-sheared sheets become a part of the 2 to 20-ton mass of steel being formed in the coiler or on the piler. In addition, some mills have auxiliary tables or holding beds which allow single-thickness cooling to a take-off temperature of 500° F or lower. The cooling practice employed largely determines the metallurgical properties of the steel, its suitability for further processing and its final applicability to the intended use.

On hot-rolled products properly finished above the upper critical temperature, a uniform ferrite grain has been established and the runout cooling practice determines the carbide characteristics and, to some extent, the grain size. The self-annealing effect of a large mass of steel coiled or piled at around 1350° F produces considerable carbide agglomeration, a coarse ferrite grain and a soft, ductile sheet. Coiling or piling around 1200° F yields a fine, dispersed spheroidal carbide in a finer ferrite matrix, resulting in a somewhat harder sheet, which still retains excellent ductility. Even more drastic quenching produces various transformation states of carbide, down to and including martensite. For most low-carbon steel made either for use as hot-rolled sheets or as breakdowns for subsequent cold reduction, coiling temperatures of 1200 to 1300° F are employed; this range provides optimum uniformity of mechanical properties without excessive scale formation or over-annealing. A few mills have resorted to quenching the hot coil in a tank of water to inhibit scale formation and provide a surface oxide which can be pickled very readily. Steels of 0.15 to 0.30 per cent carbon content and alloy steels often are quenched drastically to attain higher strength levels from finer carbide dispersions, or are coiled very hot to facilitate cold reduction.

As most heat treatment after cold reduction is carried out below the lower critical point, the cold-reduced, box-annealed microstructure usually bears a relationship to the microstructure of the hot-rolled material. In the somewhat unusual case of the aluminum-killed, cold-rolled deep-drawing sheet, coiling temperatures under 1200° F are employed when rolling breakdowns on the continuous hot mill to provide the best drawing properties in the finished cold-reduced, box-annealed sheet.

SECTION 5

HAND HOT MILLS

Development—Prior to the advent of wide continuous hot-strip mills, all sheet and tin plate products were rolled on single-stand sheet and tin plate hot mills. These mills were patterned after the crude water-power or horse-driven units used to roll lead and copper as early as 1600, which were adapted to rolling thin steel sheets for tin plating in England about 1728. Until the middle 1930's, mills of this type, designated "jobbing" mills, were used to roll heavy-gage sheets, and "pack" or "finishing" mills were used for light-gage sheets and tin plate. Except for a few isolated instances, the hand hot mill is now obsolete for the rolling of flat-rolled products. For this reason, the process will be described in the past tense.

Process—In the original manual process of hot rolling, sheet bars, a semi-finished product rolled to a specified weight per linear foot and in various widths and thicknesses, were used to produce sheets. Packs made up of multiple thicknesses of steel first were prepared by

"breaking down" or cross rolling sheet bars cut to the correct length to produce the width of sheet desired. The bars were heated in a bar or "pair" furnace to near the upper critical point for the steel, and then passed through the rolling mill repeatedly in pairs, one bar closely following the other through the rolls of a two-high or three-high roughing mill. A number of final passes with the bars "matched," that is, placed one on top of the other, might be given the "breakdown" if desired to form a "pack" for finish rolling. For light gages, the pack usually was "doubled" or folded over prior to finish rolling. It should be noted that the term "breakdown," as now applied to a semi-finished hot-rolled product of the hot-strip mill, does not describe the same physical type of hot-rolled product as in older practice, but does connote still an intermediate stage in the production of flat-rolled products.

In jobbing mills where sheets heavier than 16 gage and up to 90 inches wide were produced, packs or bar break-

downs, after roughing, might be rolled immediately on a two-high finishing mill, down to desired finished thickness. In plants equipped with wide continuous hot-strip mills, breakdowns and packs eventually were made from hot-rolled breakdowns produced on the continuous hot mill, thus eliminating the operation of breaking down sheet bars (roughing) that is described later.

The number of sheets in the finished pack varied from two to eight according to the finished gage. While practices in different mills varied somewhat, a typical arrangement was for sheets of 14 to 18 gage to be finished in packs of twos; 19 to 22 gage, in threes; 23 to 27 gage, in fours; 27 to 29 gage, in sixes, 29 to 34 gage, in eights.

Light-gage sheets and tin plate were rolled from packs on two-high finishing mills. The packs were heated and then given successive passes through the mill until the desired thickness was obtained. Finished packs were allowed to cool, then side and end sheared to desired width and length and placed on the opening floor. During the rolling operation, the individual sheets of the pack became firmly stuck together. They were pulled apart manually with the use of tongs and "sticker-opener swords."

At this stage of production, the sheets were classified as hot-rolled only. For most uses they had to be heat treated, generally by box annealing. Additional treatments, such as pickling, cold rolling, and oiling, might be given to the sheets. The sheets also might be used as the base for galvanized, terne or tin coatings. For high-grade sheets, requiring a smooth surface free from all defects, the bars or the breakdowns might have been pickled prior to rolling. The pickling operation, in addition to removing scale, exposed defects which might cause imperfections and rejection of finished product.

Equipment and Operation—The processes and equipment required to convert ingots to finished rolled products by the foregoing methods are illustrated by the flow diagram, Figure 33—9. As shown, sheet bar was

delivered from the bar mill to the bar shears. When received, the bars were usually 30 feet or more in length, up to 1 inch in thickness, and 8 to 16 inches wide. As the length sheared from the bar finally became the width of the finished sheet, bars of length and width suited to produce desired sheet length and thickness were selected for each order. The bars were sheared to length on guillotine or alligator-type bar shears, pickled when desirable, inspected and delivered in piles to the heating furnaces adjacent to the roughing mills.

The bar furnaces were simple, rectangular, brickwork chambers located and arranged for efficient handling of bars to the roughing mills. Original furnaces were heated with coal, first hand-fired and later fired by automatic stokers. The furnaces still in use usually are equipped for gas or oil firing. The furnaces might be equipped with pushers which mechanically pushed a solid row of bars, standing on edge, into the entry door, through the furnace and out the exit door. For low-capacity specialty furnaces, the bars might be laid in the furnace and removed by hand tongs, one at a time through a single door.

When the bars had been heated, they were moved by hand or on roller or chain conveyors to the roughing mills, which might be two-high units with about 28-inch diameter rolls or three-high units with 28-inch diameter top and bottom rolls and middle rolls approximately 16 inches in diameter. The rolls, which were water-cooled during rolling, were chilled cast iron of lengths determined by the maximum width of sheet to be rolled. Ordinarily only the bottom roll was driven. The driving mechanism included a motor, gear reducer, spindles and couplings. Bearings were usually brass, lubricated by grease. The housings which formed the mill frame work were of cast steel, mounted on cast-iron bed plates. The amount of pressure exerted by the rolls on the bars was controlled by adjustment of two large screws which passed through bronze nuts located

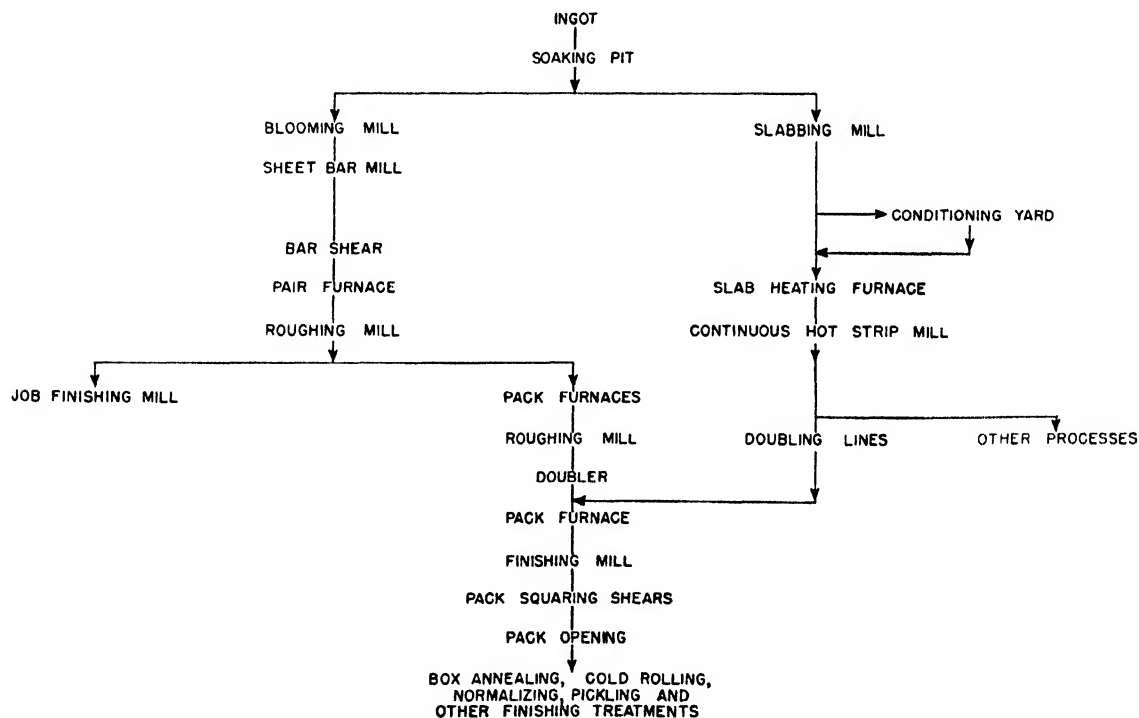


FIG. 33—9. Comparison of sequences of operations in the production of flat-rolled products.

in the tops of the housings. The screws normally were adjusted manually through hand wheels or notched spanner wheels and spanner bars, though electric motors were preferred for the larger mills. Rest bars, guides and support rolls were located between the housings at strategic points for feeding, catching and returning the bars. Mills of this type usually were installed in trains, with a number of units driven by the same drive through interconnecting spindles and couplings.

The rolls used in the finishing mills were chilled cast iron, varying from 22 inches to 32 inches in diameter, and up to 96 inches long. As the rolls were not cooled, and expanded while rolling the heated packs, they had to be shaped properly prior to use to compensate for expansion. In order to obtain the desired sheet surface, the roll surfaces had to be smooth and free from imperfections which would be transferred to the sheets.

Breakdowns from the bar mills were rolled on finishing mills either as packs of sixes or doubled sheets. The hot-mill breakdown doubler was a steam or compressed air hammer with a long face. The pack partially was folded over by hand, and the curl flattened by the hammer.

When breakdowns from hot-strip mills were used instead of bar breakdowns, they were doubled on special-purpose shearing and doubling lines which consisted of uncoilers for holding and paying-off the coiled hot-rolled breakdowns, a roller leveller which served as the flattener and feeding device, a shear and gage to cut the strip to desired breakdown length, a matching table on which two or more breakdown sheets were

piled one on top of another and a roller-type doubler.

From the doublers, the packs were taken by crane or conveyor to the entry end of the finishing-mill pack-reheating furnaces. On older-type units, or for specialty purposes, these furnaces might be rectangular heated chambers, very similar to the bar furnaces, into which the packs were placed by hand or the furnaces might be used as high-production continuous units, through which the packs were conveyed by special alloy conveyor chains. In either case, the packs were heated to desired temperature and then delivered manually or by conveyor to the finishing mills.

The finishing mills were two-high units very similar in appearance and arrangement to the two-high roughing mills. On the more modern high-production mill units (both roughing and finishing), mechanical feeder and catcher tables were used to handle the pack during the rolling operation. These tables were tilting units, on which were mounted rapidly reversing conveyor chains, side guides and stops, so arranged as to eliminate completely the need for manual handling of hot packs with tongs. On the finishing mills the sheets were rolled to final gage in lengths up to a maximum of about 12 feet.

The completed packs, consisting of from 2 to 8 sheets, were conveyed across cooling beds and then sheared to required size on guillotine-type squaring shears. All four edges were sheared and the knives were made long enough, when possible, to shear each edge of the largest rolled sheets in one stroke. The sheared packs were removed to the opening floor and the individual sheets pulled apart and piled for delivery to other processes.

SECTION 6

OXIDE REMOVAL

(Pickling and Shot Blasting)

Necessity for Removal—The presence of oxide (scale) on the surface of strip, sheet, or breakdowns, is objectionable when they are to be processed further. For example, the oxide must be removed and a clean surface provided if satisfactory results are to be obtained from the hot-rolled sheet or strip in any operation involving deformation of the material. If the sheets are for drawing applications, removal of the oxide is essential, as its presence on the steel surface tends to shorten die life, cause irregular drawing conditions and destroy surface smoothness of the finished product. Oxide removal is also necessary if the sheet or strip is to be used for further processing involving coating in order to permit proper alloying or adherence of metallic coatings and satisfactory adherence when a non-metallic coating or paint is used.

In the production of cold-reduced steel sheet and strip, it is necessary that the oxide resulting during hot rolling the steel slab to breakdown form be removed completely before cold reduction to prevent lack of uniformity and eliminate surface irregularities.

Types of Oxide—The term "oxide" as used here refers generally to the chemical compounds of iron and oxygen formed on the surface of the steel by exposure to air while the metal is at an elevated temperature. "Scale" is specifically the oxidized surface of steel produced during heating for working and during hot working of steel. Hence, the oxide produced on steel surfaces in hot-rolling processes is known as mill scale. Chemical compounds thus formed are iron oxides FeO , Fe_2O_3 , and Fe_3O_4 .

The mechanism whereby mill scale is formed generally is considered to be of a dynamic nature, whereby

alternate formation and reduction of the higher oxides of iron occur. Fe_2O_3 is formed first and then reduced

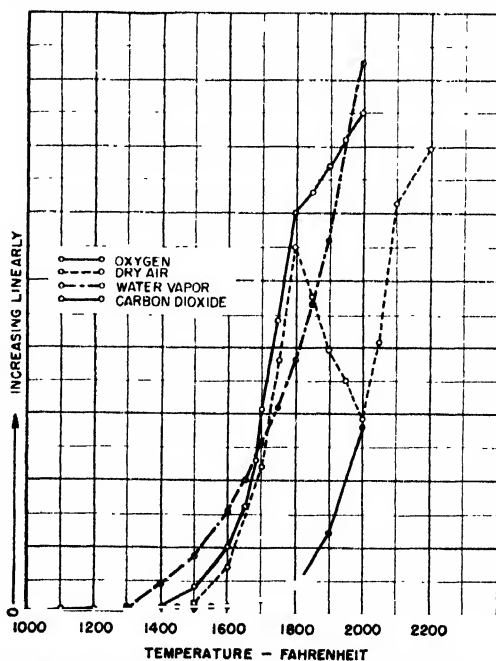


FIG. 33-10. Comparative effects of temperature and atmosphere on the scaling of plain carbon steels, all exposed for a constant time.

successively to Fe_2O_3 and FeO by the availability of iron. Additional Fe_2O_3 is formed at the atmosphere-surface interface and the process becomes continuous; the final result is a scale composed of layers richest in oxygen at the scale surface and richest in iron at the metal surface. FeO , the layer next to the steel, constitutes about 85 per cent of the scale thickness, Fe_2O_3 about 10 to 15 per cent and Fe_3O_4 about 0.5 to 2 per cent. There is evidence, too, of a molecular or ionic diffusion process involving oxygen moving inward and iron moving outward through the scale.

The rate of oxide formation is dependent on the temperature, composition and physical characteristics of the steel, and temperature, character and rate of flow of the atmosphere, as well as the length of time the steel is exposed to oxidizing conditions. Figure 33-10 shows graphically the oxide development at temperatures of 1000° to 2200° F for a low-carbon steel exposed for two hours to atmospheres of oxygen, dry air, carbon

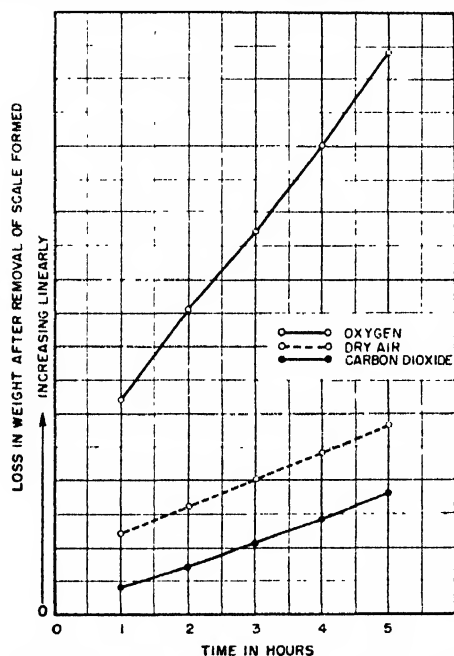


FIG. 33-11. Comparative effects of time of exposure and nature of the atmosphere on the scaling of plain carbon steels at 1520° F.

dioxide and water vapor. It is noted that rate of scaling increases uniformly with temperature when water and carbon dioxide atmospheres contact the steel surfaces and that interruptions occur in the rate of scaling when oxygen and dry air are employed. These interruptions are attributed to blistering of the scale which leads to pitting of the undersurface.

Figure 33-11, in which scale loss is plotted against time for samples of low-carbon steel exposed to carbon dioxide, air and oxygen atmospheres, shows increasing loss for an increasing time at a given temperature. Figures 33-10 and 33-11 also show relative oxidizing activity of the four atmospheres, with carbon dioxide the least oxidizing and dry air, oxygen and water vapor more strongly oxidizing in the order named.

PICKLING

Pickling is the process of chemically removing oxides and scale from the surface of a metal by the action of water solutions of inorganic acids. While pickling is only one of several methods of removing undesirable

surface oxides, this process is used the most widely in the manufacture of sheet and tin mill products, due to comparatively low operating costs and ease of operation. Considerable variation in type of pickling solution, operation and equipment is found in the industry. Among the types of pickling equipment may be mentioned the batch picklers, modified batch, semi-continuous and continuous picklers.

The reaction occurring when steel or iron materials are immersed in dilute inorganic acid solutions includes the solution of metal as a salt of the acid and the evolution of hydrogen. Steel pickled in dilute sulphuric-acid solutions is an example of this reaction, with the end products of reaction being ferrous sulphate and hydrogen. Adherent films of oxides are undermined by the acid attack through the pervious scale on the base metal. FeO is not dissolved as readily as the steel, but does have higher reaction rates than Fe_2O_3 and Fe_3O_4 , both of the latter being soluble very slowly in the acid. Ferric sulphate is formed first and then is reduced to ferrous sulphate by the free hydrogen. Sulphuric-acid baths rarely contain significant amounts of ferric sulphate since this compound is unstable in the presence of reducing agents. Certain metals, such as copper, chromium and nickel, retard the rate of pickling when they occur in the steel base, since the scale bearing these alloying metals inhibits acid attack. Silicon and aluminum form refractory-type oxides, which in turn lower the solubility rate of the oxide in the acid.

The rate of pickling is affected by numerous variables, including the aforementioned steel-base constituents and type and adherence of oxide to be removed. Solution temperature and concentration, ferrous sulphate concentration, agitation, time of immersion and presence of inhibitors influence the rate of acid attack. While the rate of pickling increases in direct proportion to the concentration of the acid from zero to 25 per cent by weight, the influence of temperature is much more pronounced. For example, in 15 per cent sulphuric acid an increase in temperature over the range 70° F to 210° F doubles the pickling rate for each rise of 15° or 20° F in temperature. Rate of solution of iron at 180° F is about five times the rate at 140° and about 100 times greater at boiling than at room temperature. The trend in recent years in batch pickling of hot-rolled steels is to maintain temperatures at 150° to 175° F if possible and to increase or decrease pickling activity with adjustment of the concentration, thus affording some savings in fuel for heating and avoiding decomposition of the inhibitor. Acid concentration is varied over a relatively wide range in the industry, dependent on the amount of pickling required to prepare the surface for the succeeding process. Much higher concentrations are used in continuous pickling methods, as the desired surface must be secured in the shortest possible time; hence, temperatures are maintained at 200° to 220° F and concentrations at 12 per cent to 25 per cent. These specific examples pertain to the commonly-used sulphuric-acid baths. The required concentration is also a function of the kind of acid used; for example, hydrochloric acid can be used in concentrations of 5 per cent to 50 per cent.

The retarding or inhibiting effect of ferrous sulphate is recognized widely and provisions are made in every pickling operation for adequate control of the salt build-up in the bath. Continued use of the bath without replenishment results ultimately in complete ineffectiveness of the solution. It is usually considered good practice to permit build-up of ferrous sulphate to 25 per cent and work the bath until the free acid content is reduced to less than 5 per cent.

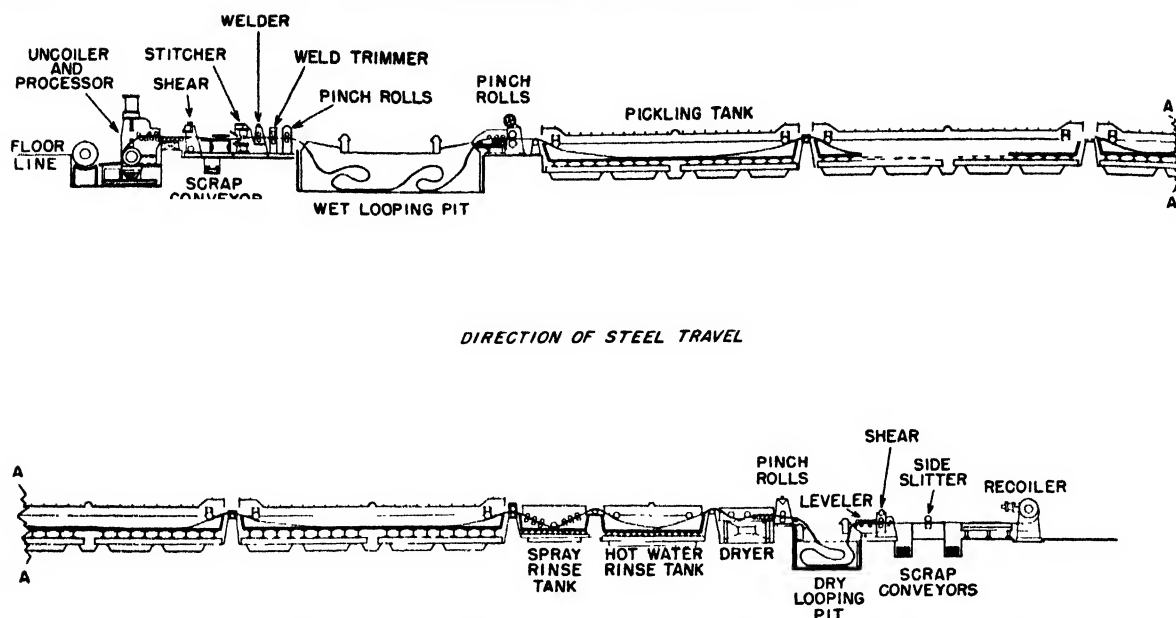


FIG. 33—12. Schematic arrangement of the equipment comprising a continuous pickling line.

Agitation of either the work or the bath saves time, metal and acid and is a practice used widely throughout the industry. Several methods have been adopted, such as raising and lowering the work in the bath, agitating the bath with plungers or circulatory systems or passing the work through the bath horizontally.

Inhibitors—The effect of inhibitors has been known and used commercially for years. An inhibitor is any substance added to a solution that inhibits or lessens acid attack on the steel itself, while permitting preferential attack on the iron oxides. Originally such substances as wheat bran were used but in recent years complex synthetic organic chemicals have been manufactured for this purpose. For economic reasons, some of the by-products or waste materials of the agricultural and oil industries have been used with varying success; these still retain a competitive position with the synthetic inhibitors. The principal requirements of an inhibitor are that it must be effective in very low concentrations, its effectiveness must be constant at all bath temperatures and concentrations, it must not leave an oily or otherwise harmful film on the steel, and it must be economical for use on the basis of cost per ton pickled. Many inhibitors promote foaming to restrict acid and heat losses from the bath so blanketed.

Wetting agents, while often constituting a supplementary ingredient to inhibitors, in most cases are somewhat of an inhibiting agent in themselves and are used principally to improve rinsing and wetting of the surfaces. Wetting agents are organic compounds that lower the interfacial tension between the steel and the liquid. Some improvement in pickling rate is believed to result from their use and improvement in rinsing is definite.

Continuous Pickling Lines—With the advent of continuous cold-reduction mills, it was necessary to design and develop suitable equipment to remove the oxides resulting from the continuous hot-rolling operation and prepare the hot-rolled breakdowns for cold reduction in coil form. This operation is performed in a continuous pickling line (Figure 33—12). The primary function of a continuous pickling line, as of other pickling processes, is the removal of oxide from the steel surface. This serves to promote maximum reduction with a minimum

of power, to assure good roll life in the cold-reduction mills and to secure the increased surface density possible with cold work.

The thickness of the oxide varies considerably on steel rolled on the hot-strip mill. Loose coiling permits greater atmospheric penetration into the wraps, with corresponding heavier oxide formation on the edge areas. Flexing of the steel in passing through the pickling line tends to break this scale or oxide film and permits more rapid attack by the acid bath.

The continuous pickler has other advantages or supplementary functions. The product of the hot-strip mill is subject to fluting (formation of creases when the steel is bent or otherwise deformed) due to lack of springiness. Continuous pickling lines usually are equipped with suitable apparatus for cold working the material so that severe local strains are eliminated and fluting largely is prevented. Another advantage is that they permit individual lengths to be joined into a single coil containing multiple lengths, often necessary because of coil-length limitations of the continuous hot-strip mill. The result is a much longer continuous coil for the cold-reduction mills. The pickling line permits inspection of the steel for defects and suitability for the next operation, and finally oiling of the steel as a protection against rusting and as an aid to cold reduction.

At the coil entry end of a typical continuous pickling line are facilities for handling and charging coiled product into the line. These usually consist of conveyors on which the coils are placed in proper sequence by overhead cranes, upenders in cases where the coil is delivered with the axis vertical, and a motor-driven integrated buggy or hoist for placing the coil in the uncoiling or pay-off equipment. The cold-working equipment, or "processor," integral with the uncoiling equipment, consists of a mandrel on which the coil is placed, a hold-down roll, and a series of smaller diameter rolls. After the coil is charged on the mandrel and the lead-end entered into the small diameter rolls, the hold-down roll is brought down and pressure applied to the material. This action alternately flexes the steel around the rolls, thus effectively "breaking" the surface scale into numerous fine cracks, and increasing the available

sub-oxide area for pickle attack. This flexing also cold works the steel enough to eliminate, in large part, the fluting tendencies of the hot-rolled steel. The group of small driven rolls immediately following the hold-down or breaker roll applies tension to the steel and also serves to straighten and flatten it. A stationary shear is located immediately after the processor for the cropping and squaring of the coil ends for butt-welding or stitching.

While most of the pickling lines in the industry include stitchers for fastening coil ends for continuously processing hot-rolled product, many also have installed flash butt-welding as supplementary equipment. The main advantage of this method of joining coil ends is that it provides a joint which can be cold reduced, whereas the lapped and stitched joint cannot.

Following welding, the flash, or excess metal resulting from the upsetting action of the welder, is trimmed off by a cutter designed for the purpose. The looping pit is next in line and provides a continuous storage space for material to compensate for short delays at the charging end and to permit a uniform rate of travel through the acid tanks. The looping pit is usually 10 to 12 feet deep and 20 to 40 feet long, depending on equipment speed. Construction is generally of concrete with adjustable wood side guides for retaining the lapped-over material and preventing twisting. Water is kept in the pit to minimize scratching and increase wetting action in the first pickling tank.

The pickling zone consists of several individual acid-proof tanks located in a series, comprising an effective immersion length of about 250 to 300 feet. While most lines have from three to five tanks, each about 70 to 80 feet long, some modern lines have only one long tank, divided by weirs into four or five sections, thereby increasing effective immersion depth about 10 per cent to 15 per cent. The inside dimensions of these tanks has been more or less standardized at four feet in depth and about one foot wider than the maximum product width. A steel shell is used for support with layers of rubber bonded to the steel and the rubber is protected from abrasion by a lining of about 9 inches of silica-base acid-proof brick. For operating temperatures in excess of 200° F, a bakelite-base cement generally is used for bonding. In modern high-speed lines operating at 200° to 220° F, the brick facing gradually is eroded away, so that replacement is required after several years of operation. Occasionally, small leaks in the rubber lining and the steel tank require patching. However, if care is taken to prevent acid attack on the outside of the tanks, the tank assemblies may be regarded as permanent for the life of the line.

Following the acid tank are rinsing tanks consisting of a cold-water spray rinse and a hot-water tank. The cold water rinses the acid carry-over from the steel. The hot-water rinse is a tank with an effective product immersion length of 15 to 20 feet. This tank completes the rinsing and by warming the steel, promotes flash drying prior to entering the succeeding set of pinch rolls. Situated between the final rinse tank and the pinch rolls are one, two or three banks of hot-air driers operating at low pressures. Pinch rolls at the exit end of the pickling tanks control the speed of product travel and, in conjunction with the pinch rolls which provide back tension at the entry end of the line, help to maintain the proper loops in the tanks.

The delivery end of the continuous pickling line has, in the order listed, a looping pit, pinch rolls, shear, oiler, recoiler and suitable supplementary equipment for conveying the finished product from the line. The pinch rolls preceding the shear are located so that product delivery

to the shear is facilitated. Stitches are removed at this point, as well as short sections which inspection has shown to be inferior quality. Some lines are provided also with rotary side trimmers at the entry end or, more commonly, at the delivery end.

Control of acid concentrations in four separate tanks within one pickling line posed a critical problem for some time. Until recently, it was considered good practice to measure the free acid and ferrous sulphate concentrations about four times per eight-hour period and make raw-acid additions to each tank on the basis of these frequent tests, until iron salts had increased to a very high concentration, at which time the tank was dumped and a new pickling solution made up. It was developed, however, that high concentration of iron salts in the entry tanks was not particularly harmful to pickling performance but that a similar condition in the final tank resulted in extremely poor line performance. The practice of adding raw acid, water and inhibitor (the latter being optional) to only the last pickling tank was then developed, and this practice, or slight variations of it, is now in general use in most continuous pickling lines. Tanks are connected with large rubber-lined or acid-proof tubes so that the acid solution will flow countercurrent to the product travel from the last pickling tank to the first pickling tank. This practice is called "cascading." The acid and water are added continuously at a prescribed rate, usually about 35 to 50 pounds of acid per ton of steel pickled. Hence, if the line production rate is 50 tons per hour, acid flow into the make-up pickling tank would be equivalent to about one-half to two-thirds gallons per minute. Steam is used generally for heating the solutions and temperatures are controlled automatically.

Maintenance of coil identity through the pickling operation is essential. Schedules are made out showing the coil number and other unit identification opposite a pickling sequence number. The coils are then charged into the line in the sequence listed on the schedule and unit identity is reassigned to the coil in the same sequence following pickling. Product delivered to the cold-reduction mills carries complete identity as to coil number, heat number, width and gage. Means of transporting the pickled coil to the storage area ahead of the cold-reduction mill varies from plant to plant, depending on the plant layout, nature of transportation available and the most economical method. Occasionally, long gravity-type conveyors are installed to deliver coils to a central point, from which a tractor or overhead crane stores the coils in areaways adjoining the cold-reduction mills. Slat-type conveyor-tractors also are employed and where the distances involved become greater, rail transportation is resorted to.

Inspection of the raw pickled product is carried on continuously at the exit end of the pickling lines. Each coil is inspected for surface and edge quality, width and gage. Some of the defects commonly causing rejection or diversion are as follows: slivers, cracked edges, laminations, off gage, off width, roll marks, underpickling, overpickling, handling damage and pitting.

Underpickling results when the steel has not had sufficient time in the pickling tanks to become free of adherent scale and occurs when acid concentration, solution temperatures and line speed are not balanced properly. Variations in the oxide and composition of the steel are also factors in underpickled product, as well as such factors as coiling temperature off the hot-strip mill and inadequate amount of cold working through the processor. Overpickling results from line delays which permit sections of the steel to remain in the acid too long. The presence of an inhibitor reduces iron loss, but



FIG. 33—13. General view of a mill department containing four continuous pickling lines.

when an inhibitor is not used, iron loss during a short delay period appreciably reduces thickness of the steel and raises the hazard of hydrogen embrittlement. Pitting is related to overpickling, the presence of non-metallic inclusions near the steel surface and to rolled-in scale, slag or a refractory substance. While overpickling is not common in continuous pickling operations, its occurrence does have a very serious effect on cold-reduction performance and surface appearance of the finished product. Product damage from handling or improper equipment adjustment can render the steel unsuitable for further processing.

Prior to recoiling, the pickled steel passes between a set of oiling rolls which cover both surfaces with a small amount of oil. The type of oil used to lubricate the steel, and protect it from rusting during storage and from scratching during handling, is determined by the type of lubricating system on the cold-reduction mill unit. Hence, palm oil, sometimes diluted with light mineral oil, is applied to the steel at the pickling lines when a straight palm oil or a solution containing palm oil is used on the cold-reduction mill. Finally, the pickled and oiled product is recoiled on a conventional up-type or down-type coiler.

Figure 33—13 shows a typical continuous pickling department consisting of four lines.

Batch Pickling—Pickling of steel sheets and other light-gage sheared lengths is performed with specialized equipment in which a batch of sheets are processed together. In general, there are two basic methods of batch pickling and several modifications of them, but the employment of agitation as a means of increasing pickling rate is common to both. One method is to employ a large wooden or acid-resistant concrete tank installed in the floor so that the top of the tank rises about two feet above floor level. The tank is divided into two sections by a wooden partition extending between 6 to 10 inches from the tank bottom; one section is smaller than the other. A large float is inserted in the smaller compartment and the material to be pickled is placed in the larger side. Forcing the float down into the bath

displaces acid solution in the smaller section and raises the level of the acid solution in the other compartment. The wood float is attached to a gear-driven crank, making the motion quite rapid, and effectively agitating the bath. Sheets are placed in a near vertical position on an acid-resistant metal rack, and pins inserted between every few sheets to facilitate exposure of all surfaces. The rack is then placed in the bath by an overhead crane and allowed to remain for several minutes. After sufficient pickling, the sheets are rinsed in a cold-water tank to wash off any iron-salt residue. This type of pickling is employed commonly in shops where a clean surface free from scale and mill dirt is required for hot-dipped galvanized or long terme products. Acid concentration is varied from 4 to as high as 12 per cent, depending on the amount of surface etch required. Temperatures usually are maintained between 150° F and 190° F.

In the second general type of batch picklers, the work is agitated instead of the bath. Equipment consists of a minimum of two, and more often three, tanks recessed into the floor and located at 90 degree steps to each other on about a 15-foot radius. At the center is a vertical mast topped by a four-armed spider from which the holding racks are hung. The pickling sequence begins at the loading station, followed by a high-concentration acid bath and, in a two-tank system, succeeded by rinsing and unloading stations. In the three-tank system, the first pickle tank is followed by either a more dilute acid solution or a cold-water rinse, followed in turn by a hot-water rinse. Racks must be unloaded at the fourth station prior to loading. A variation of this method has the tanks installed in tandem, with the first tank containing the highly concentrated acid solution, followed by a very dilute acid or cold-water rinse and finally with a hot-water rinse. The racks are suspended from a sectionalized conveyor system which plunges the work in the bath, the section immediately above the tanks being separated from the return rack conveyor off to the side of the pickle tanks. In this manner, racks are never removed from the overhead conveyor system, except

FIG. 33—14. A rack loaded with black plate is being pushed into position before a plunger-type pickling machine for "white pickling" of the plate prior to tinning.



for maintenance, and the product flow assumes a more continuous aspect. When pickled sheets are furnished the customer for use in that condition, it is necessary to force-dry them to minimize rusting. Equipment for drying consists of a short hydrochloric-acid bath, water spray rinse, high-volume V-type dryers, and pilers. Roller leveling frequently is included to obtain standard mill flatness and a rotary brush scrubber may be incorporated in the drying sequence.

Figure 33—14 shows a plunger-type pickling machine used for "white pickling" of plate for tinning.

Adjustment of the pickling bath for batch-type picklers is accomplished by adding raw acid after the water has been brought to the correct level. Sulphuric acid is used for low- and medium-carbon steels in concentrations ranging from 1½ per cent to 4 per cent for tin plate and from 4 per cent to 8 per cent for sheet products, with the temperature maintained between 150° F and 190° F. During working of the solution the acid is depleted and fresh acid must be added periodically as indicated by suitable tests for free acid and ferrous sulphate. Inhibitors generally are used so that iron loss is low and the hazard of hydrogen embrittlement reduced to a minimum.

After pickling certain steels containing small amounts of copper or nickel, a "smut" or dark film appears on the sheet surface. As a rule, the film contains relatively high percentages of the alloying elements and a dilute solution of hydrochloric acid must be used to attain the necessary cleanliness.

SHOT BLASTING

Shot blasting is employed in the flat-rolled steel industry for cleaning and smoothing the edges of plate, and for the removal from sheet surfaces of mill dirt, scale and rust in the manufacture of sheet for special applications such as galvanized culvert stock, where service requirements demand good adherence of relatively heavy coatings. In this operation, hot-rolled prod-

uct is fed continuously from coils or as cut-length sheets through the equipment, which consists usually of two separate shot blasters, one for each surface, interconnected with common abrasive-feeding auxiliary equipment. The type of abrasive used for this application is crushed-steel shot, which is more effective than round shot in giving the desired cutting action with a minimum of peening. Auxiliary equipment includes abrasive storage and feeding facilities and dust separators to eliminate the fines that result from the breakdown of the particles upon striking the sheet surface.

Shot blasting, or more correctly, grit abrasive blasting, has several advantages over pickling for preparing heavy sheet for galvanizing. The principal advantage is that of obtaining better adherence with heavy zinc coatings. Secondly, hydrogen embrittlement in heavy-gage sheets presents a greater hazard than in lighter gages, hence this type of cleaning is preferred over pickling. Storage of blasted product in dry areas permits processing of product ahead of the galvanizing pot, while pickling must necessarily be tied in with pot operations as pickled product rusts rapidly after pickling. On the other hand, shot-blasting equipment requires considerable maintenance, as the wear of abrasives on all contacting parts is relatively high under a wide range of operating conditions, and production rates are relatively low on such units.

Chapter 34

MANUFACTURE OF COLD-REDUCED FLAT-ROLLED PRODUCTS

SECTION 1

PRINCIPLES OF COLD REDUCTION

Introduction—Cold-reduced flat-rolled products are made by cold rolling hot-rolled breakdowns in coil form from the continuous hot-strip mill after the breakdowns have been descaled by continuous pickling. The purposes of cold rolling are to reduce thickness (25 to 99 per cent), to produce a smooth, dense surface and, with or without subsequent heat treatment, to develop controlled mechanical properties. The dimensional classifications of cold-reduced products are given in Table 33-II of Chapter 33.

The introduction of continuous methods for cold reduction of steel was the final step in the development of present-day rolling facilities for sheets and tin plate. As in the case of the hot-strip mill, cold reduction of wide hot-rolled breakdowns in coil form was developed from the rolling of narrow products. The original purpose of cold rolling was, however, to achieve the desired surface and mechanical properties, and reduction in thickness was of incidental importance. Cold rolling of strip probably originated in Germany early in the nineteenth century as a process for cold rolling high-carbon wire to produce a flattened cross-section. Similar practices were adopted thereafter in this country. Low-carbon cold-rolled strip, the forerunner of today's cold-reduced strip sheets and tin plate, was first made in this country at the Stanley Works at New Britain, Connecticut. For years, the raw material for such strip was produced on merchant bar mills. Later, cold-reduction equipment developed rapidly, concurrent with the introduction of narrow continuous hot-strip mills and their evolution to the modern wide mills. Today, the narrow and wide cold-reduction mills provide the major outlet for hot-rolled breakdowns in coil form.

In the cold reduction of wide hot-rolled breakdowns in coil form, the prime objective is reduction in thickness of the material, since the modern hot-strip mill cannot reduce the steel to the gage required for many applications (such as tinplate). The development of the cold-reduction process for wide products has been even more rapid than that of the continuous hot-strip mill. The first efforts toward cold reduction of wide hot-rolled breakdowns in coil form were made in the 1920's and the process changed rapidly from single-stand non-reversing mills through single-stand reversing mills and then to the present-day high-speed tandem mills. Using tin plate as an example of the rate of change in the methods for reducing very thin flat-rolled steel, it is found that, in 1936, 24 per cent of the black plate for this country's tin plate production was cold reduced, with the balance made by the old hot pack-rolling method. By 1939, 75 per cent of all tin

plate was made by the cold-reduction process and this figure reached 100 per cent by 1943.

Sequence of Operations in Cold Reduction—After hot rolling, the hot-rolled breakdowns in coil form are uncoiled, passed through a continuous pickler, dried, oiled, and re-coiled. The oil serves as a protection against rusting and as a lubricant during cold reduction. There are several types of cold-reduction mills which vary in design from single reversing two-high, four-high or multiple-roll units to continuous four-high stands with up to five units in tandem. In rolling on any of the single-stand reversing mills, the product is rolled back and forth between the work rolls, through which very high pressures are exerted, until the desired thickness is reached. The steel may be forced through the roll pass by rotation of the power-driven rolls or may be pulled through non-driven rolls by tension reels in front and back of the mill or a combination of these effects may be used. In tandem rolling, the product is given one pass through three, four or five stands (Figures 34—1, 34—2, 34—3 and 34—4), each contributing to thickness reduction and each driving the material being rolled, usually at speeds so synchronized that the steel is under tension at each of the stages between the pay-off reel, the various sets of work rolls and the re-coil reel. The total thickness reduction may vary from 25 per cent to as much as 90 per cent of the hot-rolled thickness for sheets and black plate and up to 99 per cent for strip. Unlike hot-rolling, no scale is formed, but much greater pressures and driving forces are required to effect a given reduction in thickness.

For any given pass in the cold-reduction process, the resultant of the compressive forces of the rolls on the steel and the tensional forces along its length between the reels and rolls must exceed the elastic limit of the steel to produce permanent deformation. For the first pass after hot rolling, these forces are at their minimum, and for each succeeding pass they increase progressively, because the effect of the cold working through each pass is to increase the elastic limit sharply. The ultimate strength also is increased, but more slowly. As a result, much of the cold reduction is done on very hard steel having very little residual ductility.

Heavy reductions at high speed on any of the various types of mills generate considerable heat and not only raise the temperature of the product but also that of the rolls. The heat generated must be dissipated by a system of flood lubrication in which a water-soluble oil or a mixture of oils is directed in small streams or jets against the roll bodies and the surface of the steel. Some mills use palm oil on the steel and high-pressure water on the rolls. In any event, the resultant steel

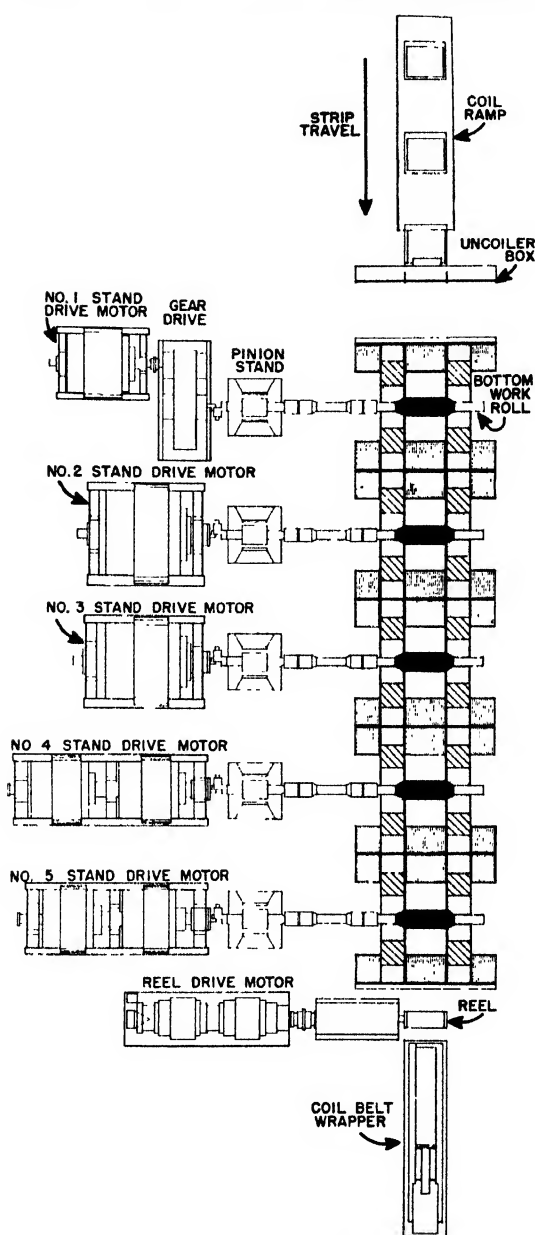


FIG. 34-1. Schematic arrangement of the mill stands and auxiliary equipment comprising a modern five-stand tandem cold-reduction mill. The shaded sections of the drawing represent the mill housings of the five four-high mill stands. Only one work roll of each stand is indicated in black; no other rolls are shown.

temperature generally runs between 150° and 250° F. On high-speed tandem mills, rolling 0.010-inch thick tin plate stock, delivery speeds up to 5000 feet per minute are now being used. The temperature of the steel leaving such high-speed mills may be as high as 400° F.

The proper reduction of thickness which should be made at each pass of a reversing mill or in rolling on each stand of a tandem mill can be expressed only in very general terms. The work load should be distrib-

uted as uniformly as possible at the various stages without falling very much below the maximum capacity of any stage. The maximum is determined in each case by several factors, of which the most important are mill design, power available, steel width and total reduction to be taken, steel lubrication, steel cross-sectional contour, steel hardness, steel tension, steel surface, roll diameter and roll surface. Generally, the lowest percentage of reduction is in the last reduction



FIG. 34-2. An 80-inch, three-stand tandem cold-reduction mill. The motors and driving mechanism are in the left foreground, with the three four-high mill stands in the middle background.

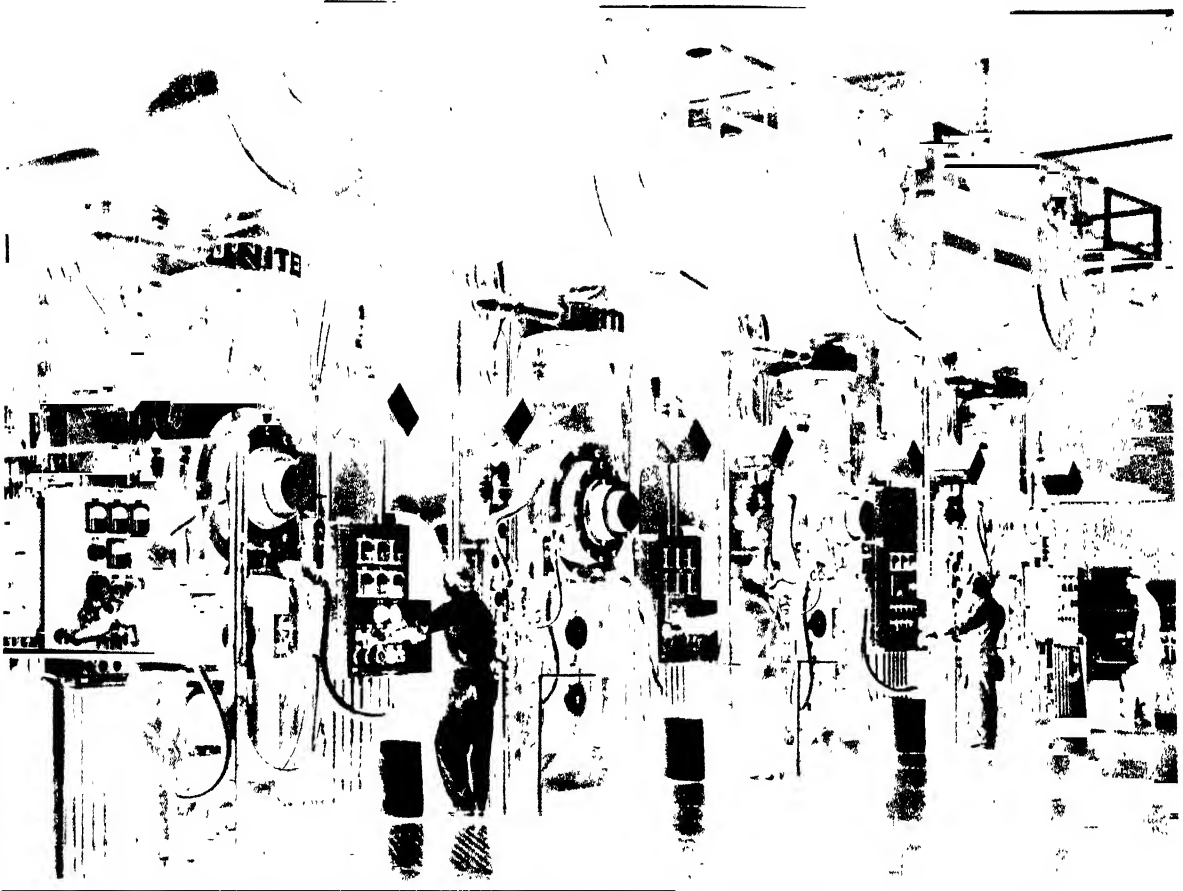


FIG. 34-3. A 54-inch, high-speed, four-stand tandem cold-reduction mill. The delivery end of this mill is at the left of the illustration.



FIG. 34-4. A 48-inch, high-speed, five-stand tandem cold-reduction mill. Product being rolled travels through the mill from right to left in this view.

pass to permit better control of the flatness of the finished product. On the conventional wide four-high tandem mills used to cold reduce most sheet and tin plate stock, the individual pass reduction will range generally between 25 and 45 per cent on all stands but

the last, where it will fall between 10 and 30 per cent. The resultant total reduction on these mills from hot-rolled breakdowns in coil form to finished product will usually run 45 to 65 per cent for most sheet gages and 80 to 90 per cent for most tin plate.

SECTION 2

ROLL ARRANGEMENT FOR COLD REDUCTION

In reducing the thickness of steel, the action of a pair of rolls may be visualized as being somewhat similar to that of two very blunt knife edges in cutting partially through a stationary piece across which the knife edges are opposed under pressure. The smaller the roll diameter or the sharper the knife edges, the less pressure is required to do the necessary work. Other factors being equal, the force imposed across a set of work rolls to obtain a given reduction in thickness of steel varies with the square root of the roll diameter. To minimize the power requirements and mill size, it is, therefore, advantageous to use work rolls of as small diameter as possible.

The steel being rolled is necessarily narrower than the length of the work rolls. In a two-high mill, these rolls are supported only at their ends and rolling pressure is applied there. In view of this, it is obvious that rolls of some minimum diameter no longer will be sufficiently rigid to withstand the necessary pressures without bending. In extreme cases, this bending along the length of the roll might be sufficiently great to cause the rolls to touch each other at the ends. The tendency, of course, is for the bending to increase as the mill width (the roll length) becomes greater. Using a "crowned roll," one that is slightly larger in diameter in the middle than at the ends, will compensate partially for the roll deflection just described.

In order to overcome the difficulties arising from bending of work rolls of small diameter without sacrificing the advantage of the small diameter, the modern mill backs up the work rolls with rolls of larger diameter. These more massive rolls resist the bending force of the screw-down pressures applied to their ends, and the pressures are transmitted to the work rolls along their entire length. For example, four-high cold-reduction mills in the 42-inch to 98-inch width range, which produce nearly all cold-rolled sheets and tin plate, will have two work rolls of 16-inch to 21-inch diameter, each backed up by a roll of 42-inch to 56-inch diameter. The driving power is applied at one end of each of the work rolls of such a mill and screwdown pressure at both ends of the backing-up rolls.

While the widespread use of such conventional four-high mills, in single stands or, more commonly, in tandem arrangements, has long ago proved their economic suitability for producing most of the cold-rolled flat steel products, many departures from this type have

been tried to obtain even smaller work-roll diameters. In the four-high Steckel mill, none of the rolls are driven and the steel is pulled back and forth between the 2 to 5-inch diameter work rolls by reversing power-driven reels. The back-up rolls in this case are 24 to 36 inches in diameter. Most Steckel installations are confined to the rolling of narrow strip. Other mills, known as the "six-high" or "cluster" types, use two or more backup rolls for each work roll. The currently most successful of the multiple-roll class is the Sendzimir mill, with two work rolls of 1 to 2½-inch diameter. Each work roll is backed up by two rolls of twice that diameter, each of which in turn is backed up by two segmented rolls of even larger diameter. The intermediate backup rolls are driven and the outer segmented rolls, which are actually rows of bearings on common shafts, provide a caster-like support action and permit the application of screw pressure by rotating these shafts eccentrically. As a result, a light, wholly-enclosed housing is possible and the entire roll system is immersed in lubricant. This mill appears to be well-adapted to taking heavy reductions on steels of the harder grades such as stainless and high-silicon steels.

Even with the rigidity provided by massive backup rolls, work rolls of the common four-high mills will flex or bend somewhat under the rolling pressures used; this tendency is greater in the wider mills. Accordingly, on mills rolling sheets and tin plate, work rolls are "crowned," that is, ground with roll diameters which increase very gradually from end to center. This convexity, which may consist of a diameter differential of 0.010-inch on the widest mills, compensates for normal roll flexure. On mills about 40 inches wide, the crown may be 0.001-inch or less, and on narrow cold-reduction mills it is usually unnecessary. The work rolls themselves are made of forged, hardened steel, surface ground to a high finish on specially-designed precision lathes. Work rolls for the last stand of a tandem mill, especially in a three-stand mill for sheets, generally are roughened slightly by shot-blasting to impress a matte pattern on the sheet as an aid in preventing sticking of stacked sheets in annealing. Work rolls for narrow cold-reduced strip, which requires a much brighter surface than sheets, are highly polished. Backup rolls on wider mills are generally made of cast steel and are always surface ground. They can be shaped "flat," that is, with no crown, or with up to 0.010-inch convexity.

SECTION 3

TYPICAL MILL LAYOUTS

Four-High Tandem Mills—The bulk of the cold-reduced flat steel made in this country is rolled on three-, four- or five-stand four-high tandem mills. Most tin plate falls in the thickness range of 0.008 to 0.014-inch and requires 80 to 90 per cent cold reduction from the thinnest hot-rolled breakdowns in coil form available in the width range of 24 to 36 inches. Tin plate, therefore, usually is rolled on five-stand mills around 42 to 48 inches wide. Most cold-rolled sheets for

automotive bodies, agricultural implements, architectural use, furniture and household equipment are required in thicknesses of 0.025 to 0.065-inch and widths of 30 to 72 inches. Material for these applications, most of which is given 45 to 65 per cent cold reduction, generally is rolled on three-stand mills producing material ranging from 54 to 98 inches in width. Four-stand mills, usually 48 to 56 inches wide, may be used to roll tin plate but are best adapted to produce 0.015 to 0.030-inch

thick, 24 to 47-inch wide sheets for such applications as roofing, signs and containers. Maximum delivered cold-reduced strip speeds on such tandem mills may run as high as 1000 feet per minute on a three-stand, 3000 feet on a four-stand, and 6000 feet on a five-stand mill. The average operating speeds vary greatly throughout the industry.

All such mills have uncoiling reels, cradles, or boxes from which the coil is fed into the actual roll train, coil ends being started either by hand or by pinch rolls. At the discharge end the finished coils are re-coiled on a mandrel, being started by a belt wrapper (a continuous driven fabric belt bearing against the mandrel), which guides the head end of the cold-reduced strip around the mandrel and is withdrawn when the re-coiling has been so started.

Mill housings are massive castings, similar to those on finishing stands of a hot-strip mill. On most tandem mills, both work rolls of each stand are driven through a pinion arrangement by a single motor, up to 5000-horsepower motors being used, although some of the latest mills have individual motors for each work roll on some stands. For the first stand or stands, in which the rolling speed is relatively low, motors are geared down; direct drive may be used for one or more of the remaining stands of such a tandem mill.

The 48-inch five-stand tandem mill at United States Steel Corporation's Irvin Works serves as an example of a unit of this type (Figure 34-4). This mill rolls a pickled, 12,000 or 24,000-pound, hot-rolled breakdown in coil form from approximately 0.085-inch thickness down to an average gage of 0.010 inch at a speed of 3300 feet per minute. The five stands, plus the reel, require a total of 11,000 horsepower for driving. From the moment when the head of the coil has been threaded through all stands until the mill reaches top speed, roughly 40 per cent of the horsepower is used for acceleration; the balance is required for reducing the metal. On this mill, the motors on No. 4 and 5 stands and also on the reel are of the double-armature type. Such motors provide a marked reduction in accelerating time and improved control at high speeds due to their lower inertia. The work roll and backup roll diameters are 21 inches and 53 inches, respectively. All work rolls are equipped with roller bearings while the back-up rolls are equipped with oil-film bearings. On an average, the work rolls in No. 5 stand are changed for redressing the surface after rolling every 150 tons; No. 4 stand, after 250 tons; and the first three stands after rolling somewhat over 1000 tons. Back-up rolls on the 5-stand tandem mill average approximately 20,000 tons between dressings. The cold-reduced strip tension reel at the delivery end is designed to pick up the end of the strip coming from the mill, grip it slowly and put it under tension without sudden or undue stress. The reel also winds the cold-reduced strip under constant tension, slowing down automatically without altering the tension as the coil increases in diameter. Such control is necessary to prevent tearing of the very thin, hard, cold-reduced strip. The reel is of the collapsible type and easily stripped by an air-operated plunger.

Four-High Reversing Mills—A typical four-high re-

versing mill consists of a single stand with reels located on each side of the mill. The mill itself is essentially the same in design and arrangement as the individual stands of a tandem mill. In the reversing mill, the steel must be passed back and forth until the required reduction is obtained. On the entry side of the mill, means are provided for uncoiling and feeding the coil through the mill to the tension reel on the delivery side. After the first pass, the tail end of the coil coming from the uncoiler is gripped by the second tension reel on the entry side of the mill. In each pass, the reel serving as the payoff unit is operated as a generator, providing back-tension to minimize rolling friction and feeding of current into the drive-reel motor. On the last pass, the tail end of the coil is released from the unwinding tension reel, completely wound on the winding reel and stripped in a manner similar to the action on the delivery reel of the tandem mill. For the rolling of tin plate product, a reversing mill usually requires five passes; for relatively heavy sheet product, usually three passes. All passes are considerably slower than the delivery pass of a tandem mill. From this it can be seen that a reversing mill is inherently flexible but cannot compete with a modern tandem mill from a production or cost standpoint where large tonnages are involved. Low installation costs in comparison to tandem mills make reversing mills popular for the production of specialty items that vary widely in dimensions and are ordered in small tonnages for each specification.

Two-High Cold Mills—Two-high cold mills may be used singly or in tandem for reducing narrow coils. Low-carbon steel may be reduced as much as 50 per cent on such mills, usually in four passes. In practice, however, no more cold working is done than is necessary to obtain the finish desired, and it has been found that best surface finishes can usually be obtained in a four-gage reduction; i.e., from 16 to 20 gage. The general practice on two-high cold-rolling mills is to roll coils down to the finished gage but not less than 0.020 inch in thickness for widths up to the diameter of the rolls. For coils under 0.020 inch thick, this relation between width of steel and diameter of rolls decreases, until for a thickness of 0.010 inch, the width of steel is about half the roll diameter. Aside from surface finish and control of hardness, the chief object of cold rolling narrow coils on the two-high mills is to obtain product as free as possible from camber, buckles, ruffles and gage irregularities of the hot-rolled product. These defects are overcome to a high degree by the process of rolling under tension. Unwinding reels with sufficient drag capacity, together with a delivery reel of sufficient power to stretch the steel, are a necessary part of any modern two-high mill installation for the rolling of narrow coils. In addition, most mills rolling narrow coils are equipped with rolls to control the form of the edges of the steel and also to control its width. These edge rolls are relatively light in construction and are grooved to produce the required edge. They revolve on vertical axes and are mounted in housings immediately in front of the horizontal work rolls. Edge rolls may be power driven or the steel may be pulled through idle edge rolls by the horizontal work rolls.

SECTION 4

DISPOSITION OF PRODUCT

Disposition of Strip After Cold Reduction—Disposition of the cold reduced, hard strip depends on prior processing and ultimate use. Much narrow high-finish strip is sold as cold reduced, for "full-hard" applica-

tions, on which heavy cold reductions have been used, or as intermediate tempers, in which the cold-reduced strip has been reduced a predetermined percentage from hot-rolled breakdowns in coil form or inter-

mediate-annealed gage to develop specified mechanical properties.

For sheet products, the coil may be taken to an uncoiling, side-trimming and sheet-shearing line, where it is cut to sheets of accurate size prior to heat treating. When annealing is to be done in coil form, the coil is

either recoiled through a side trimmer or, if side trimming to width has been done on the continuous pickling line before cold reduction, the coil is conveyed directly from the mill to the annealing department. For tin plate, the coil is subjected to an alkaline cleaning process for removal of surface dirt and oil prior to annealing.

SECTION 5

COLD-REDUCED STRIP CLEANING

In order to satisfactorily coat tin and terne plate either by the hot-dip or electrolytic process, it is necessary to clean the cold-reduced steel to remove the lubricant used in cold reduction, since the lubricant film left on the steel will decompose during annealing, and leave undesirable residues of carbonaceous material. Such residues are not removed satisfactorily in the subsequent pickling operations. Without exception, cold-reduced strip cleaners currently employ alkaline detergent solutions to remove the rolling-mill oils or solutions. The most commonly used reagents are caustic soda, sodium orthosilicate and trisodium phosphate; sodium metasilicate and sesquisilicate also have found

some application for certain types of product cleaning.

The type and amount of lubricant to be removed should be evaluated, as this may have a bearing on the required characteristics and operating conditions of the cleaning solution. Electrolytic action, while not universally employed, is believed to have merit. Figure 34-5 shows diagrammatically a continuous cold-reduced-strip cleaning line designed to employ the electrolytic principle. The detergent solution cleans the steel and serves as a conductor of electric current between the electrodes shown and the products being cleaned. Thorough rinsing of the steel after cleaning is essential to completely remove all contamination.

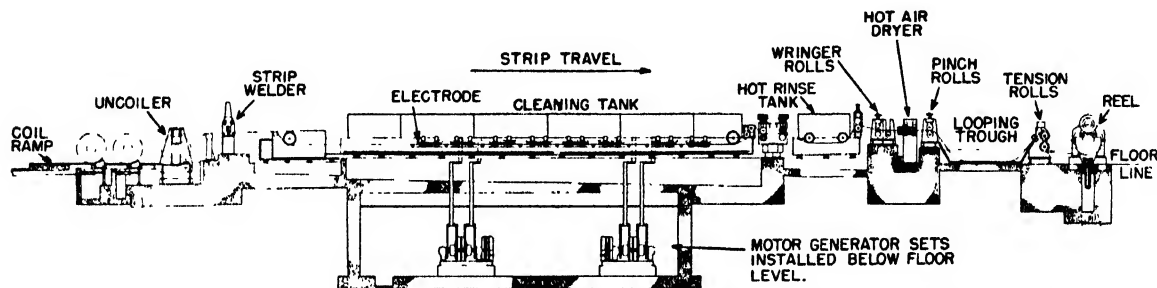


Fig. 34-5. Schematic arrangement of the equipment comprising a typical electrolytic cleaning line for processing cold-reduced steel.

SECTION 6

HEAT TREATMENT OF COLD-REDUCED STEEL

Purposes and Types of Heat Treatment—As is true of other steel products, sheets, strip and tin plate are heat treated primarily to effect changes in mechanical properties which will render the material suitable for the intended purpose. Other heat-treating objectives, involving relatively small tonnages of flat-rolled steel but highly important to the specialized commodities to which they apply, include: solution of chromium carbides to attain maximum corrosion resistance of austenitic stainless steels; development of optimum magnetic properties and establishment of the thin insulating oxide on silicon-bearing electrical sheets; dispersion or spheroidization of carbides to influence later heat-treating characteristics of alloy and high-carbon sheets.

Except for the very small proportion of "full-hard" cold-reduced strip and sheets used in the as-cold-reduced state to take advantage of the high strengths developed by cold reduction, some form of heat treatment is applied as a separate operation to all cold-reduced flat products to restore the ductility lost in cold reduction. In the case of hot-rolled sheets and strip made on a modern continuous hot-strip mill, supplementary heat treatment is usually unnecessary. On such hot-rolled material, rolling practices are used

which include definite finishing, piling and coiling temperatures as the steel leaves the last pass of the mill; the resultant "mill heat treatment" can be varied to provide mechanical properties in the as-rolled state which are satisfactory for most uses. This section will consider only those types of heat treatment conducted as separate operations.

Heat treatment of cold-reduced sheets, strip and tin plate may be divided into (a) "batch" operations—commonly called "box annealing"—in which a large stationary mass of steel is subjected to a long heat-treating cycle by varying the temperature within the furnace which surrounds it; and (b) "continuous" operations—including "continuous annealing," "strand annealing" and "normalizing"—in which a single thickness or a few thicknesses of cold-reduced sheet or strip are passed through a furnace in a relatively short time and receive a heat-treating cycle determined by the temperature distribution in the furnace and the dimensions and rate of travel of the steel.

Virtually all box-annealing practices slowly raise the steel to a temperature level at or below the lower critical point and "soak" it there for several hours. Such a cycle provides full recrystallization for steel in

the severely-deformed cold-reduced state and results in the softest possible finished product; accordingly, box annealing accounts for about 95 per cent of the heat treatment applied to cold-reduced steel and about 90 per cent of the heat treatment applied to all flat-rolled steel in the United States.

Depending on the nature of the steel and the results to be obtained, in continuous heat treatment the steel may be heated quickly to a maximum temperature at or somewhat above the lower critical temperature (continuous, strand or open annealing), or it may be heated slightly above the upper critical temperature (normalizing). In any case, the time at temperature is only a few minutes and the cooling rate is fast as compared to the hours-long cooling cycle of a box-annealed charge. The ferrite and structures resulting from transformation in cooling are fine-grained and the finished steel is generally harder than that produced by box annealing; continuous heat treatment is used, therefore, to attain properties not attainable by box annealing, but usually the product is somewhat harder.

While the great bulk of flat-rolled steel products can be heat treated adequately at temperatures around

1250° F, a few, because of their sluggish recrystallization tendencies after cold reduction, or their need for relatively quick cooling from the austenitic state to attain optimum properties, must be heated to temperatures ranging from 1400° to 2200° F. In such cases, continuous heat treatment is the most convenient method, as it avoids the hazard of pressure welding ("sticking") inherent in large masses of cold-reduced products held for hours at such temperatures. Continuous heat treatment is the only way to attain the quick cooling necessary to hold chromium carbides in solution in austenitic steels such as AISI Type 304 or to develop high strength levels in alloy sheets of the AISI 4130 type. It permits maximum reaction between a decarburizing atmosphere and the steel being treated (in single thicknesses) for such products as silicon-bearing sheets, whose magnetic properties require very low carbon contents. The controlled recrystallization possible in continuous heat treatment makes a fine-grained microstructure easy to obtain when such is desired, and minimizes the tendency for retention of directional properties of rolled steel which may be responsible for undesired irregular shapes in attempting to produce cylindrical stampings.

SECTION 7

EFFECTS OF HEAT TREATMENTS ON MICROSTRUCTURE

Box Annealing—Prior to cold reduction, low-carbon rimmed steel in the form of hot-rolled breakdowns has more or less equi-axed ferritic microstructure, with the carbides visible as pearlite or cementite (depending on whether the product was coiled cold or hot); it is relatively free of internal stresses, particularly if coiled hot and so "self-annealed." Cold reduction, however, elongates the grains from one- to ten-fold, greatly distorts the crystal lattice, and induces heavy internal stresses; the resultant product is very hard, with little ductility. This high degree of plastic deformation, however, renders the steel capable of returning to microstructural "equilibrium" by recrystallizing during heat treatment at temperatures well below those representing the thermal zone of conventional phase transformations. If such recrystallization is allowed to continue to completion by holding the steel at the proper temperature for sufficient time, the resultant structure again will consist of clearly-defined equi-axed ferrite grains with undistorted lattices and the steel again will be soft and ductile. In whatever state they were in the hot-rolled product, the carbides will have formed cementite, either small scattered spheroids (from hot-rolled breakdowns coiled cold) or massive agglomerates (from hot-rolled breakdowns coiled hot or from annealing slightly above the lower critical temperature).

Assuming sufficient time at the annealing temperature, steel given a heavy cold reduction will begin to crystallize at a lower temperature, will complete recrystallization more quickly, and finish with a finer ferrite grain, than steel given a light cold reduction. This is because the former material is more distorted before annealing and so has more centers of nucleation and higher localized stresses to induce the crystalline realignment. Similarly, those variations in practice during the hot-rolling of breakdowns which affect grain size of the product will affect similarly the microstructure of the cold-reduced, box-annealed steel, within the limits determined by steel grade, degree of cold reduction and annealing practice.

Recrystallization begins at each nucleation center with a return from distorted to "normal" atom alignment and is propagated by absorption of the surround-

ing distorted material into that alignment until the growth stops, establishing a single ferrite grain. Grain formation generally is stopped by the advancing fronts of differently-oriented adjacent grains. Within the practical range of recrystallization temperatures, however, the tendency of adjacent grains to assume the same lattice alignment, and to merge to form a larger grain, increases with increasing temperature. The maximum annealing temperature at which the steel is held for a significant time, therefore, determines the finished grain size for a given steel grade, hot-rolling practice and degree of cold reduction.

Beyond the period permitting full recrystallization—one to four hours at maximum temperature is sufficient for common steel grades and mill practices—time at subcritical temperature has relatively little effect on the grain size of the common steels. Extended "soaking" times do, however, lower the hardness for a given grain size, so that the annealing cycles applied generally hold the steel at temperature two to four times as long as is necessary for recrystallization.

A box-annealing practice developed to attain a given end result in grain size and physical properties, therefore, must balance several factors, some of them outside the annealing operation itself and invariable to any effective degree. Fortunately, minor variations in annealing practice have little effect on results and effective variations aimed at attaining lowest steel hardness values are not difficult to make or control.

Box-annealing equipment consists of annealing bases on which to place the steel charge, furnaces to apply the heat, and, generally, inner covers which fit over the charge in the furnace and contain the protective atmosphere which prevents oxidation of the steel (Figure 34-6). Each of these basic units may vary considerably in design, with little or much auxiliary equipment; in any steel plant several sets of units are grouped together into an annealing department, which is serviced as a whole by tracks, cars, tractors, cranes and atmosphere preparation equipment.

In most modern equipment, the bases are stationary and the portable furnaces are lowered by crane onto the loaded base and attached to fuel and control con-

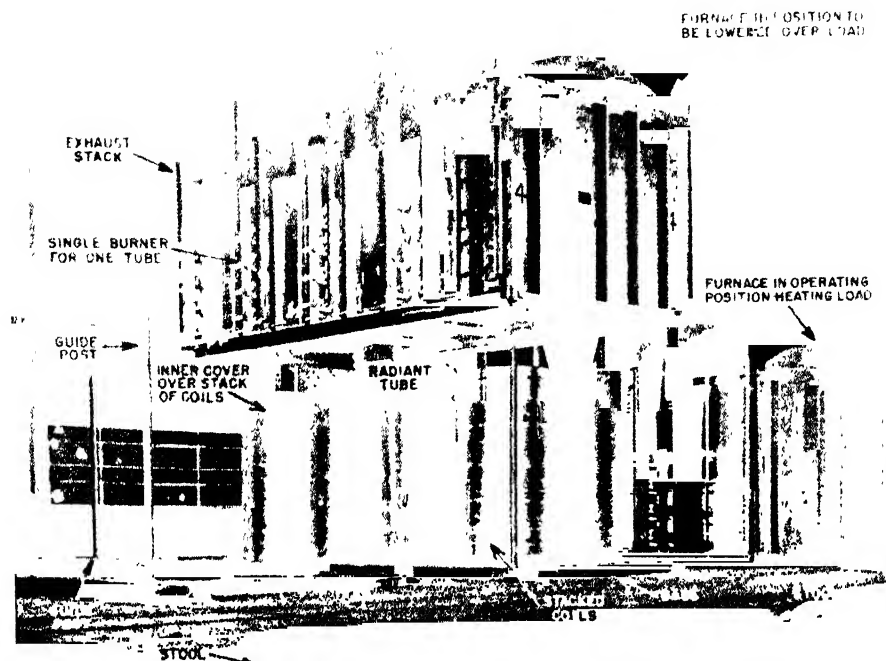


FIG. 34-6. A 500-ton capacity annealing furnace for coils. The furnace is being lowered by crane over the base loaded with eight stacks of coils. The burner tubes may be seen extending across the interior of the furnace. Note guide posts at the corners of the base.

nections for the annealing operation. To attain maximum furnace utilization, two, three, or four bases and an equivalent number of inner covers are provided for each furnace. Thus, no furnace time is wasted while the bases are being loaded or cooled to handling temperature. Some plants use stationary "in-and-out" furnaces with doors at one end; the loaded bases, with inner covers in place over the charge, are pushed into and withdrawn from the furnaces on tracked rollers.

Annealing bases are usually rectangular, although circular bases (and furnaces) are not uncommonly used for coil annealing of cold-reduced products. A base consists of a shallow tray of cast iron or, more commonly, refractory-lined steel. For sealing the open down-end of the inner cover, the bottom of the base may be covered with a layer of sand or the sand may be contained in a trough around the periphery of the base. Where expensive protective atmospheres are used, such as cracked ammonia (to protect high-carbon steel against decarburization), the trough into which the edge of the inner cover fits is filled with oil or low-melting alloy for perfect sealing.

Annealing bases for cut-sheet charges generally have a steel plate, set on the sand or raised on rails or piers, on which the piles of sheets rest to permit circulation of the contained atmosphere. For annealing coils, rectangular bases have two to eight raised "stools," each capped with an annular plate and generally containing a motor-driven fan for atmosphere circulation and better heat transfer. In addition, most bases are fitted with thermocouple inlets and atmosphere inlets and outlets.

The dimensions of a base are, of course, determined by the length and width of the furnace used on it. In the United States, they vary between 5 feet and 18 feet in width, 8 feet and 35 feet in length.

Inner covers are commonly open-bottomed thin-walled steel boxes or cylinders to fit, respectively, rectangular charges of cut sheets or cylindrical charges of stacked coils. These shells commonly are formed and welded either from $\frac{3}{16}$ to $\frac{1}{4}$ -inch thick low-carbon steel, or $\frac{3}{32}$ to $\frac{1}{8}$ -inch thick 18-8 columbium-bearing or 25-12 stainless-steel sheets. They may be strengthened by beading or corrugating the walls and the carbon-steel covers may be coated with aluminum oxide, sodium silicate or other protective compounds to reduce their oxidation rate.

The older in-and-out furnaces use thicker cast-iron covers to withstand the more severe oxidation conditions typical of open-flame firing.

Two innovations in inner covers are receiving some attention. In one design, the radiant tubes in which the fuel is burned are contained in the cover rather than in the furnace as in conventional equipment, and the "furnace" is simply a heat-insulating unit. This arrangement is intended to effect fuel economy and faster heating. Another new type of cylindrical cover has an inner shell to force the protective atmosphere being circulated by fans into better contact with the inner surface of the cover proper, and so effect better heat transfer and distribution.

Box-annealing furnaces are stationary or, more commonly, portable. Stationary furnaces, rectangular in shape, may be built singly or in batteries of two or four. Each furnace consists of a rectangular steel frame lined with 10 to 14 inches of fire-brick. Overall dimensions vary from 7 to 15 feet in width, 7 to 10 feet in height and 15 to 27 feet in length. Charging ends are fitted with hinged or counter-balanced doors, the latter traveling in vertical tracks. Floor tracks extend from the loading area into the furnace hearth and one or two charged bases are pushed into the furnace on cast-iron

balls or carried in on a charging machine. Such furnaces may be fired with coal, oil, or gas.

Stationary furnaces are seldom used for annealing coils. Most of them employ heavy cast-iron inner covers and cast-iron bottoms, which together weigh as much as the 15 to 25 tons of sheets being annealed. The resultant fuel consumption of 2 to 4 million Btu per ton of sheets annealed represents very low efficiency. More modern in-and-out furnaces, with light steel inner covers over the charge, can anneal up to 150 tons per charge with a fuel consumption of 1.2 to 1.6 million Btu per ton of sheets.

Portable annealing furnaces consist of a structural-steel frame covered with steel plate and lined with 6 to 12 inches of refractory insulating brick. The rectangular furnace generally used varies greatly in size and in annealing load weight, depending on mill requirements, handling facilities and effectiveness of design in providing temperature uniformity. Dimensions of furnaces now in use range from 5 to 16 feet in width, 6 to 14 feet in height and 8 to 33 feet in length. Loads of sheared sheets vary from 25 to 150 tons and of coils from 50 to 500 tons; fuel consumption ranges between 0.8 and 1.5 million Btu per annealed ton and annealing rate from 1 to 10 tons of steel per furnace hour.

Such furnaces commonly are gas-fired, although any fluid fuel may be used and electrical resistance heating has been tried. Burners usually fire into "radiant tubes" of 25-12 stainless steel. These tubes are 3 to 6 inches in diameter and as long as the furnace size and desired temperature distribution dictate. They may run vertically or horizontally along the inner walls of the furnace or, in large furnaces for annealing coils, may span the distance between the walls for better temperature distribution. The tubes vent the products of combustion on the outside of the furnace.

Heating of the inner cover thus occurs by radiation. Proper tube location and the flexibility possible in having each tube served by an individual burner permit the necessary uniformity of heating of the large masses of steel contained in one annealing charge. For minor adjustments, burners can be controlled individually. They are manifolded together, however, and the fuel flow to the individual burners of one furnace is usually determined by the main fuel valve of the furnace which is controlled automatically by predetermined tube-temperature control settings and by charge-temperature controls.

Some use is made of tubeless furnaces, with the burner discharging into a refractory baffle which prevents flame impingement on the inner cover but permits circulation of the hot combustion gases between the furnace wall and the inner cover. Convection plays an important part in heating the inner cover in this design. The decision to use a tubeless furnace will be influenced by the relative importance of its possible fuel economy, freedom from radiant tube costs, higher inner-cover costs and different temperature-distribution characteristics.

Box Annealing Practices—To begin an annealing operation in portable furnace equipment, sheared sheets are loaded on the base in piles 3 to 5 feet high and coils on the base stools in stacks 8 to 12 feet high; loading is done by traveling overhead cranes equipped with hooks, slings, retractable racks or magnets. Thermocouples are inserted in standard locations in the charge. One rectangular inner cover is lowered over the entire sheet charge or an individual cylindrical "ash can" inner cover is lowered over each stack of coils and settled in the sand seal. The furnace is then lowered onto the base, fuel line and thermocouple connections

are made, the flow of deoxidizing gas to purge the air from the space under the inner cover is begun, and the burners are ignited. Base fans, if available, are turned on to effect high-speed circulation of the atmosphere in the inner cover.

The subject of deoxidizing gas protective atmospheres is treated more fully elsewhere in this volume (see Index). It is sufficient to say here that it generally consists of the products of partial combustion of a fuel gas in a limited volume of air, these products then being treated to remove most of the resultant water vapor, dirt and carbon particles, and in some cases further treated to remove sulphur dioxide and carbon dioxide. The resultant mixture of 75 to 85 per cent nitrogen plus varying percentages of hydrogen, carbon monoxide, carbon dioxide and methane protects the bright steel surface from visible oxidation throughout the annealing cycle. Other effective atmospheres include dissociated ammonia and high-purity nitrogen.

During the heating-up period, fuel consumption is maintained at a constant rate until "tube" or "furnace" temperature, determined by a thermocouple connected to a recording-controlling instrument, reaches a predetermined level chosen to protect the equipment from damage. Fuel feed then is controlled automatically or manually so that this temperature is not exceeded. The next control point is reached when the thermocouple located in the hottest part of the load reaches a temperature preset to avoid overannealing or welding of the charge. Finally, the soak is begun when the "control couple" (either at the coldest spot in the charge or, if this is inaccessible, in a location bearing a known temperature relationship to the coldest spot) reaches the specified soaking temperature. Another fuel cutback takes place here, and the control-couple temperature curve flattens out through the specified duration of the soaking period. At the end of the soaking period, the fuel is shut off, the furnace is removed to begin a cycle on another base, and the charge is allowed to cool (still in a protective atmosphere under the inner cover) to about 300° F, when it can be exposed to air without oxidizing. The cooling period takes at least as long as the combined total time of heating and soaking.

The temperatures and times specified for annealing practices vary greatly from plant to plant, even in the manufacture of the universally similar dead-soft low-carbon sheet. This is true because of inherent differences in the cold-reduced steel to be annealed and differences in furnace size and design, load size and piling pattern, and thermocouple locations. As an example of the effect of one of these factors, heat conductivity through a load of sheets parallel to the sheet plane is somewhat less than through a solid mass of steel, but four to seven times greater than heat conductivity perpendicular to the sheet plane. A stack of coils consists of a hollow cylinder with a 16-inch to 30-inch inside diameter and a 36-inch to 84-inch outer diameter, receiving inner-cover radiant heat perpendicular to the curved planes of the layers of steel comprising the coils. This handicap can be overcome in part by blowing the atmosphere at high speed through the spaces inside and outside the coil stack, and so transferring some of the heat from the inner cover to the inner diameter of the coils. Another means is to separate each pair of coils in a stack with "convector plates," resembling flattened doughnuts with passages for the hot circulating atmosphere between top and bottom surfaces. Thus, some heat is transmitted to the ends of each coil, from which it has an easier path to follow.

Regardless of the wide differences necessary in specified control cycles, the coldest spot of an annealing

charge, whether or not actually accessible to a thermocouple under production conditions, will be found to have been annealed for 10 to 20 hours at a temperature in the range of 1225 to 1275° F in virtually all plants making deep-drawing rimmed-steel sheets, with the proper formula within these limits being determined by the prior history of the cold-reduced steel. Depending on furnace and load characteristics, the hottest part of the charge will have been at 1275° to 1350° F for 20 to 50 hours, and the furnace will have been under fire for 30 to 90 hours. When no steel decarburization occurs (as is the case with adequate prepared-atmosphere protection), the gross differences in time at temperature throughout an annealing load have little effect on sheet properties if the cold portion has been adequately annealed and so approaches the inherent limit of grain growth and steel softening.

Such temperature differences are more critical in annealing tin plate, where definite higher hardness levels are wanted. Partial protection against over-annealing exists in this case because the heavy cold reductions used limit the grain size to a relatively fine, and therefore relatively hard, structure, regardless of any annealing temperature employed below the transformation-temperature range. Such heavy cold reduction also induces full recrystallization at a lower temperature than is necessary on steel thicknesses given less cold reduction, and tin plate can be annealed fully in the slightly lower temperature range necessary to attain proper hardness.

For products other than low-carbon cold-reduced sheets, strip or tin plate, annealing practices can vary considerably from the outline given. Box-annealing furnaces are used occasionally at temperatures as low as 1000° F to stress relieve, rather than recrystallize, certain specialty steels, and temperatures up to 2000° F have been employed on other specialties.

Normalizing—The most common type of continuous heat treating employed at this time is normalizing, in which the sheet is heated above its upper critical temperature, around 1800° F for low-carbon steel, and cooled at a rate which permits the formation of the proper ferrite grain size. Normalizing is used only where box annealing is inadequate; typical applications include the heat treatment of alloy sheets to attain a fine pearlitic structure or the heat treatment of cold-reduced "pure iron" which recrystallizes sluggishly in conventional box annealing.

Normalizing furnaces are designed to heat and cool the sheets singly or in thin packs of two, three, or four sheets. Therefore, they are built in the form of long low chambers, usually in three sections, known as the preheating zone (12 to 20 per cent of the total length), the heating, or soaking, zone (about 33 per cent of the total length), and the cooling zone, which occupies the remaining 40 to 50 per cent of the length. In modern furnaces of the conveyor type, the only type adaptable for treating short lengths, the sheets are carried successively through each of these zones upon disc rollers made of heat-resistant alloys with polished surfaces to avoid scratching the sheets, and with the discs staggered to assure uniform heating of the sheets. The discs are mounted upon water-cooled shafts, which are driven by variable-speed motors through chains and sprockets or shafts and gears. These furnaces are built up to 100 inches in width and vary from 120 to 200 feet in length; fuel consumption is 2.0 to 4.5 million Btu per ton and production rates vary from 3 to 12 tons per hour. A typical furnace of this type was illustrated in Chapter 21. Normalizing furnaces usually are heated with gas or oil and do not employ protective

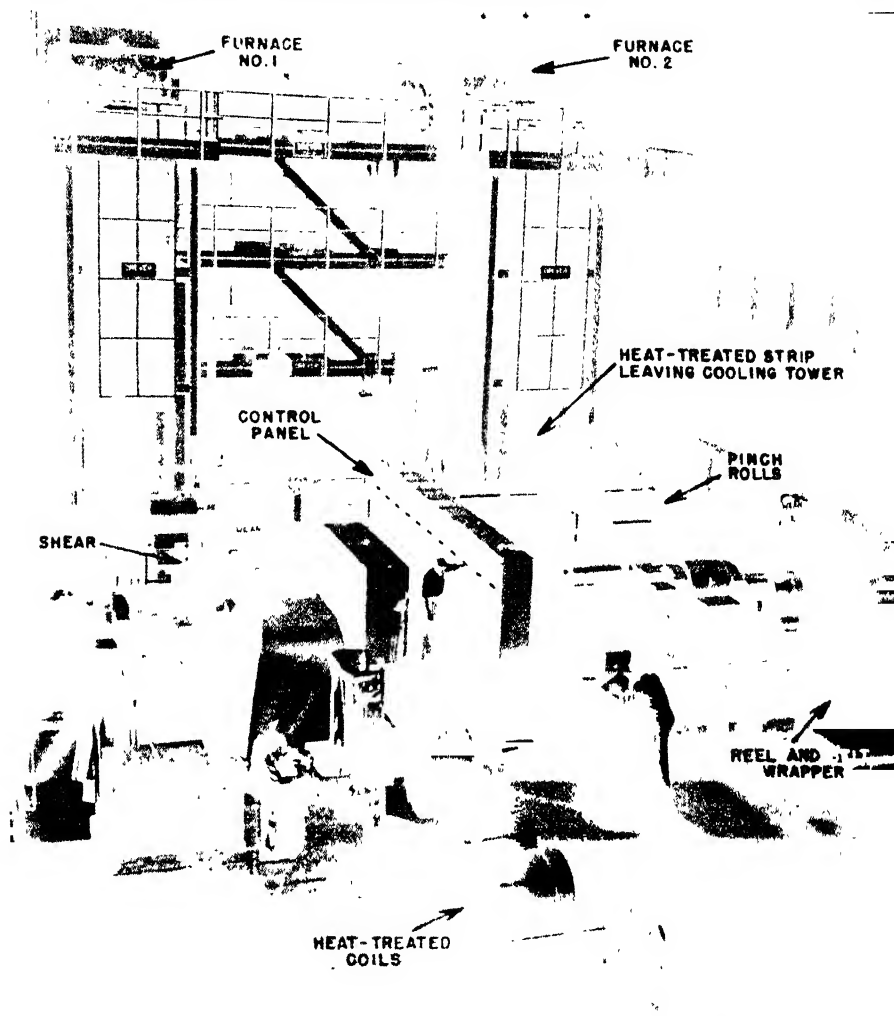
atmospheres. The sheets are, therefore, scaled during the heat treatment. Burners are arranged, along each side of the heating zone, usually above the conveyor rolls, but occasionally both above and below the conveyor. The roof, which is higher over the preheating and soaking zones than over the cooling zone, usually is built in sections. In most types, both the preheating and cooling zones are heated by the hot gases from the heating zone. This distribution of heat and gases within the furnace is obtained through two stacks, one being located at the entering end to draw the gases through the preheating zone and the other near the exit end to maintain a neutral atmosphere in the cooling zone. However, both the preheating and the cooling zones may be equipped with burners, the better to control their temperatures. Air is excluded by regulating the draft to maintain a slight pressure within all the zones. Heat losses are overcome by the generous use of heat-insulating materials. Furnaces designed to normalize sheared sheets are equipped with roller tables, one at each end, for charging and receiving the sheets.

The free-loop or catenary type of furnace is designed to normalize continuously cold-reduced steel unwound from coils, and neither rolls nor any other type of conveyor support material passing through the heating zone. The heating zones of these furnaces vary from 20 to 50 feet in length, and the preheating and cooling zones are usually shorter than in the conveyor type and may be omitted for some kinds of work. Such furnaces may have pickling or other descaling equipment at the exit end to remove, in the same operation, the surface oxides formed on the steel during normalizing. Catenary furnaces so equipped, but without a cooling zone, are widely used for the heat treatment of stainless steel, as the temperatures of 1900° to 2200° F used on the austenitic grades would result in short life for rollers used in the furnace. Steam or water quenching facilities are usually provided at the exit end of furnaces for heat treating stainless steel in this manner.

Assuming that the normalizing is to be done in a three-zone conveyor-type furnace, equipped with pyrometric controls, the operations appear extremely simple. If it is necessary to avoid all danger of scratching the sheets, they are brought to the charging table, and laid one or more at a time by hand upon a rider, or conveyor, sheet. The heaviest sheets (length, width and gage considered) are normalized singly, but lighter sheets may be laid two or more in a pile. Sometimes single sheets are laid upon a rider sheet and covered with another, called a cover sheet, to control heating and retard scaling. The sheets are carried by rolls into the preheating zone, where they at first absorb heat very rapidly because of the great temperature differential between them and the furnace interior. As the sheets become heated and this difference grows less, the absorption of heat becomes slower, so that after traveling 15 to 20 feet, the sheets enter the soaking zone several degrees below the normalizing temperature. Heating is completed in the soaking zone, which is maintained at a constant temperature, and sheets are at the necessary temperature for a time sufficient to convert the microstructure to austenite before they pass into the cooling zone. The sheet or sheets emerge from the cooling zone at a temperature which can be varied between 300° and 1000° F, and are conveyed for a short distance upon the run-out table, where they are cooled rapidly in contact with the air and carefully removed from the rider sheet. The trip through such a furnace is carried out at a uniform speed of 5 to 20 feet per minute and requires 5 to 20 minutes to complete.

Roller-hearth sheet-normalizing furnaces are some-

FIG. 34-7. Two tower-type continuous annealing furnaces for processing silicon steel.



times modified by removing the cooling zone and substituting air, steam, oil or water quenching facilities, the latter two as sprays or tanks. Such facilities are utilized to retain the austenitic structure developed in the heating zone on sheared sheets of stainless or Hadfield manganese steels, and to develop a martensitic structure (for subsequent tempering) in the higher-carbon alloy steels of the AISI 41xx or 86xx types.

Continuous Annealing—Continuous annealing of light-gage cold-reduced steel in a deoxidizing atmosphere has received increasing study since the modern cold-reduction process has provided single lengths of material thousands of feet long; its potentialities for tin plate, which generally requires higher hardnesses than sheets, are very attractive. In this treatment, a single strand of steel, cold reduced to the thin section used for tin plate, travels at high speed through a heating zone having a controlled atmosphere, where it is brought to a temperature just above the lower critical in a very short time, recrystallizes almost instantaneously, passes through a cooling zone and emerges into the air cold enough to avoid oxidation.

Such an operation provides fully-recrystallized steel, ductile in spite of its relative hardness and nearly free of directionality in physical properties. The extremely short time at temperature is effective because recrystallization has been suppressed by the rapid temperature

rise; the resultant increase in energy level at all potential nucleation centers causes the microstructure to "flash over" once recrystallization begins. From an engineering standpoint, the operation is made practical by building the heating and cooling zones as towers and increasing their effective length by threading the steel back and forth around rolls at the top and bottom of the towers.

A furnace of this sort for full-width tin plate is shown in Figure 21-21 of Chapter 21. In the direction of strip travel, the unit consists of a double pay-off reel; a welding unit; an alkaline electrolytic cleaner with a brush scrubber, water rinse, and drying unit; and a looping tower from which strip is taken to maintain constant line speed when a weld is being made. The furnace proper consists of a gas-fired "heating zone" wherein the strip makes six passes; an electrically-heated "holding zone" of six passes; an electrically-heated "slow-cooling zone" of two passes; a water-jacketed "fast-cooling zone" of twenty passes; a final cooling zone of two passes; another looping tower and two recoilers. Each pass consists of 55 feet of strip and all passes are vertical. The strip is heated to approximately 1200° F in the heating zone and maintained at this temperature in the holding zone. Upon leaving the holding zone the strip is cooled to about 1000° F in the slow-cooling zone and thence to about 240° F in the

fast-cooling zone. During this period, the strip is protected from oxidation by a gas atmosphere consisting of 95 per cent nitrogen and 5 per cent hydrogen. Subsequent cooling is done in air. At an operating speed of 900 feet per minute, only two minutes elapse from the time a given section of strip enters the heating zone to the time it leaves the fast-cooling zone.

Two continuous furnaces of somewhat different design were put into operation in 1949 at the Vandergrift plant of United States Steel Corporation (Figure 34—7). These units were built for heat treating high-silicon electrical steels, which require precise recrystallization control at higher temperatures and very low carbon contents to attain optimum electrical properties. With a floor length of 175 feet, the furnaces utilize the tower principle to permit several hundred feet of steel to be treated at temperatures up to 1900° F for controlled lengths of time, and under atmospheres of varying de-

carburizing power, as determined by the desired magnetic properties of the product. Speeds up to 240 feet per minute and production rates up to 10 tons per hour are possible.

Chapter 35

THEORY OF CORROSION

SECTION 1

CAUSES OF CORROSION

The Mechanism of Corrosion—The corrosion of the common metals in usual environments is an electrochemical phenomenon. That is, it is associated with the flow of electric currents over finite distances. Electric currents associated with corrosion have been detected in numerous cases, and in a limited number of instances the amount of corrosion occurring has been accounted for quantitatively by the amount of electric current which passed.

Knowledge that corrosion is electrochemical is important, since it assists in the development of methods for combating corrosion. For instance, it is obvious that, in order for electrochemical corrosion to occur, there must be differences in potential between different areas of the corroding structure. Such differences can be caused by the use of dissimilar metals or alloys in contact with each other. However, differences in potential can be caused by heterogeneities of any kind in the metal surface or in the environment contacting the metal. Some of the most important of these heterogeneities will be discussed below.

FACTORS WHICH AFFECT CORROSION RATE

The fundamental reason why metals corrode is that the corrosion products are more stable than the metals themselves. Thermodynamically speaking, in order for metals to corrode there must be a decrease in free energy associated with the formation of the appropriate corrosion product from the metal. This free energy decrease is the **driving force** of the corrosion reaction. However, the magnitude of this driving force gives little information regarding the rate at which corrosion will occur. The rate of corrosion is determined by other factors which will be described later in this chapter.

An **electrolytic cell**, in its simplest elements, consists of an **anode** and a **cathode** in a conducting solution (electrolyte). **Polarization** consists of a behavior at an electrode and results in what might be defined as a "back e.m.f.," which has the effect of opposing the flow of current in the electrolytic cell. Polarization thus is a phenomenon analogous to mechanical friction. **Hydrogen overvoltage** is one of the components of the total effect of polarization, and results from the deposition of hydrogen on the surface of one of the electrodes (anode or cathode). It will be recognized that polarization (and hydrogen overvoltage) always retard current flow and tend to lower rates of corrosion. Conversely, any factors that reduce polarization tend to accelerate corrosion.

Moisture—The presence of liquid or gaseous water is the factor of most importance in stimulating normal types of corrosion (although in special cases the presence of some moisture may retard corrosion). The reason for the customary stimulation in corrosion rate

caused by the presence of moisture is that moisture generally increases the electrical conductivity of the environment contacting the metal surface. Since corrosion is commonly electrochemical in nature, an increase in electrical conductivity of the environment will permit flow of larger electrical currents and, therefore, result in higher corrosion rates for given potential differences from point to point on the metal surface. Examples of the effect of moisture in stimulating corrosion are so well known that it is hardly necessary to elaborate greatly on this point. It has been established that even unalloyed steels will remain uncorroded if they are exposed to air with a relative humidity less than about 30 per cent. At higher humidities appreciable rusting will occur. Similarly, contact between steel and dry cloth or paper causes no serious attack of the steel, whereas, contact with damp cloth or paper may.

Salts—Neutral salts may stimulate corrosion in the presence of moisture by either or both of two mechanisms. They increase the electrical conductivity of the solution and thus increase corrosion currents. In addition, certain salts may form complexes with the metal corrosion products, thus increasing the solubility of the metal ion. This also stimulates corrosion. Local differences in salt concentration in a liquid which contacts a metal surface can cause severe localized attack as a result of the formation of concentration cells. These will be discussed in greater detail later.

Acids—There are two broad classes of acids:—(1) oxidizing acids such as concentrated nitric acid and (2) the non-oxidizing acids such as hydrochloric acid. The non-oxidizing acids stimulate corrosion by permitting the more rapid evolution of hydrogen as a result of reduction of the hydrogen overvoltage. Oxidizing acids may or may not be corrosive, depending on whether they form thin protective films on the metal surface. The so-called **passivity** of steel in concentrated nitric acid is the result of the formation of a thin, insoluble film on the steel surface upon contact with nitric acid above a certain concentration. Very concentrated sulphuric acid also forms a protective film upon contact with steel. Under usual service conditions, an increase in the acidity of a solution generally tends to increase its corrosivity.

Alkalis—For most ferrous materials, an increase in the alkalinity of a solution generally will tend to reduce the total amount of corrosion, although it may increase the intensity of attack at local areas. Very strong caustic solutions, particularly at elevated temperatures, or molten caustic materials also may be corrosive to a serious extent on account of the amphoteric nature of iron.

Oxygen and Oxidizing Compounds—Free oxygen and many oxidizing compounds have a complex effect on

the corrosion of steel. In aqueous solutions, they stimulate the total amount of corrosion but tend to restrict the area which is attacked. On the other hand, if sufficient oxygen or oxidizing compound is present in the solution, attack may be prevented entirely. This inhibiting action of oxygen and oxidizing compounds is of most importance in the case of stainless steels, while the accelerating action of dissolved oxygen is of great importance in the corrosion of unalloyed steels by natural waters and many chemical solutions.

The behavior of free oxygen and oxidizing compounds in general can be described in electrochemical terms. These materials stimulate the cathodic reaction by depolarization, but they tend to retard the anodic reaction by forming films on the surface of the anode. Thus, their behavior is complex.

In gaseous exposures at elevated temperatures, the presence of oxygen in the atmosphere generally results in increased rates of scaling or oxidation.

Sulphur Compounds—Sulphur compounds, either in solutions or in gas atmospheres, generally accelerate the corrosion of ferrous materials. Practically, such compounds give the most trouble in gaseous exposures at elevated temperatures. In the petroleum industry alone, corrosion of ferrous materials resulting from the presence of sulphur compounds in the products being handled, annually causes millions of dollars worth of damage. Apparently the sulphur compounds render the scale less protective, thus permitting increased rates of attack.

High Temperatures—In general, the higher the temperature of the exposure, the faster corrosion will proceed. There are many exceptions, since change in temperature can affect simultaneously several factors which all may influence corrosion rates. For example, raising the temperature of an aqueous solution exposed to air may either increase or decrease the rate of attack on metallic surfaces contacting the solution. Some of the effects of raising the temperature of a solution can be described in electrochemical terms. Increase in temperature increases the conductivity of the solution and also tends to decrease cathodic polarization. Both of these factors tend to stimulate corrosion. However, raising the temperature may decrease the concentration of oxygen dissolved in the solution but may cause more continuous films to be formed on anodic areas. Both of these factors tend to reduce corrosion rates. In practice, increase in the temperature of solutions freely exposed to the air usually results first in increasing the corrosion rate, but as temperatures above about 180° F are exceeded, the corrosion rate may decrease until the boiling point is reached.

Increase in temperature generally increases corrosion by gases, although here also special effects may come into play so that for some regions of temperature variation, relatively small increases in temperature may actually reduce corrosion rates. One of the most obvious examples of this is where the gas contains water vapor. At temperatures below the dew point, liquid water condenses from the gas and corrosion may be rapid. At somewhat more elevated temperatures, the humidity is less than 100 per cent, so liquid water does not form and the corrosion rate is much lower.

Galvanic Action—When dissimilar metals are placed in electrical contact and exposed to a conducting solution, generally corrosion of one member of the combination is accelerated while that of the other member is retarded. The metal of the combination which has the most active solution potential under the particular conditions of exposure is anodic and sends electric current through the solution to the cathodic metal. The

direction of the current flow determines the member of the couple which will suffer accelerated or **galvanic corrosion**. The magnitude of the current determines the corrosion rate. The electromotive force series indicates roughly which metal of the combination will be anodic and thus suffer special attack. However, since the potentials of the different metals vary with respect to different solutions, the electromotive force series is only a very rough guide.

For example, in sea water, steel is anodic to tin. Thus, contact with tin will accelerate corrosion of steel in sea water. However, when exposed to fruit juices under food container conditions, tin is anodic to steel and will tend to protect steel when in electrical contact with it.

The solution potentials of different alloys of the same base metal may be quite different in specific exposures. Thus, galvanic corrosion can be caused by contact between two different alloys of the same metal. In general, contact between any two metals or any two alloys in any conducting solutions should be suspected of resulting in galvanic corrosion unless there is definite evidence to the contrary.

In some cases, different metallurgical structures of the same metal or alloy have different solution potentials. Thus, corrosion cells are set up. This cause of corrosion should be suspected in welded structures unless they have been heat treated after welding.

Stray Currents—Stray direct-current electricity is a common cause of corrosion of underground steel pipe lines and other buried steel structures. It is also encountered in harbor structures of steel, steel-hulled vessels while tied up at docks, and in lake or river structures. In some cases, it has caused severe corrosion in chemical plants or other plants where large amounts of electrically conducting liquids are handled. Stray current causes more rapid attack than almost any other commonly encountered cause of corrosion. In fact, wherever very rapid attack of a metallic structure buried in the ground, immersed in water or exposed to conducting liquids is encountered, stray currents should be investigated as one of the most likely causes.

Alternating current electricity is much less likely to cause severe corrosion of ferrous structures unless it is of very low frequency (say one cycle per second).

Concentration Cells—When one portion of a metal surface is exposed to an electrolytically conducting medium which differs in any way from the electrolytically conducting medium which contacts another portion of the same metal surface, selective corrosion of the portion of the surface contacting one of the two types of media is likely to occur.* For example, suppose a steel tank is partially filled with a salt solution. If, for any reason, the concentration of salt is different in the layers of solution near the bottom of the tank than it is toward the top of the tank, comparatively more severe corrosion of the steel contacting either the more dilute or the more concentrated solution is likely to occur. This is true even though no corrosion would occur if the tank were full of solution having a uniform concentration from top to bottom. Corrosion in this case is caused by **concentration cell action**.

A special type of concentration cell action is caused by local differences in oxygen concentration of the liquid. For example, let us suppose that liquid saturated with oxygen is circulating freely in contact with most of the inside surfaces of a steel vessel. However, there is a narrow channel or fissure in some part of the surface where stagnant liquid is trapped. The oxygen content of this stagnant liquid will generally be less than that of the freely circulating liquid in contact with it.

* References are at end of chapter.

Thus, an oxygen concentration cell is set up which causes more severe corrosion of the metal surface in contact with the portion of the liquid having the lower oxygen content.

Oxygen concentration cells are a very common cause of corrosion under service conditions. The severe localized corrosion at joints and crevices, and on surfaces in contact with wet insulating materials, is largely the result of oxygen concentration cells.

Stress—Applied or residual stresses, either static or dynamic, can greatly accelerate corrosion.¹ Generally speaking, the acceleration of corrosion by static stress is greatest in environments which do not cause appreciable general corrosion. In fact, stress corrosion often occurs under conditions which would cause almost negligible attack in the absence of stress. A particularly dangerous form of stress corrosion is **stress corrosion cracking**. Under some specific sets of conditions, a stressed part of almost any metal or alloy may suddenly crack, although there may have been no appreciable corrosive attack evident prior to the sudden development of the stress corrosion crack. Stress corrosion cracks rarely occur when metallic parts are stressed appreciably below the yield strength. Since design stresses for most stressed assemblies are normally kept well below the yield strength, it is normally not the stresses for which the structure was designed which cause stress corrosion cracking. Instead, it is normally the residual or so called "internal" stresses which cause this type of failure. These residual stresses are induced during the fabrication of the structure, by bending, welding and other fabricating procedures. The magnitudes of such stresses are frequently not known to the design engineer and are consequently not considered by him in designing the structure.

Cyclic stresses also frequently accelerate corrosion. In cases where the frequency of the alternation in cyclic stresses is high, failures accompanied by corrosion are said to be caused by **corrosion fatigue**. Cyclic stresses, especially if of high frequency, normally cause more rapid failures when accompanied by corrosion than do static stresses of the same magnitude.

Abrasion, Erosion and Cavitation—Surface effects such as abrasion or wear, erosion, impingement of

liquid at high velocity, or cavitation effects caused by the collapse of gas or vapor bubbles at the metal surface can all contribute to intensified corrosion damage in specific cases.

Other Surface Effects—Practically any local difference at the metal surface can cause corrosion under suitable conditions. Thus, a scratch in the metal surface can form a weak point at which localized attack may occur. Local differences in temperature, velocity of liquid flow, degree of surface roughness and even amount of illumination can all cause localized corrosion in specific cases. In fact, any heterogeneity in environment, metal surface or, as will be indicated subsequently, in the metal itself, can cause localized attack.

Metallurgical Factors—As mentioned just above, any heterogeneities in the metal or alloy itself can give rise to localized corrosion. For most commercial metals or alloys these "internal" effects are normally a minor factor in influencing corrosion under service conditions. Generally, environmental factors are of much greater importance. This is very fortunate since it is impossible to avoid internal heterogeneities in commercial metals and alloys. All such commercial materials contain inclusions or minor segregations. Whenever localized attack develops in service, it is common practice to put the blame on these small particles of different phases. Actually, as has been shown by the work of Homer², Mears³ and others, very careful control of the uniformity of the corroding environment and of the metal surface is normally required to reveal the effect of inclusions on corrosion. In special cases, with specific metals or alloys, metallurgical factors may be important. For example, in the case of 18 per cent chromium-8 per cent nickel stainless steels, it is well known that slow cooling from the austenitizing temperature or reheating in a critical temperature zone for an appropriate time can cause the precipitation of carbides at grain boundaries, thus sensitizing the alloy to intergranular attack by certain media. In this case, the metallurgical condition is of great importance in controlling corrosion behavior. However, the fact remains that, in general, the metallurgical condition of most commercial materials is not of great importance in affecting corrosion rates.

SECTION 2

METHODS OF PREVENTING CORROSION

A knowledge of the factors which accelerate corrosion is of value to most people only as it aids them to understand and to guard against corrosion failures in service. A knowledge of appropriate methods of preventing corrosion is important for the same reason.

Material Selection—The most obvious method of preventing corrosion is to build the structure of a material which is unaffected by the service. Unfortunately, it is not always feasible to do this. The most inert materials may be too expensive or otherwise unsuited for the article to be built. Generally, the engineer must make a compromise. He cannot afford to use the most corrosion-resistant material but instead must compromise on a material which has the lowest combined initial cost plus maintenance costs for some selected period of time. The more accurately the engineer knows the corrosion behavior of the various competitive materials under the desired service conditions, the more accurately can he select the most economical material to use. As a guide to proper selection, nothing has yet replaced previous service use. Since small variations in service conditions can sometimes affect corrosion rates

greatly, even previous service use is not infallible. Nevertheless, it is the most trustworthy criterion available.

If it is desired to select material for equipment required for some new process or chemical, there will be no previous background of experience. In such a case, the engineer must be guided by knowledge of the behavior of various materials when used as equipment for similar processes. Better yet, a pilot plant or small-scale service test can be made using a material or materials of construction deemed likely to be satisfactory. The selection of materials for these small-scale tests can be based on laboratory tests or published information. In conducting the laboratory tests, it should be kept in mind that the closer these tests can be designed to simulate actual practice, the more accurately can the results be used to predict satisfactory service performance.

Selection of a suitable material of construction may eliminate the need of using any other form of corrosion prevention. However, it is frequently more economical to use some less resistant but cheaper material and to

employ one or even several protective measures. In most cases it is not economical to use the most resistant material of construction, but instead to use the cheapest material which will do a satisfactory job.

Appropriate Design—It is frequently overlooked that small changes in design may make it possible to use cheaper materials of construction.⁹ For instance, it might be feasible to use carbon-steel pipe as a vapor line for handling gaseous chemicals if the line were insulated to prevent condensation. If the insulation is omitted, such severe corrosion of the steel might occur that it would be necessary to employ stainless steel or non-ferrous pipe in order to reduce corrosion to a tolerable value.⁴

Similarly, the design of a processing tank might be such that liquids could lodge in crevices, pockets, joints or other dead spaces. Severe corrosion is likely to occur in such areas as a result of concentration cell formation. By altering the design to eliminate these regions, the same material of construction can be used with greatly reduced corrosion damage. In the case of tanks or processing vessels for chemicals, the most severe corrosion often occurs on the *outside* of the vessel, not on the inside as might be anticipated. Such equipment should be designed so that moisture will not be trapped in external joints between the vessel wall and the supporting members, between the bottom of the vessel and the supporting wood or concrete base, or in absorptive thermal insulation which contacts the external surfaces of the vessel. Moisture in these locations, even condensed moisture or tap water, can cause severe corrosion. To prevent corrosion, the design may be as important as the selection of the material of construction.

Protective Coatings⁸—Protective coatings are discussed in greater detail in Chapters 36 to 39, inclusive. There are a large number of different types of protective coatings. They can be classified in various ways. Based on their means of preventing corrosion, it is convenient to classify them as:

1. Anodic coatings
2. Cathodic coatings
3. Inert coatings
4. Inhibitive coatings

When a coated metal article is exposed to an electrolyte, if there are discontinuities in the coating, several possibilities present themselves. A positive electric current may flow from the coating through the solution (electrolyte) to the base metal. If this happens, the coating is anodic to the base metal. Furthermore, if the current density at the exposed area (or areas) of the base metal is of the correct magnitude, corrosion of the base metal will be prevented. Thus, anodic coatings tend to prevent corrosion of exposed areas of the base metal by sending electric current to them through any contacting film or layer of an electrolytically conducting medium.

In contrast to this, cathodic coatings tend to stimulate corrosion at exposed areas of the base metal under similar conditions of exposure.

Coatings showing the most pronounced anodic or cathodic behavior are the metallic coatings. Non-metallic coatings, especially oxide or sulphide coatings under some conditions of exposure, act as cathodic coatings. However, there is no definite evidence of non-metallic coatings acting as anodic coatings. It should also be pointed out that the same metallic coating on the same base metal can behave as an anodic coating under one set of exposure conditions, as a cathodic coating under other conditions, and as an inert coat-

ing or even as an inhibitive coating under still different conditions.

Tin coatings on steel form a good example of this variation in behavior under different environmental conditions. When exposed outdoors, in sea water, in most natural water, or even to many food products in the presence of air, tin is cathodic to exposed areas of the steel base. However, when exposed to nearly air-free food products, tin is generally definitely anodic to steel. Presumably, if the exposure were to food products containing some critical amount of oxygen, tin would be neither anodic nor cathodic to steel. That is, it might then be classed as an inert coating. Again under different conditions, the accumulation of dissolved tin compounds from the coating in the food product might reduce greatly the corrosiveness of the product to steel. In this last case, the tin compounds would be serving as corrosion inhibitors; therefore, the tin coating could be classed as an inhibitive coating.

It should also be pointed out that, although cathodic coatings tend to stimulate corrosion of exposed areas of the base metal, this does not always mean that increased attack of these areas will in fact occur. For example, if the cathodic coating is thick and if the exposed areas of base metal are small, attack may be stifled by plugging of the small pores in the coating with corrosion products from the base metal.

Inorganic coatings are sometimes inert, sometimes cathodic and sometimes inhibitive. Organic coatings are generally either inert or inhibitive. Either inorganic or organic coatings which contain water-soluble chromates generally function as inhibitive coatings in most natural environments. It is obvious that the inhibitive value of such coatings is greatest when there is only limited opportunity for leaching of the soluble inhibitor to occur. For example, it would be expected that the inhibitive action of a coating containing a soluble chromate would be more in evidence if the coating were on the interior of a tank which contained only a small amount of stagnant water than if it were on the interior of a pipe through which unrecirculated water was passing continually.

Organic coatings can also function in a manner which does not permit their classification in one of the four simple groups mentioned above. For example, in special cases, organic coatings when exposed to liquid media can serve as semipermeable membranes. Then, by osmosis, dialysis or electro-dialysis, liquid can be transferred through the organic coating to the metal-coating interface. The composition of the liquid which collects at the interface may differ markedly from that of the liquid on the outside of the coating. It may be either more or less corrosive to the base metal than the parent liquid. This behavior is still obscure but it is known to exist.

Treatment of Environment—Corrosion can sometimes be prevented either by adding something to the corrosive medium or by removing some corrosive agent from the medium.^{8, 10} For example, a certain tap water may be highly corrosive to the steel tank in which it is stored. By adding a corrosive inhibitor, such as sodium chromate, to the water the attack may be prevented. Alternatively, by removing dissolved oxygen from the water, the corrosion rate may be greatly reduced.

Corrosion prevention by treating the environment is normally employed when there is only a limited amount of the corrosive material. Thus, it is more widely used for waters which are recirculated than for waters which are flowing continuously from the source without recirculation.

Chromates are by far the most versatile corrosion in-

hibitors, although phosphates, silicates and various complex organic compounds are also used. The mechanism of inhibitor action differs for the various inhibitors and types of uses. In some cases, inhibitors function simply by forming a protective film or layer on the metal surface. In others, they retard one or more of the electrolytic processes necessary for corrosion to occur.

Inhibitors can be classified according to their behavior as "safe" or "dangerous." A safe inhibitor is one which will not cause intensified attack at local areas, even when added in insufficient amounts to prevent attack completely. A dangerous inhibitor will stimulate attack at a few local areas while reducing attack at most of the exposed areas. Unfortunately, most of the inhibitors at present used in other than mineral acid media are dangerous.

Inhibitors can also be classified as "expansive" or "contractive." As the terms imply, expansive inhibitors increase the area of the metal which is attacked, while contractive inhibitors reduce the attacked area.

Thus, a specific inhibitor may be classified as "safe, expansive," "dangerous, contractive" or "safe, contractive."

Cathodic Protection¹—It has already been mentioned that some coatings have the property of sending an electric current through an electrolyte to exposed areas of the base metal. This current flow tends to prevent corrosion of the base metal. Protection in this manner by current flow from any source is termed **cathodic protection**, since the metal being protected is made the cathode of an electrolytic cell.

Sometimes it is not convenient to use a coating metal to provide cathodic protection. Cathodic current from pieces of an anodic metal in electrical contact with the article to be protected and also exposed to the electrolyte can be used. These are termed "galvanic anodes." Alternatively, direct current from a storage battery or from a generator, or rectified alternating current from a power line, can be used. The only essential is that a sufficient cathodic current density must be maintained at all areas of the protected article which contact the corrosive solution.

Determining the magnitude of the current density just sufficient to prevent corrosion under various types of service conditions is a difficult problem. Fortunately, for most ferrous structures, it is not necessary to know the limiting current density with great accuracy. Any cathodic current density which is applied to a given area will reduce corrosion. Furthermore, if the limiting current density is exceeded, no harm is done to the structure. This means, simply, that protection is costing more than necessary because some of the current is being wasted.

Cathodic protection is a very effective way of preventing corrosion by most types of electrically conducting media. Also it is relatively safe, since as long as all the current flows to the structure, corrosion will be reduced even if protection is not complete. It is relatively easy to apply cathodic protection to small, geometrically simple structures. However, skilled electrochemical engineers are required to develop efficient cathodic protection for large or geometrically complicated structures.

At the present time, cathodic protection is applied widely to the steel hulls of marine vessels, to the interiors of small and large steel water tanks, to the ex-

teriors of buried steel pipe lines and to a variety of types of chemical equipment. It should also be mentioned again that the usefulness of tin plate for food containers and of galvanized sheet for roofing, siding and the like depends upon cathodic protection of the steel base by the coating metal.

Periodic Cleaning—It is frequently overlooked that periodic cleaning may greatly reduce corrosion damage. The fundamental reason why cleaning is beneficial is that it removes moist layers of solid matter from the metal surface. Corrosion products may be hygroscopic and generally are water sorptive. Thus, a metal surface coated with a heavy layer of corrosion products and exposed to the atmosphere will be wet for a considerably greater proportion of the time than will a similar clean surface. In like manner, layers of dust or soot of many types stimulate corrosion. Some materials hold moisture, others contain soluble products which in themselves are highly corrosive. It is only rarely that clinging debris is protective.

Chapter 36

PROTECTIVE COATINGS FOR STEEL

SECTION 1

PURPOSE OF PROTECTIVE COATINGS

Functional Coatings—All metals, when exposed to certain media, will corrode, the extent of corrosion depending on the nature of the metals and the environment to which they are exposed. It is common knowledge that all steels, except the more expensive, high-alloy, corrosion-resistant and heat-resistant steels, commonly called "stainless steels," will rust or corrode eventually, if unprotected by a coating. The rate of corrosion will, however, vary when steels are exposed to weathering, underground or underwater conditions. The action of oxygen and moisture in air and of oxygen in water to produce rust is greatly accelerated by chemical contaminants in the environment. Some of the most noteworthy of these are acids, and certain salts of mineral acids, such as ferric sulphate and sulphur-dioxide gas which are present in many industrial atmospheres. The corrosion resistance of an unprotected steel varies greatly depending on composition and environment. In the atmosphere, it has been conclusively proven that the nature of the rust formed affects greatly the subsequent corrosion rate. Some rusts are tight and non-porous and are more protective. Steels containing a small percentage of copper (0.20 per cent), on exposure to atmosphere, form a tighter rust than steel containing little or no copper. Likewise the rust on many high-

strength, low-alloy steels is more adherent than that formed on the copper-bearing steels (see Figure 36—1).

To prevent or inhibit the formation of rust, however, in any environment a protective coating of some sort is required. These coatings are of many kinds, and the methods by which they are applied to the surfaces to be protected, must be adapted to the nature of the coating material. Coatings may generally be classified as metallic or non-metallic, the latter often comprising mixtures of various substances of organic or of both organic and inorganic nature. Many coatings do not adhere well to steel. Some are porous, not being wholly impermeable to the gases and liquids which may attack the steel base. To improve adhesion, pretreatments of the steel base are often used. Porosity may be lessened by the application of heavy coatings or of several coats of the same or of varied compositions if the purpose for which the product is to be used will permit such a practice.

Although coatings are applied mainly for lasting or long time protection, sometimes it is merely necessary that they afford temporary protection, to prevent rust in transit or storage, in which case it may be required that they be removed easily or possess properties that permit the steel to be used without their removal. In some cases the coating may have to be a good primer and

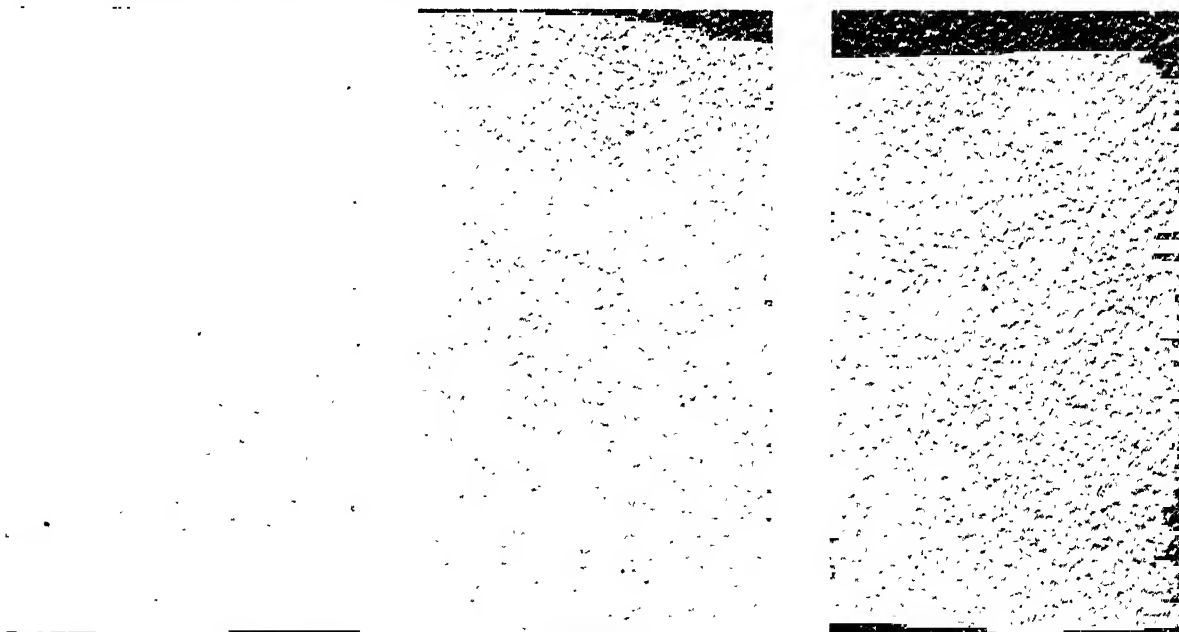


FIG. 36—1. Typical sheet samples of uncleaned high-strength, low-alloy steel (U.S.S. Cor-Ten) at left, copper steel at center, and low copper steel at right, after exposure to an industrial atmosphere for 3.6 years. The hard smooth adherent oxide on the high-strength, low-alloy steel is characteristic. The actual size of each of these three panels is approximately 4 inches by 6 inches.

compatible with coatings applied to the finished article or serve as a drawing compound to aid in the forming of the final product.

Coatings Combining Decorative and Protective Properties—Coatings may be not only protective but also decorative and, in fact, more often than not the feature of appearance is of prime importance. For example, galvanized sheets have often been required to exhibit a pleasing spangled surface. The presence of this spangle

or bright appearance has been associated with the highest quality product although tests have shown no longer actual service life than that of galvanized sheets of duller lustre with equivalent weight of coating. Thus, dependent on the required appearance of the ultimate product, a variety of bright or dull coatings may be manufactured. In the field of organic coatings, a variety of hues may be obtained by a variation of the pigments or stains used in the coating mixture

SECTION 2

PREPARATION OF STEEL FOR APPLICATION OF COATINGS

Importance of Clean Surface—Before application of any protective coating, it is imperative that the surface of the steel be thoroughly cleaned. This feature is neglected more often than any other by those unskilled in the application of a protective film and is often the cause of coatings of poor quality even when applied by those with experience. Scale from hot rolling or annealing should be removed from the steel by shot or sand blasting, by pickling in acid, or by flame descaling with an oxygen-acetylene torch, after which any loose dirt or acid should be removed by brushing or rinsing. This cleaning is important for the application of all types of coatings. Tight scale frequently is not removed on heavy steel sections prior to the application of priming paint, for economical reasons; in such cases the use of special inhibitive primers may result in satisfactory service. Nevertheless, experience has shown repeatedly that superior protection is obtained if all the scale has been removed. Removal of temporary protective coatings, drawing compounds, oil, grease, or dirt from shop operations is absolutely necessary. This may be done by solvents, by vapor degreasing, or by alkali cleaning. Rinses are used to insure thorough removal of cleaner from the metal surface.

Pickling—In large-scale operations, pickling in hot sulphuric acid (H_2SO_4) is a common practice to remove mill scale or rust. The time, concentration, and temperature will vary depending on the type of scale to be removed or the type of steel to be pickled. In continuous strip operations or where size and shape do not vary greatly, electrolytic pickling may be employed at room temperature. The article to be pickled is made the anode, if some actual solution of the steel is required, or the cathode where only cleansing by the blast of hydrogen evolved at the cathode is necessary. Cold muriatic acid (HCl) often is used in small shops to pickle steel and does so very efficiently, although this acid is more expensive than sulphuric acid. Other acids are not used as commonly, although phosphoric acid (H_3PO_4) is find-

ing wider usage where a phosphate coating is desired adjacent to the steel base.

Shot or Sand Blasting—The impingement of steel shot or sharp sand by a blast of air on the scaled or rusted steel surface imparts a very desirable surface texture. The surface finish created is usually quite rough and the roughness of surface is regulated to some extent by particle size of the shot or sand. This type of surface is desirable in that it promotes adherence of some coatings. A rough or eggshell finish may be imparted to steel sheets by the working rolls of a cold mill, which have been shot blasted. This produces a finish to which many protective and decorative coatings can be applied readily. Such sheets commonly are supplied to auto body makers.

Alkaline Cleaning—This method is used where mineral and animal fats and oils must be removed. Mere dipping in solutions of various compositions, concentrations and temperatures are often satisfactory. The use of electrolytic cleaning may be advisable for large-scale production or where this method yields a cleaner product. Caustic soda, soda ash, alkaline silicates and phosphates are common cleaning agents. Sometimes the addition of wetting agents to the cleaning bath will facilitate cleaning.

Solvent Degreasing—This treatment consists of cleaning steel by dipping in a commercial solvent, such as naphtha, trichlorethylene or carbon tetrachloride and rinsing in a clean solution of the same solvent or simply hand cleaning with rags wet with the appropriate solvent. Since the use of volatile solvents in the open constitutes a fire hazard, the less inflammable compounds are favored. Since the vapors of most solvents are toxic in nature adequate ventilation is necessary. Vapor degreasing, usually done in specially built machinery, is a similar process except that solvent vapor is the cleaning agent.

These methods are usually employed where organic coatings are to be applied.

SECTION 3

METALLIC PROTECTIVE COATINGS

The most commonly used metallic coatings include tin, zinc,terne metal (lead + tin), nickel, chromium, cadmium, copper, aluminum, bronze, brass, silver, gold, lead, iron and steel, and sometimes other alloy combinations of the above mentioned metals. A metallic coating may be anodic or cathodic to the metal to which it is applied for protection. If anodic it is "sacrificial" or less noble than the base. If cathodic, it is more noble and its protective value is due to its own relative chemical inactivity in the environment to which it is exposed.

A rough indication of the activity of the metals may be obtained from the "electrochemical" or "electromotive"

series, a classification of metals in the order of electrode potential referred to the standard hydrogen electrode at a temperature of 25°C (see Table 36-I). This table is often used mistakenly as if the order of metals were invariable, each metal displacing from solution or protecting from corrosion those below it. That this is often not true is shown in the second column of the table, where the values of potential in a normal salt solution are given. From the "electrochemical" series it would be anticipated that the corrosion of zinc would be retarded by contact with aluminum, while the potential measurements in salt solution indi-

Table 36—I. Comparison of the Electromotive Series and Solution Potentials in Sodium-Chloride Solution of The Common Metals and a Few Alloys

Metals or Alloys	Potential in Volts	
	From emf Series (Normal hydrogen scale)*	In 1 N (5.85%) NaCl containing 0.3% H ₂ O ₂ (0.1 N) Calomel Scale**
Magnesium, Mg	+ 2.37 ^(b)	+ 1.73 ^(a)
Aluminum, Al	+ 1.66 ^(b)	+ 0.85 ^(a)
Zinc, Zn	+ 0.76 ^(b)	+ 1.00 ^(a)
Chromium, Cr ⁺⁺	+ 0.74 ^(b)
Iron, Fe ⁺⁺	+ 0.44 ^(b)	+ 0.63 ^(a)
Cadmium, Cd	+ 0.40 ^(b)	+ 0.82 ^(a)
Cobalt, Co	+ 0.28 ^(b)
Nickel, Ni	+ 0.25 ^(b)	+ 0.07 ^(a)
Tin, Sn ⁺⁺	+ 0.14 ^(b)	+ 0.49 ^(a)
Lead, Pb ⁺⁺	+ 0.13 ^(b)	+ 0.55 ^(a)
Hydrogen, H ₂	0.00
Copper, Cu ⁺⁺	- 0.34 ^(b)	+ 0.20 ^(a)
Silver, Ag	- 0.80 ^(b)	+ 0.08 ^(a)
Gold, Au ⁺⁺⁺	- 1.50 ^(b)
Brass (60-40)	+ 0.28 ^(a)
Stainless Steel (18-8)	+ 0.15 ^(a)
Monel Metal	+ 0.10 ^(a)
Inconel	+ 0.40 ^(a)

* In some handbooks the sign of potential is reversed from that given.

** These values vary somewhat, depending on the particular lot of material investigated and on surface preparation employed.

^(a) Mears and Brown; Causes of Corrosion Currents. Industrial and Engineering Chemistry, Vol. 33, Page 1008, Table XII.

Note—The sign of potential is the reverse from that given here.

^(b) W. M. Latimer, Oxidation Potentials. Prentice Hall, Inc. New York (1952).

cate that the zinc should protect aluminum. In sea water and many natural waters, this protection of aluminum by zinc actually occurs. The electrochemical series is useful since metals near the top are generally protective to those near the bottom of the table. When, however, two metals differ little in potential, i.e., are close together in the first column of Table 36—I, one cannot predict which will protect the other, without actual tests. In many instances, the results of potential measurements in a salt solution (second column of Table 36—I) are a better guide as to the ability of one metal to protect the other

Table 36—II. Solution Potentials of Several Metals in Various Solutions (all 1 molar in concentration). Referred to a 0.1 N calomel half-cell*

Metal	Sodium Chloride	Sodium Chromate	Nitric Acid	Sodium Hydroxide
Magnesium	+ 1.72	+ 0.96	+ 1.49	+ 1.47
Aluminum	+ 0.86	+ 0.71	+ 0.49	+ 1.50
Zinc	+ 1.15	+ 0.67	+ 1.06	+ 1.51
Iron	+ 0.72	+ 0.16	+ 0.58	+ 0.22

* From: Light Metals for the Cathodic Protection of Steel Structures, by R. B. Mears and C. D. Brown; Corrosion, Vol. 1, No. 2, September, 1945. National Association of Corrosion Engineers. (Sign of potential reversed.)

under natural conditions of exposure than is the electromotive series. Many factors, such as the environment to which exposed, the magnitude of current generated, the relative area of metals exposed, the texture of metal surface and the inherent tendency of the metal to form an insoluble protective film notably affect the corrosion of metals themselves or their corrosion rate when used as coatings for steel.

Table 36—II illustrates the variable effect of environment or exposure conditions on the solution potentials of several common metals. It will be noted that the relative potentials of these metals vary when they are exposed to different solutions. Thus, no one table of solution potential values can indicate the electrochemical behavior of the different metals under all conditions of use. Actual tests under conditions similar to those of service are required before it is possible to make accurate predictions. The electrochemical behavior of metallic coatings on steel under conditions of atmospheric corrosion is now fairly well established. Zinc is anodic to steel under most exposure conditions and will prevent corrosion even at small discontinuities in the coating. The behavior of aluminum coatings is more complex. In some environments, they tend to protect exposed steel areas but in others there is no evidence of electrochemical protection. Tin, terne metal, nickel, copper, silver, gold and lead are all cathodic to steel under most conditions of atmospheric exposure and, if used as coatings for steel, will tend to accelerate corrosion at pores, scratches, and pinholes. In some cases, the corrosion products formed in the areas where steel is exposed, will stifle corrosion. This is particularly true for heavy coatings of lead, tin, terne metal and aluminum.

From the above it will be concluded that protection against corrosion is not a simple problem, but an extremely complex one.

METHODS OF APPLYING METALLIC COATINGS

Metallic coatings are applied to steel surfaces by the following methods:

Hot Dip Processes—The steel article to be coated is immersed, after thorough cleaning, in a molten bath of the metal forming the coating. Zinc, tin, terne metal, aluminum, and lead are applied commercially in this manner and are discussed fully elsewhere in this book (see especially Chapters 37, 38 and 39).

Metal Spraying—This method, introduced about 1910, may be used with most of the common metals including aluminum, copper, lead, nickel, tin and zinc and alloys such as brass, bronze, babbitt metal, monel metal and stainless steel. The coating metal usually is drawn into wire and fed into a specially constructed spray gun. This gun is operated with compressed air and a fuel gas. It is small and compact although it contains an air-gas mixing chamber, a special nozzle for burning the mixture and melting the wire, an outer compressed-air nozzle concentric with the inner nozzle, and an air turbine driving knurled rolls which draw the wire from its spool and feed it through the inner nozzle. The gases at the nozzle are ignited, the wire is melted as it is fed to the nozzle and is projected against the surface to be coated at a speed of over 500 feet per second. Although the particles of molten metal are cooled instantly to a temperature of about 80° F, the impact causes them to adhere firmly to the steel surface, provided it has been cleaned thoroughly, as by machining or by sand or shot blasting. Metal spraying is used for building up surfaces and sometimes for the application of thin coatings as a protection against corrosion.

Metal Cementation—The metals zinc, chromium, aluminum and silicon are successfully applied in this man-

ner, in which the protecting metal is alloyed into the surface layers of the steel.

In *sherardizing*, practiced since about 1900, the parts to be coated, usually small articles such as nails, are thoroughly cleaned by pickling or sand blasting; packed in metal drums with fine zinc dust, usually containing 5 to 8 per cent of zinc oxide; and heated for several hours at between 650° and 750° F, the drums being slowly rotated in the furnace during the heating. The coating is thin and consists of intermetallic compounds of iron and zinc, ranging from an iron-rich alloy next to the steel base to almost pure zinc at the surface, but it affords good protection against atmospheric corrosion. The process was invented by Sherard Cowper-Coles.

Chromizing is a cementation process analogous to sherardizing. The parts to be treated are packed in a container with a mixture of 55 parts of chromium or ferrochromium powder and 45 parts of alumina by weight. They are then heated "in vacuo" or in a protective atmosphere (preferably hydrogen) at 2370° to 2560° F for three or four hours, although a shorter time and lower temperature may be used when less penetration is desired. Although producing a high-quality corrosion-resistant product, the process has been superseded largely by chromium plating or by the use of corrosion-resistant alloy steels.

In *calorizing*, developed by General Electric Company about 1925 to 1930, the thoroughly cleaned steel articles are packed in steel drums containing a mixture of aluminum and aluminum oxide. A reducing gas is passed into the drum, which is rotated in the furnace and heated for about 5 hours at between 1730° and 1750° F. The resulting coating is said to be a solid solution of aluminum in iron, richest in aluminum at the outer surface, and is used principally to protect the steel from oxidation at elevated temperatures, as in pyrometer tubes, superheater tubes and oil-refinery equipment.

Thirigizing is a special type of siliconizing, or impregnation of the surface of low-carbon steels with silicon. In this process, the surface of low-carbon steel, freed of sand and heavy scale, is packed with silicon carbide or ferrosilicon mixed with mill scale (iron oxide), heated to a temperature of 1300° F or higher and exposed to the action of chlorine for two hours or more depending on the temperature used and depth of case desired. At this temperature the chlorine reacts with the carbon or the iron of the silicon-bearing substance, leaving the silicon in nascent form to combine with the iron in the steel. The siliconized layer, usually 0.005 to 0.1 inch thick as desired, is very hard and resistant to corrosion by non-oxidizing acids, such as hydrochloric and sulphuric acids, and to wear and oxidation at temperatures up to 1600° F, and is capable of absorbing and retaining substantial amounts of oil. In usual practice, the silicon content of the case remains practically constant for the first 0.040 inch, varying from about 14 per cent at the surface to 12 per cent at 0.050 inch below, then decreases gradually in the next 0.020 to 0.025 inch to the silicon content of the core. The process was developed in 1935-1938 by H. K. Ihrig (U. S. Patent Reissue 20719).

In *corronizing*, developed about 1938 by Standard Steel Spring Company of Coraopolis, Pennsylvania, the steel is electroplated with nickel and subsequently with zinc or nickel-zinc alloy (U. S. Patent No. 2,419,231). The plated steel may be heated to about 750° F to form a nickel-zinc alloy, if zinc alone is the final coating.

Metal Cladding—*Copper cladding* processes give bi-metal products. Usually those containing steel consist of an inner steel core covered with a heavy layer of copper. In the usual process, the steel core, with a clean surface, is mounted in a covered mold and heated out

of contact with air to a temperature slightly above the melting point of the copper, which then is cast about it. Other methods consist of dipping the solid steel core into a bath of molten copper, or of depositing the copper electrolytically. Starting material for copper-clad steel wire is made by forcing a steel rod into a closely fitting copper tube. Semi-finished products prepared by any of these methods may be heated to around 1700° F and hot rolled, then finished by cold rolling or drawing, as in forming copper-clad wire. The wire is used widely for electrical conductors, combining the strength of steel with the high conductivity of copper. Bundy-weld steel tubing is hydrogen-welded, copper-coated, rolled steel tubing. It is used for gasoline and oil lines in automobiles and for refrigerator coils. Copper-clad sheet steel was produced during World War II for the fabrication of copper-jacketed bullets.

Aluminum cladding is accomplished best by rolling flat steel almost to gage, cleaning it thoroughly and either placing it between two sheets of aluminum and cold rolling, or heating to between 600° and 750° and rolling. The latter method results in a better bonding of the aluminum with the steel. Subsequent annealing above 1000° F causes the aluminum to unite with the iron forming the very brittle FeAl. With basic Bessemer steel or open-hearth steel containing above 0.25 per cent silicon, the temperature of this reaction is raised above that for regular open-hearth steel so that the coated strip or sheet may be annealed at a somewhat higher temperature after cold rolling without becoming brittle.

Stainless cladding may be accomplished by (1) electro-welding stainless steel onto the carbon steel (2) casting the stainless steel around a solid carbon-steel slab or (3) placing a slab of carbon steel between two plates of stainless steel, and hot rolling them. In the last mentioned method, fluxes or metals have been used to facilitate bonding but are not necessary if both steels are cleaned thoroughly before making the "sandwich." Welding is usually done around the perimeter of the slab. Considerable care is necessary in the preparation of such duplex material to avoid formation of blisters.

Fusion Welding of Coatings may be accomplished in different ways, as by depositing weld metal under a slag covering by the electric-arc method, or by fusing the surfaces of two bodies of metal in contact by passing a current of sufficiently high density. These initial steps are followed by heating and forming operations carried out in the usual manner, as by rolling. Similarly, weld metal may be deposited upon metal of another kind to afford greater resistance to abrasion, such as manganese-nickel steel welding rod used to face excavating and similar tools.

Electroplating—This process is an old art, practiced not only to protect the base metal from corrosion but also for decorative purposes and, more recently, to protect the base metal from wear by friction or abrasion. Metals used for coatings include cadmium, chromium, copper, gold, tin, lead, nickel, silver and zinc, and alloys such as brass, bronze and lead-tin as well as cobalt-tungsten, tungsten-nickel and cadmium-tin alloys. It will be noted that with the exception of zinc, which is anodic to steel, i.e., protects by sacrificing itself, nearly all electroplated coatings are cathodic to steel and provide protection through surface coverage. The decorative coatings commonly used vary in thickness, according to the life required. Durability usually depends upon the properties of the coating, especially adhesion and porosity. The severity of conditions of exposure, particularly with reference to acidic gases in the atmosphere, also affect the service life. In plating, the preparation of the base metal is very important, not only in that a clean

Table 36—III. Electroplating Baths

Kind of Coating	Type of Coat	Typical Composition of Baths Water to make 1 Gallon	Operating Conditions			
			pH	Temperature (° F)	Amperes per Sq. Ft.	Volts
Nickel	Matte or Dull	40 oz. nickel sulphate, (1) 8 oz. nickel chloride, (2) 5 oz. boric acid.	1.5-2.5	110-125	40	4-6
	Bright	40 oz. nickel sulphate, (1) 6 oz. nickel chloride, (2) 4½ oz. boric acid and 1 to 2 per cent of addition agents.	2 to 5	125-140	10	2-3
	Hard	24 oz. nickel sulphate, (1) 3.3 oz. ammonium chloride, 4.0 oz. boric acid.	5.6-5.9	120-140	25-50	4-6
Chromium	Bright	33-55 oz. chromic acid, 0.35-0.55 oz. sulphuric acid. (3)	110-120	150-300	8-12
	Hard	35-55 oz. chromic acid, 0.2-0.7 oz. sulphuric acid. (3)	140-150	200-300	8-12
Zinc	Acid	32 oz. zinc sulphate, (4) 2 oz. sodium acetate, (5) 4 oz. aluminum sulphate, (6) plus 0.13 oz. addition agent (licorice).	3.5-5.0	75-120	25	4-6
	Cyanide	8 oz. zinc cyanide, 3 oz. sodium cyanide, 7 oz. sodium hydroxide, plus brighteners (mercuric salts) 1/16 oz.	12-12.2	105-120	10-20	4-10
Cadmium	3 oz. cadmium oxide, 14.5 oz. sodium cyanide, plus brighteners.	13.0	70-95	25	4-6
Tin	Alkaline	16 oz. sodium stannate, 1 oz. sodium hydroxide, 2 oz. sodium acetate, (5) 1/16 oz. hydrogen peroxide. (7)		140-180	10-40	4-6
	Acid	8 oz. stannous sulphate, 9 oz. sulphuric acid, (3) 7 oz. cresol sulphonc acid, plus addition agent.	70	35	1-2
Copper	Cyanide	3 oz. copper cyanide, 4.5 oz. sodium cyanide, 2 oz. sodium carbonate. (9)	11.8-12.2	75-120	15	1.5-2
	Acid	28 oz. copper sulphate, (8) 6.5 oz. sulphuric acid. (3)	70-80	30	4-6
Brass	3.6 oz. copper cyanide, 1.2 oz. zinc cyanide, 7.5 oz. sodium cyanide, 4 oz. sodium carbonate. (9)	11	75-100	3-5	2-3

(1) $\text{NiSO}_4 \cdot 7\text{H}_2\text{O}$ (2) $\text{NiCl}_2 \cdot 6\text{H}_2\text{O}$ (3) H_2SO_4 —100%(4) $\text{ZnSO}_4 \cdot 7\text{H}_2\text{O}$ (5) $\text{Na}_2\text{C}_2\text{H}_3\text{O}_2 \cdot 3\text{H}_2\text{O}$ (6) $\text{Al}_2(\text{SO}_4)_3 \cdot 18\text{H}_2\text{O}$ (7) H_2O_2 (30%)(8) $\text{CuSO}_4 \cdot 5\text{H}_2\text{O}$ (9) Na_2CO_3 (anhydrous)

surface is necessary to obtain good adherence but also because surface preparation has much to do with the final appearance, since the intermediate or final coating is frequently buffed to a high lustre. Therefore, the base metal should be smooth if polishing and buffing costs are to be kept to a minimum. The decorative coatings most commonly applied are nickel, cadmium, nickel followed by chromium, or copper followed by nickel or by nickel and chromium. To protect hard steel and iron surfaces from wear or abrasion, coatings of chromium or alloy coatings of tungsten and cobalt or nickel and tungsten are sometimes used although some of these may not always be applied by electroplating. A few typical plating baths with operating data as commonly used by job platers are shown in Table 36—III.

Miscellaneous Metallic Coatings—(1) Cathode Sputtering. When a relatively high voltage is applied between

two electrodes in a partial vacuum, inducing a glow discharge, the cathode disintegrates and the metal thus removed can be deposited in a thin film on near-by objects within the chamber. In suitably designed chambers, objects may be arranged with respect to the cathode so that they will receive a uniform, thin coating of metal. The process is particularly suitable for the metallization of electrically non-conducting materials such as fabrics and phonographic recording waxes.

(2) *Evaporation or Condensation.* This process, closely related to cathode sputtering, is of more recent origin in practical application. The metal vapor is produced by thermal instead of electrical means. A coiled filament of platinum or tungsten in a higher vacuum than for cathode sputtering is a convenient heat source. The process is usually confined to the deposition of pure metals.

SECTION 4

SURFACE CONVERSION COATING

Steel is often treated in various ways by heating to form a uniform blue or black coating of oxide which, although not thoroughly protective, is pleasing in appearance and, especially if coated with oil, wax or other clear protective coating, is much more resistant to corrosion than steel not so treated. "**Blue-annealed**" plate now produced on continuous plate mills is one example of this finish. **Steam-blued** and **air-blued** finish on thin sheets used for common stove-pipe stock are others and the **gun metal** finish applied to gun barrels is another. Strip steel is satisfactorily colored by passing through heated sand. From three to ten minutes treatment at 650° F will produce a rich blue color. Highly polished steel may be blued by placing it in a bed of hot charcoal about two feet deep. The lower part of the charcoal is in a state of incandescence whereas the upper layers are lower in temperature and suitable for the development of oxide colors. After development of the desired shade of blue, the article is rubbed vigorously with waste or cloth dipped in raw sperm oil.

The **gun-metal** or **carbonia** finish used on rifles, shot-guns and revolvers, as well as on many other metal parts, is obtained by placing the steel loosely in a retort with a small amount of charred bone and heating to 700 to 800° F. After the articles are thoroughly oxidized, the temperature is dropped to about 650° F and a mixture

of bone and carbonia oil are added, after which heating is continued for several hours. On removal, the articles are dipped in sperm oil or tumbled in oily cork to develop a uniform, black finish. If a lower temperature is necessary to prevent excessive softening of the steel, a longer time is usually required and the color obtained is less permanent than at the higher temperatures. Articles which have been first nitrided, when treated by the **gun-metal** or **carbonia** process, take a pleasing, rust-resistant finish and retain their surface hardness since coloring temperatures do not temper nitrided articles.

Barfing is a process somewhat analogous to steam bluing of sheets and black plate. The steel articles are cleaned, placed in air-tight ovens and they then are heated to a dull red heat. Super-heated steam at 60 to 100 lb. per sq. in. pressure is introduced and a slate-blue coating is obtained consisting largely of magnetic oxide of iron. The coating is of considerable depth and is quite durable especially when oil or wax coatings are used as the final application. The **Bower-Barff process** is similar except that, after the steam treatment, the steel articles are cooled to 300° F, dipped in hot linseed oil and kept at 300° F until the oil becomes oxidized. Later, the process was modified by introducing benzene with the steam, thus shortening the treatment and producing a heavier coating.

SECTION 5

CHEMICAL TREATMENT OF STEEL SURFACES

Black, blue or brown finishes on steel also may be produced in various shades by a wide variety of chemical treatments. Molten-salt baths produce effective colors on clean, polished steel, but often the temperature is so high that a change in hardness and other mechanical properties may result. The method may be high in cost because of drag-out salt adhering to the metal on removal from the molten-salt bath.

Niter Baths—Molten mixtures of sodium and potassium nitrates are effective bluing agents in the absence of rust. Manganese dioxide is generally added to the extent of about 2 per cent by volume and seems to aid in the production of good colors. Potassium nitrate is used when a bath operating at low temperature is employed, although the sodium nitrate alone may be used without

affecting the quality of the work. The temperatures used vary from 600° to 1000° F. If a lower temperature is necessary, a black color may be obtained by using a 40 per cent aqueous solution of sodium hydroxide to which about 5 per cent each of sodium and potassium nitrates are added. This solution operates at 250 to 285° F. The colored articles are usually finished by immersing in hot oil followed by wiping and polishing.

Polished steel is **oil blackened** by packing in a carburizing box with spent carburizing compound, excluding air, and heating to 1200° F for about one and one-half hours, and then quenching in oil. Variations of this process at lower temperatures may be effected by heating the articles rapidly to 1000° to 1200° F in air, and quenching in oil. Small parts are colored by introducing

them into a rotary furnace retort operating at about 750° F. A small amount of linseed or fish oil is added to the charge and the parts rotated for three to ten minutes after this addition, after which parts are cooled in air and dipped in a rust-retarding oil.

Articles which have been quenched in oil may be placed, without removal of oil, in a rotary, unperforated-drum retort and heated to 500° to 650° F, maintained for proper tempering time according to color desired. The longer the time the deeper and more desirable will the black color be. The retort is allowed to cool to 500° F, then articles are removed and tumbled in slightly-oiled granular cork to brighten. This method gives a combination tempering and oil-blackening treatment.

Browning of steel is accomplished by a wide variety of processes. The thoroughly cleaned steel is coated by spraying, brushing or dipping with two coats of a browning solution which generally consists of a mixture of metallic salts, acids, alcohol or water. The coating is allowed to dry, heated to 140° to 175° F, and then placed in a humidity room or chamber at the same temperature as that used for preheating, where it is allowed to rust. It is then washed in boiling water for 15 minutes, dried and cleaned with a wire brush or fiber wheel to remove loose particles of rust. Three more rustings with intermediate cleanings are applied after which the browned surfaces are coated with a rust-preventive oil.

Solutions for the coloring of steel by chemicals are very numerous and often contain lead, iron, mercury, antimony or copper salts in combination with sulphur or selenium compounds. Usually the colors obtained by the relatively-cold chemical methods are not as brilliant as those from heat tinting in air or steam, or the product

from salt baths. Nevertheless, certain new chemical treatments have been widely adopted, not because of their superior appearance, but because they are beneficial in bonding paints and lacquers, especially when these are baked. Such coatings have a very durable, final finish and one in which corrosion due to porosity or imperfections in the paint film is minimized. Among these are Coslettizing in which the cleaned and polished steel articles are boiled for about two hours in a solution of phosphoric acid and ferrous sulphate or zinc sulphate, thus forming a basic iron or zinc phosphate which is very adherent. Parkerizing consists essentially of treating the clean surface with manganese dihydrogen phosphate. Later the Parkerizing process was shortened by incorporating a copper salt in the bath to which the name "Bonderizing" was given. Still further improvements have recently been made in the direction of time reduction and spray application. Some of these improved solutions contain zinc phosphate along with alkali nitrates or nitrites.

The Laxal treatment substituted oxalic acid for phosphoric acid and was satisfactory when the enamel- or lacquer-baking temperature could be maintained under 300° F.

Chromodizing makes use of a solution of chromic acid and sodium chloride which is sprayed upon clean sheets at a temperature of about 180° F.

There are a number of other proprietary or commercial coatings which depend upon surface conversion. They usually employ treatment with phosphoric-acid, phosphate, chromic-acid, chromate or strong alkali solutions. In some instances the application of an electrolytic treatment is employed.

SECTION 6

CHEMICAL TREATMENT OF METALLIC COATINGS

Metallic coatings such as zinc, tin and aluminum may be subjected to chemical treatments to increase their durability or facilitate the application and adherence of enamels, paints and the like.

Parkerizing or **Bonderite** solutions, somewhat different from those used on steel but essentially phosphate in composition, are used on galvanized or zinc coatings to increase the adherence of the lacquer or enamel. Patents are owned by the Parker Rustproof Company, Detroit, Michigan.

The "Cronak" treatment likewise is used on zinc and consists essentially of sodium-dichromate solutions acidified with dilute sulphuric acid. The process was developed by The New Jersey Zinc Company, New York City.

The **Iridite** treatment consists of immersion in chromium-salt solutions compounded to yield coatings of various shades. These give a reasonable amount of resistance to wear and corrosion. They may be applied in 10 to 20 seconds to freshly-plated zinc or cadmium coatings and are followed by a cold and a hot water rinse (see "Iron Age," May 24, 1945; pages 34-40).

Anozinc is an electrolytic treatment for zinc coatings and zinc die castings presumably consisting of a chromium-salt mixture which, with suitable variation, results in several shades or colors. The zinc-plated article is made the anode and treated at a current density of 5 to 50 amperes per square foot for 1 to 5 minutes. This process was developed by United Chromium, Incorporated, Detroit, Mich.

Banox consists of the application of an amorphous or

vitreous phosphate generally applied to zinc coatings, although it is modified to apply also to uncoated steel. It affords temporary protection and serves as a bonding agent for paints, enamels and the like (see "Steel," June 17, 1946; pages 122-129).

Other proprietary or commercial coatings have been used more or less successfully as a treatment for metallic coatings. Also numerous formulae, given in handbooks, are adaptable to coloring metallic coatings either for protective or decorative purposes. Typical are two given for blackening electroplated articles in either of the following solutions heated to 120° F:

Antimony black

Water	1 gallon
Sodium hydroxide	4 ounces
Antimony oxide	¼ ounce

Nickel black

Water	1 gallon
Nickel chloride	4 ounces
Ammonium chloride	6 ounces
Sodium sulphocyanide	2 ounces
Zinc chloride	½ ounce
Copper carbonate	¼ ounce

After the black coatings are developed, the articles are rinsed in cold water, then in boiling water, dried at 200° F and oiled or lacquered.

SECTION 7

VITREOUS-ENAMEL COATINGS

These coatings consist of a layer of glass fused to the properly prepared steel base and thus are quite different from enamels of organic origin which will be discussed later. To adjust the properties of the finished articles to the ultimate uses, wide variations in the composition of this glass are required but, in general, it must adhere well to the steel base and possess a coefficient of expansion adjusted to that of the base metal. Good adherence is achieved by incorporating in the enamel certain oxides, usually cobalt oxide. Adjustment of coefficient of expansion is accomplished by a variety of compositions which are compounded by fusing together quartz and feldspar, with fluxes such as borax, fluorspar, cryolite, soda ash, sodium nitrate and litharge. Opacifiers such as oxides of tin, zirconium, or antimony are usually added when the glass is ground to a fine powder. This is generally accomplished in a pebble or ball mill.

Ground coat application is made to the thoroughly cleaned sheet steel (or cast iron) article by immersing in a water suspension or "slip" of pulverized enamel ingredients. The prior cleaning may consist of degreasing, pickling in acid, rinsing in a neutralizing bath, sometimes followed by a nickel-solution dip to improve enamel adherence and behavior during firing, after which the articles are washed and dried. Cast-iron articles are frequently cleaned by sand or shot blasting. After application, the slip is allowed to drain, dried, heated (fired or burned) at as high a temperature as 1500° to 1600° F for 1 to 4 minutes and cooled to room temperature. This fired ground coat offers sufficient pro-

tection to the steel base and may be used alone. However, since the "slip" usually contains cobalt oxide, the resulting coating is dark blue and may not be suitable for all purposes, so that a finish coat may be applied.

Finish Coats are applied when a light color or additional protection such as acid resistance is desired. The fired ground-coated article is sprayed with a slip of the required finish-coat composition, dried, fired for 1 to 3 minutes and cooled. The operation may be repeated several times using the same or different slips and many attractive color combinations can be obtained if desired.

Single-Finish Coats—Much experimental work has been directed toward obtaining a white enamel (since white is used most widely as the final coat) which can be applied directly to the steel or cast-iron base and result in a good appearance with one coat only. This can and has been done successfully but, frequently, the one coat may be inferior in lustre when compared with a piece having two coats. Also, imperfections are very noticeable in white and the economy of using only one coat may be offset somewhat by higher rejections.

Low-temperature vitreous enamels—To provide a coating where the high gloss or decorative enamels are not required, ceramic coatings have been developed which can be applied by spraying and firing at a temperature of as low as 700° F for about 15 minutes. These have found a limited field of use for containers, marine hardware, ammunition boxes, radio parts, cartridge cases and machine-gun mounts. The composition is said to consist of an alkaline aluminum silicate.

SECTION 8

MISCELLANEOUS INORGANIC COATINGS

Cements—These coatings differ from vitreous enamel in that they are not always fused to the steel, although the constituents consist largely of finely ground, vitrified products. Cracking vessels in the oil industry are sometimes protected by a mixture of furnace cement and sand to which short fiber asbestos and water glass are added. In this instance, curing is facilitated by heating to about 900° F. The interior of cast-iron or steel pipe and steel tanks may be coated with cement to resist corrosive waters, salt solutions, oil having a high sulphur content and the like. Concrete coatings are used on the exterior of pipe when it is buried in extremely moist or corrosive soils. This concrete is a rich mixture approximating two parts sand and one part Portland cement and may be two to four inches in thickness. The alkalinity of cements

usually inhibits corrosive attack of the steel to which they are applied.

Core Plate—These coatings, which are discussed in greater detail under silicon or electrical-steel sheets (Chapter 46), may sometimes be inorganic in nature and are usually applied to silicon-steel sheets used for transformer laminations to improve the insulating properties. Other core plates used for a similar purpose or for insulating motor laminations are organic-varnish coatings.

Metal Powders in Inorganic Vehicles—Metal powders may be incorporated with inorganic silicates or phosphates to produce a protective coating. Usually, such coatings require some baking in order to obtain satisfactory adherence and hardness. Bronze paint and aluminum paint are examples.

SECTION 9

ORGANIC COATINGS

Temporary Organic Coatings—For temporary protection in shipment or storage, steel is usually coated with oils known as *slushing oils*. Mineral oils alone are sometimes used where corrosive conditions are not severe. More often, the mineral oils are mixed with inhibiting or polar compounds which deter rust formation. Slushing oils are generally removed subsequent to fabrication and prior to application of a permanent organic coating. At

times, when diluted with solvents which vaporize and leave only a very thin coating, the permanent organic coating can be applied satisfactorily without removal of the temporary protective film. Often, however, unless the temporary protective film is removed, troubles are encountered due to its incompatibility with the permanent organic coating. For severe conditions such as outdoor storage or overseas shipment, heavy greases or waxes

Table 36—IV. Some Substances Commonly Used in Compounding Organic Coating Mixtures

Organic Coatings Composed of	Pigments	Carbon—Lampblack	
		Metals	{ Zinc Aluminum Brass Bronze
			{ Zinc Oxide Red Lead
			{ Ferric Oxide Antimony Trioxide Titanium Dioxide
		Hydrates	{ Iron (Ochres) Aluminum Lead Hydroxide
			{ Lead Carbonate (White Lead) Zinc Chromate Lead Chromate Ferro Cyanides
		Salts	{ Linseed Soybean
			{ Chinawood or Tung Oil Perilla Castor
		Binders	{ Oils
			{ Copal Dammar Kauri Rosin Ester Gum Mastic Sandarac Shellac Bakelite Vinyl Vinylidene Vinylid Pliolite Styrene Alkyd Ester Urea Formaldehyde Rubber Tormesit
			{ Synthetic
			{ Natural
			{ Asphaltum Gilsonite Tar and Oil Pitches
			{ Bitumens
			{ Nitrate—Pyroxylin Acetate Ethers Di-Butyl Phthalates Di-Ethyl Phthalates Di-Amyl Phthalates Tri-Cresyl Phosphates Tri-Butyl Phosphate Butyl Stearate
			{ Cellulose Derivatives
			{ Nonvolatile Solvents
			{ Turpentine Dipentene Pentene
			{ Hydrocarbons
			{ Petroleum { Hydrogenated Paraffin Petroleum Extracts Toluol Xylol Hi-Flash-Naphtha
			{ Aromatic
		Thinners	{ Acetates { Amyl Butyl Ethyl Ethyl
			{ Butrates { Ethyl Butyl
			{ Ethers { Cellosolve Carbitol
			{ Lactates { Ethyl Butyl
			{ Alcohols { Ethyl Butyl Amyl Hexalin (Phenol Derivatives) Methyl
			{ Ketones { Ethyl Butyl
			{ Oxides { Lead Iron
			{ Salts { Lead Manganese { Resinates Linoleates Cobalt { Naphthenates
			{ Antidrying Agents { Hydroquinone Phenolic Bodies

compounded with inhibitors are applied. When conditions are unusually severe and the value of the steel product justifies the cost, plastic films are applied, which can be removed by stripping. This latter expedient may be used for overseas transportation and storage of expensive equipment.

Permanent Organic Coatings are commonly known as paints, varnishes, enamels, lacquers and japans. The pigments for organic coatings are generally of inorganic nature. **Paints** are mixtures of pigments with drying oils, usually linseed or tung oils, which dry by oxidation. **Varnishes** are solutions of resinous material in oils or volatile liquids which dry by evaporation or oxidation. **Enamels** can be called varnishes to which some pigment has been added. **Lacquers** comprise solutions of shellac, resins, cellulose derivatives or polymerization products in suitable solvents, all of which dry by evaporation. **Japans** are varnishes compounded to permit drying at a temperature high enough to keep their oleoresinous compounds in a liquid state until oxidation is completed. As indicated in Table 36—IV, the subject of organic coatings is too extensive and complex for discussion in this book.

The importance of such coatings in the protection of steel is, however, too great to dismiss without a brief discussion of some of the types of excellent coatings which have been developed. This field is in a constant state of flux or change due to rapid application of newly developed materials.

As mentioned previously, paints may be considered as mixtures of pigments with drying oils. The service life of the painted steel will depend on the quality of the paint applied, the type of environment to which the painted article is exposed, the surface preparation and the care exercised in application of the paint.

Special Polymers—Rapid development in the synthetic-resin field has produced many synthetic finishes of extraordinary resistance to various corrosive conditions. Many of these are limited to special fields because of high cost but their use has increased as means have been found to produce them more economically. Some, such as the polyvinyl-acetate resins and copolymers of vinyl acetate and vinyl chloride, are finding wide application because of their resistance to chemical attack, superior adhesion and unusual toughness. Polymers of certain derivatives of acrylic and methacrylic acids possess optical clarity and superior light transmission as well as resistance to chemical attack. Certain pigments incorporated in paint films inhibit both the start and spread of corrosion. Zinc and lead chromates, and red and blue leads are useful in this respect. Rubber modified with chemicals such as chlorine is a very suitable substance for paints intended to resist corrosive chemicals.

Asphaltum finishes based on coal tar or petroleum asphalt are black and possess good heat resistance where temperatures are not too high. They are commonly incorporated with other materials to increase their durability.

Application and Drying of Organic Finishes—Since the bond between metal and coating is the weak point in most paint systems for metals, the method of application is extremely important. Prior to application, thorough cleaning of the surface is necessary. Since no paint system is entirely impermeable to moisture, durability is increased by chemical treatments or inhibitive washes of the steel base, many of which are proprietary in nature such as "Bonderizing." The application of such treatments prior to painting is practiced extensively in

plants fabricating steel articles, although recently their use has increased on structural-steel work after erection. Tests at the National Bureau of Standards indicate that the rust-inhibitive solution should be allowed to react with the steel, which then dries. Thorough washing is then advisable, to remove the excess inhibitor, followed again by drying prior to application of the primer paint. A satisfactory wash originated by the U. S. Engineer's Office has the following composition:

Ingredients	Per Cent by Weight
Commercially-pure ortho phosphoric acid (H_2PO_4)	7.00
Sodium dichromate ($Na_2Cr_2O_7 \cdot 2H_2O$)	0.75
Caramel solution	0.50
Wetting agent (such as "Cellosolve," ethylene glycol monoethyl ether, $C_2H_5OCH_2CH_2OH$)	0.50
Water	91.25

The older and more common methods of application of finishes by brushing or dipping are still widely used but in factories the installation of conveyor systems has increased the use of roller coating methods for flat products and spraying for more complicated or formed shapes. These operations are usually performed mechanically and are often followed by a closely controlled baking treatment. Banks of infra red lamps, or gas fired or electric ovens, may be used. In some instances, this heating merely serves to drive off excess solvents. In other cases the oxidation of the coating, required for a durable finish, is also accomplished. Here, the composition of oven atmosphere is important. Baked coatings are usually harder, tougher and often more durable than those which are air dried, and quite often the coating mixture for an air-dry coating is different than one for baking. With the more general use of continuous coating methods, continuous baking is a common practice in quantity production. These methods are utilized in the production of small and large parts of the majority of manufactured articles.

Chapter 37

THE MANUFACTURE OF TIN PLATE

SECTION 1

TIN-MILL PRODUCT TERMINOLOGY

Tin-mill products originate with flat-rolled, mild (low-carbon) steel in relatively thin gages. They comprise **black plate**, **tin plate**, and **short terne plate**. Of these, tin plate is the most important commercially, accounting for about four-fifths of the combined tonnage of all tin-mill products. Black plate accounts for most of the balance. Short terne plate, at one time produced in quantity, is now a relatively minor factor in tin-mill production, and is discussed in Chapter 38.

Black Plate—Although originally designating thin steel plates produced by hand hammering, the term "black plate" has persisted and now defines the product of the cold-reduction method in gages No. 29 and lighter (thicknesses 0.0141 inch and under). Some of the details of rolling and heat treating such light-gage product were discussed in Chapter 34.

"Black" plate does not have a black appearance. Present methods of manufacture generally result in a flat-rolled product having the typical appearance of clean steel. The appearance of black plate may be affected, however, by modifications of annealing practice, or of processing methods that affect the roughness of the surface texture.

Black plate as such is sold either in cut sheets or in coil form and is used for fabricating a variety of items including containers, trays and toys. When coated with suitable organic coatings, it exhibits considerable resistance to corrosion and its use for containers is increasing. Uncoated, black plate is extremely susceptible to rusting and precautions to prevent condensation of moisture must be taken during shipment and warehousing. Some manufacturers are equipped to apply thin films of protective oils to minimize rusting. Other mills are equipped to produce chemically treated steel (CTS), which is black plate given a protective chemical treatment to enhance rust resistance and adhesion of organic coatings. Such treated plate usually requires a protective organic coating applied by the user. Precautions to prevent condensation of moisture are necessary during shipment and warehousing.

The most important use of black plate is in the manufacture of tin plate, as described hereafter.

Tin Plate—Tin plate may be described as full-finish black plate additionally processed and coated on both sides with commercially pure tin. The widespread use of this major steel-mill product arises from its combination of the strength of steel with the protective properties and solderability of tin. When coated by the hot-dip process (see Section 5), the tin plate is termed **coke tin plate**, **charcoal tin plate**, or **silver tin**; these terms are defined under "Hot-Dipped Tin Plate," immediately following this paragraph. When coated by the electrolytic process (see Section 6), it is termed **electrolytic tin plate**. In the United States at the time this is written, approximately

80 per cent of the tin-plate tonnage produced represents electrolytic tin plate and the remaining twenty per cent tin plate made by the hot-dip process.

Hot-Dipped Tin Plate—Tin plate formerly was produced by the hot-dip tinning of thin plates hammered or rolled from bars of puddled iron. The pig iron used in puddling might have been made in blast furnaces using charcoal or coke as fuel. Plates made from "charcoal iron" were considered a more ductile and higher grade product; hence, tin plate made by coating charcoal iron plates with tin (charcoal tin plate) was regarded as a product of higher quality than tin plate with a coke iron base (coke tin plate). At present, the designation "charcoal tin plate" merely indicates plate with a relatively heavy tin coating as compared with "coke tin plate" as described below, and has no significance so far as quality of the steel base is concerned.

Various grades of hot-dipped tin plate are produced at present, the terminology indicating in a general way the weight of tin coating or, indeed, more exactly indicating the amount of tin used to produce the given unit or base box of plate. Coke tin plate has always designated the plate produced with the lowest amount of tin and formerly was called "Cokes" or "Common Cokes." Currently, in ascending weight of coating, the various grades are Common Cokes, Standard Cokes, Best Cokes, Kanners Special Cokes. "Charcoal Tin Plate" carries still heavier coatings including 1A and 2A. Silver tin is made by hot-dip tinning a mechanically roughened steel base. It has a dull surface, but carries a heavy coating of tin.

Electrolytic Tin Plate is available in both melted and matte (unmelted) finish. The former has bright luster and is similar in appearance to hot-dipped tin plate. Matte finish tin plate is lacking in luster.

Electrolytic tin plate theoretically can be produced in any coating weight. The commercial grades of electrolytic tin plate available at present include coating weights of 0.25, 0.50, 0.75, and 1.00 lb. of tin per base box, identified by the numerals 25, 50, 75, and 100, respectively; in addition, tin plate identified as 100-25 differential tin plate is available that carries a 1.00 lb. per base box coating on one side and a 0.25 lb. per base box coating on the other. Although normally sold in sheet form, electrolytic tin plate is available in coil form also.

The 100-25 differential plate was developed to conserve tin, the 100 surface being employed as the inside surface of containers where maximum protection is required. The light-coated surface of 100-25 differential is identified by making it less lustrous than the heavy-coated surface. This is usually done either by roughening the steel surface during temper rolling or by anodizing the plated surface in a suitable electrolyte just before melting. Other methods for producing a non-lustrous surface are under consideration.

Symbols and Definition of Base Box—Tin plate is sold on a weight per unit area basis rather than a gage thickness basis. The unit of area is the base box, equal to the area of 112 sheets, 14 by 20 inches, or 31,360 square inches (217.78 sq. ft.).

In the early eighteenth century in England, 14 by 20-inch plates were packaged in lots of 112 sheets to make one hundred-weight (112 lb.). When the plate was heavier, it was identified as IX or one cross. The cross denotes an increase in weight of 28 lb. or one quarter hundred-weight.

The use of symbols to designate gage now has been displaced by base weight classification. Base weights (or basis weights) which indicate the approximate thickness of the plates are expressed in pounds per base box. Estimated weights of tin plate of other sizes than 14 by 20 inches are computed readily through the use of base box ratio tables. Table 37—I shows the nominal weights in pounds per base box of the plate commonly produced. The table applies to hot-dipped tin plate produced in sheet form. Present continuous strip plating lines for electrolytic tin plate are normally designed for producing the lighter basis weights, up to about 135 pounds per base box.

Table 37—I. Nominal Weights of Tin Plate Commonly Produced

Weight (lb. per base box)	Equivalent Weight (lb. per sq. ft.)	Weight (lb. per base box)	Equivalent Weight (lb. per sq. ft.)
55	0.2526	155	0.7117
60	0.2755	168	0.7714
65	0.2985	175	0.8036
70	0.3214	180	0.8265
75	0.3444	188	0.8633
80	0.3673	196	0.8954
85	0.3903	208	0.9551
90	0.4133	210	0.9643
95	0.4362	215	0.9872
100	0.4592	228	1.0469
107	0.4913	235	1.0791
112	0.5143	240	1.1020
118	0.5418	248	1.1388
128	0.5878	255	1.1709
135	0.6197	268	1.2306
139	0.6383	270	1.2398
148	0.6796	275	1.2628

SECTION 2

OCCURRENCE, MINING, AND REFINING OF TIN

Tin, though one of the common metals, is the most sparsely distributed metal in common use. Its main sources at present are the Malay States, Dutch East Indies, Bolivia, Siam, China, Belgian Congo and Nigeria. These deposits produce probably 85 per cent of the world's supply. The large portion of the remaining 15 per cent is accounted for by the deposits in Australia, the centuries-old mines in Cornwall, the Union of South Africa and India.

The most abundant source of tin is the oxide, cassiterite, or tin stone, and the greater portion of the world's supply from workable deposits is derived from alluvial deposits in river beds. The cassiterite originally occurred in veins and lodes in highly acid igneous rocks. The ore may be found also as primary veins in metamorphosed sedimentary rocks. Cassiterite has a relatively high specific gravity (6.4-7.1).

Alluvial deposits of tin stone occur associated with gravel, which may be bonded with clay and covered with more or less overburden consisting of soil, clay, etc. This type of deposit, found in the Malay States, Dutch East Indies, Siam, China and Nigeria, is worked by various types of hydraulic mining. Washing of the deposit with running water causes a breaking up of the gravel and a carrying away of the lighter pebbles while the heavier tin stone is retained and recovered. The concentrates are hand picked to remove non-stanniferous metals, and may be rewashed in small sluice boxes and concentrated by hand jiggling.

Tin ore in Bolivia and Cornwall is practically all vein tin. Here the tin ore is found in lodes and beds, in older rocks such as granite, gneiss and mica schist, associated with a large proportion, perhaps 95-99 per cent, of gangue consisting principally of silica and silicious minerals, metallic minerals like the sulphides of iron, copper, lead and zinc, iron oxides and wolframite (a tungstate of iron and manganese). The ore first is crushed to pass approximately a twenty-mesh screen and then concentrated by washing. The finely washed ore in the slimes is treated in settling tanks, or on slime tables.

Regardless of which method is used to concentrate

the ore, the concentrates then are roasted to remove sulphur and arsenic. This operation in turn is followed by a second washing, although in some cases the roasted ore is treated in magnetic separators to remove iron oxide and wolframite prior to washing. The tin stone is then ready for the smelter. Reduction of the tin ore is accomplished either by the use of a reverberatory furnace or a blast furnace. The crude tin thus obtained then is refined by a liquating operation in which advantage is taken of the low melting point of tin. The impure metal is heated on the inclined bed of a furnace to a temperature just above its melting point. Comparatively pure tin trickles down to a basin below, leaving the higher melting point impurities on the bed of the furnace. The low melting point impurities, lead and bismuth, are removed by either or both of two oxidizing methods. The first of these is an operation called **boiling**, in which sticks of green wood are immersed in the molten metal and undergo destructive distillation to produce bubbling by the steam and gases generated to agitate the molten metal bath. Different portions of the metal thus are exposed to air and are oxidized. The oxidized impurities float on top of the molten metal to form **dross**. The oxide drosses then are skimmed from the surface of the bath. Similarly, pouring ladlefuls of molten metal from a height into the bath permits the oxidation of impurities. This latter operation is called **tossing**. Drosses may be resmelted to recover tin losses.

In more modern extraction methods, ore concentrates are calcined with or without additions intended to facilitate further processing. The iron and other impurities then are removed readily by leaching the ores in solutions of hot hydrochloric acid. Leaching is followed by filtering of the chloride solution. After washing with dilute acid, the residues are of such purity that subsequent treatment in reverberatory furnaces will yield a metal containing at least 99.80 per cent tin. It is possible to obtain metal of 99.98 per cent tin content through refinement by electrolytic means, but electrolytic methods on a commercial scale were abandoned in the United States in 1923.

Properties and Uses of Tin—Tin has a silver color with a slight bluish tinge, a brilliant luster, a structure which is distinctly crystalline, and is soft and malleable at ordinary temperatures. Other physical properties of tin are:

1. Atomic Weight—118.7
2. Atomic Number—50 (isotopes with masses 112 to 124)
3. Density—7.31
4. Specific volume—0.1395 at 20° F
5. Hardness on Mohs' scale—1.8
6. Tensile strength—about 2200 lb. per sq. in. and elongation of 86 per cent

7. Melting point—449.4° F (231.9° C)

8. Boiling point—4120° F (2270° C)

Tin is alloyed with other metals to make bronze, Britannia metal, pewter, solder, or white bearing metal. Pure tin or lead-tin alloy, rolled very thin, is known as tin foil. Tin amalgam is used in making mirrors, and tin condenser tubes are used in laboratory stills. Tin, in conjunction with an acid, is used as a reducing agent.

Stannic oxide is used as a polishing powder, and as an opacifying agent in glasses, glazes and enamels. The chlorides are used as a mordant in weighting silk and in dyeing. The largest use of tin at the present time is the manufacture of tin plate.

SECTION 3

USES AND IMPORTANCE OF TIN PLATE

Tin plate represents one of the major items produced by the steel industry in the United States. In normal non-war years, the tonnage has represented approximately 5 to 7 per cent of the total steel production. Statistics compiled by the American Metal Market show that production has increased steadily from about 340,000 tons in 1900 to about 2,750,000 tons in 1940; new facilities have increased capacity so that by 1948 it was about 4,000,000 tons and by 1955 it was about 5,600,000 tons. This increase has kept pace with the growth of the food preservation industry.

The continued increase in consumption of tin plate attests to the unique properties of this product, in which are combined the strength of steel and the corrosion resistance of tin. Tin plate is fabricated and soldered

readily on high-speed forming equipment. It has a pleasing appearance. It is relatively inexpensive and is non-toxic. Because of these and other properties, tin plate has been found to be the ideal fabrication material for food and other containers (tin cans), crown caps and other bottle caps or closures, for kitchen utensils such as baking pans, for various drawn or fabricated parts in radios, such articles as electrical equipment, and toys.

The largest use of tin plate is for containers, and the improvements in its manufacture have been dictated largely by the requirements of the container industry. The cans are used not only for foods, but also for paints, oils, tobacco, detergents, insecticides, proprietary drugs and beverages.

SECTION 4

PROCESSING OF STEEL FOR TIN PLATE

Types of Steel Used—Most of the steel used for the production of tin plate is made by the open-hearth process. Depending on the stiffness required for the various applications, the phosphorus content may vary from residual to as high as 0.15 per cent. Acid Bessemer steel, because of its relatively higher phosphorus content, is stiffer than average basic open-hearth steel, and has a distinct advantage where greater stiffness is desirable.

The steels utilized for tin plate production commonly are classified as "dead soft" carbon steels having a maximum carbon content of 0.15 per cent. The great preponderance of tin plate steels are produced as either rimmed or capped steels. The rimmed steels are utilized in general for the softer and deep-drawing tin plate requirements, whereas the capped steels find application in meeting the stiffer tin plate requirements of stamping or forming. The capped steels for tin plate have the definite advantage of being relatively uniform throughout in grain size, cleanliness, and chemical composition.

Of special importance in the manufacture of hot-dipped tin plate is the selection of the proper composition of steel base for the job. Virtually all tin plate is made with a base having composition (of the metal in the ladle) falling within the following composition ranges:

Carbon	0.04 to 0.15
Manganese	0.20 to 0.60
Phosphorus	0.15 max.
Sulphur	0.050 max.
Silicon	Residual
Copper	0.20 max.

For particular applications, as determined by temper requirements and pack corrosivity, the desired ranges will be selected. The acid fruit products represent in general the most corrosive media. For these applications the phosphorus, silicon, copper and "tramp" elements in the steel are held to low limits.

Equipment and Practice—The sequence of operations in the manufacture of tin plate is as follows: slabs are heated and hot rolled to coiled breakdown form on the hot-strip mill. The breakdowns are continuously pickled and taken to the cold-reduction mills where they are reduced to black-plate gages. The cold-reduced material is cleaned, annealed and temper rolled in coil form; these coils are the starting material for the hot-dip tinning process described in Section 5 of this chapter, beginning with shearing of the plate into sheets.

As preceding sections of this book discuss and describe in detail the equipment and operation of the continuous hot-strip mill, the continuous pickler, the cold-reduction mills, and all other equipment necessary for the processing of hot-dipped tin plate up to the actual coating operation, the following explanations will be limited to general control measures designed to produce the most suitable product.

In continuous hot-rolling, the slabs of steel, ranging from 4 to 7 inches in original thickness, are hot reduced at an elevated temperature to a single continuous length which, for ease in handling, is rolled up into the form of a coil. This coil product may be as thin as 0.065 inch or as thick as 0.125 inch, depending upon the desired thickness of the product after cold reduction; it will

generally be between 0.070 and 0.090 inch for the bulk of the tin plate produced in this country. Factors such as type of hot-rolled surface desired, ease with which oxide can be removed prior to the cold-reduction operation, hardness of the hot-rolled product desired and resultant mechanical properties, grain size and corrosion resistance of the final product, all must be considered in establishing an optimum hot-rolling practice. In general, the 0.070 to 0.090 inch thick hot-rolled product will have a temperature between 1500° and 1600° F at the exit side of the last finishing stand. Similarly, this product going into the coiler generally will be in the range of temperature between 1100° and 1250° F.

For the production of tin plate, it is sufficient to say that the primary functions of continuous pickling are to remove uniformly all of the hot strip mill scale and oxide from the surface of the steel and subsequently to oil the pickled product. These operations are necessary prior to taking the very heavy cold reductions necessary to obtain the light tin plate gages and to ensure the clean surface finally necessary for the tinning operation.

The importance of the cold-reduction operation to the over-all quality of tin plate cannot be over-emphasized. This operation determines the gage uniformity and, to a great extent, the surface quality and flatness of the final tin plate product. These three properties require particularly close control because of the demands of the very sensitive coating operations and the high-speed automatic can-making operations. To attain the 80 to 90 per cent cold reduction used on the base metal for tin plate manufacture, nearly all plants in this country use four-stand or five-stand four-high tandem cold-reduction mills with delivery speeds of 2000 to 6000 feet per minute. The design and operation of these mills are described in Chapter 34. It should be mentioned here that the mills used are generally from 42 to 56 inches wide to produce the usual 24 to 36-inch widths and that, where a mill is installed solely for the manufacture of tin plate, the five-stand mill is more satisfactory than the four-stand for rolling the light gages involved.

The cleaning operation is particularly important in the manufacture of coated products, as the efficiency with which the cleaning operation is performed has a marked bearing on the subsequent coating operation and the quality of the final coating. Its function is to remove all traces of cold-reduction lubricant from the coil product, leaving a bright, clean surface for subsequent operations.

This cold-reduced product is quite hard, having a Rockwell (30-T scale) hardness of approximately 80 to 85, and must be softened by annealing. Both batch-type (box anneal) and continuous annealing furnaces are used and the trend is toward further use of continuous-annealing furnaces. In the **box-annealing process**, steel for tin plate is heat treated in coil form in a protective atmosphere. Considering both equipment limitations and quality evaluation of the annealed product, it has been found generally desirable to box anneal the coiled product within the steel-temperature range of 1150 to 1250° F, with soaking periods of 4 to 12 hours. This type of subcritical annealing is known as **process annealing**. While the heating cycle in relation to the metalloid content

of the steel largely determines grain size of the finished product and exerts a strong influence on the mechanical properties of the finished tin plate, the cooling cycle plays an important part as regards surface. A clear, bright, relatively oxide-free coil product can be obtained if the deoxidizing gas is allowed to circulate under the inner cover until the steel temperature drops below approximately 250° F; such a surface is considered desirable for tin plate. In the **continuous-annealing process**, the steel is heated in protective atmospheres to about 1200° F in a fraction of a minute at strip speeds of 1000 feet per minute in existing furnaces and at projected speeds of 2000 feet per minute in newer furnaces. Continuously annealed strip is inherently stiffer than box-annealed strip, but possesses good formability; thus, continuous annealing offers a method for obtaining tin plate of higher temper without change in steel composition.

The function of the temper mills in the manufacture of tin plate is threefold: to impart the desired surface finish to the product, to produce acceptable flatness and to develop the desired mechanical properties.

The surface finish is controlled largely by the smoothness of the exit work rolls. Smooth-ground rolls impart a bright, dense surface finish whereas rough-ground or shot-blasted rolls impart a rougher finish. All grades except matte finish (No. 5 finish) and silver tin are finished with smooth rolls.

Flatness is obtained by proper adjustment of such factors as roll contour, finish, pressure and strip tension.

For the softest tempers, temper rolling (also called skin rolling) is done on a conventional four-high single-stand coil temper mill although these tempers can be produced with care on two-stand tandem temper mills. The objective here is to flatten the strip and impart the proper surface in one pass with a minimum of hardening effect due to cold working of the product, the thickness reduction being held to less than 1 per cent. For the higher tempers, a two-stand four-high tandem temper mill is employed, or the strip is given two passes through single-stand mills. On such material, temper rolling has the added function of increasing the steel hardness, and thickness reductions of 2 per cent and higher are common. As an aid to temper rolling, the work-roll surface of the first mill stand is relatively rough, being rough-ground or shot blasted, and the work rolls of the second stand or finishing mills are ground smooth. Thus, superficial hardening, arising from second-stand smoothing of the rough strip surface imparted in the first stand, is added to the full-section hardening of the actual strip extension under rolling pressures and reel tension. Modifications of temper mills for tin plate include means for applying high strip tensions and for developing heavier drafts in the first stand, as mentioned in the section on finishing.

Rockwell hardness-testing machines are used to measure the "temper" of the coils. If the hardness values are too low, the coils are rerolled.

After temper rolling, coils are conveyed either to a shear line, where they are cut to sheets for sale as black plate or for hot-dip tinning, or to a side-trimming and recoiling line for preparation for electrolytic tinning.

SECTION 5

HOT-DIPPED TIN PLATE

Shearing Practice—At the flying shears, the coils for conversion to hot-dipped tin plate are side trimmed and cut to sheets. A micrometer, operating continuously, rejects off-gage sheets which are diverted automatically

into an off-weight piler and thus separated from product of correct thickness. Perforated sheets are detected with an "electric eye" and are rejected automatically.

White-Pickling Practice—Pickling to remove all sur-

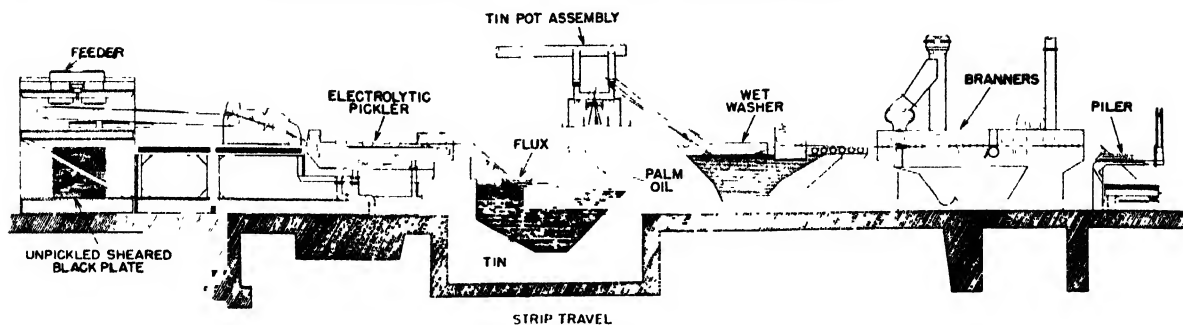


FIG. 37-1. Schematic arrangement of equipment comprising a modern tinning stack.

face contaminants, such as light oxides resulting from the annealing treatment before tinning, is commonly termed **white pickling** to differentiate from other pickling processes. A clean, uniformly pickled surface is a requisite for successful hot tinning. The cold-reduced product is pickled in sulphuric acid of 2 to 6 per cent concentration at 150° to 180° F for 2 to 4 minutes, followed by rinsing in cold water. Pickling inhibitors frequently are used to prevent over-pickling or etching of the plate. The pickled steel is stored in a "bosh" or steel storage tank containing a 0.10 per cent hydrochloric-acid solution until ready for tinning.

Electrolytic pickling is supplanting white pickling in many plants. Sheared plate is fed automatically into the electrolytic pickler and thence into the tin pot (Figure 37-1). Various feeding devices may be employed and common acids used in such a unit. Its advantages include the elimination of the heavy manual labor necessary in hand-loading and unloading the white pickler and in providing uniform and controllable pickling action.

The Hot-Dip Tinning Operation—A typical hot-dip tinning shop or tin house normally consists of a long rectangular building with the tin stacks arranged side-by-side in a straight line. The individual tin stacks usually are hooded completely to draw off all fumes from the flux, palm oil and tin baths and to minimize as much as possible loose bran particles in the air from the branning unit. A modern tinning stack consists of an assembly of equipment arranged to perform a number of operations in a continuous manner (Figure 37-1). The integral parts of this assembly, in the order in which they perform their function on the sheet being coated are as follows:

1. The feeder
2. The tin pot and tinning machine
3. The wet washing machine
4. The branner
5. The piler

Modern hot-dip tinning lines are commonly of two sizes,

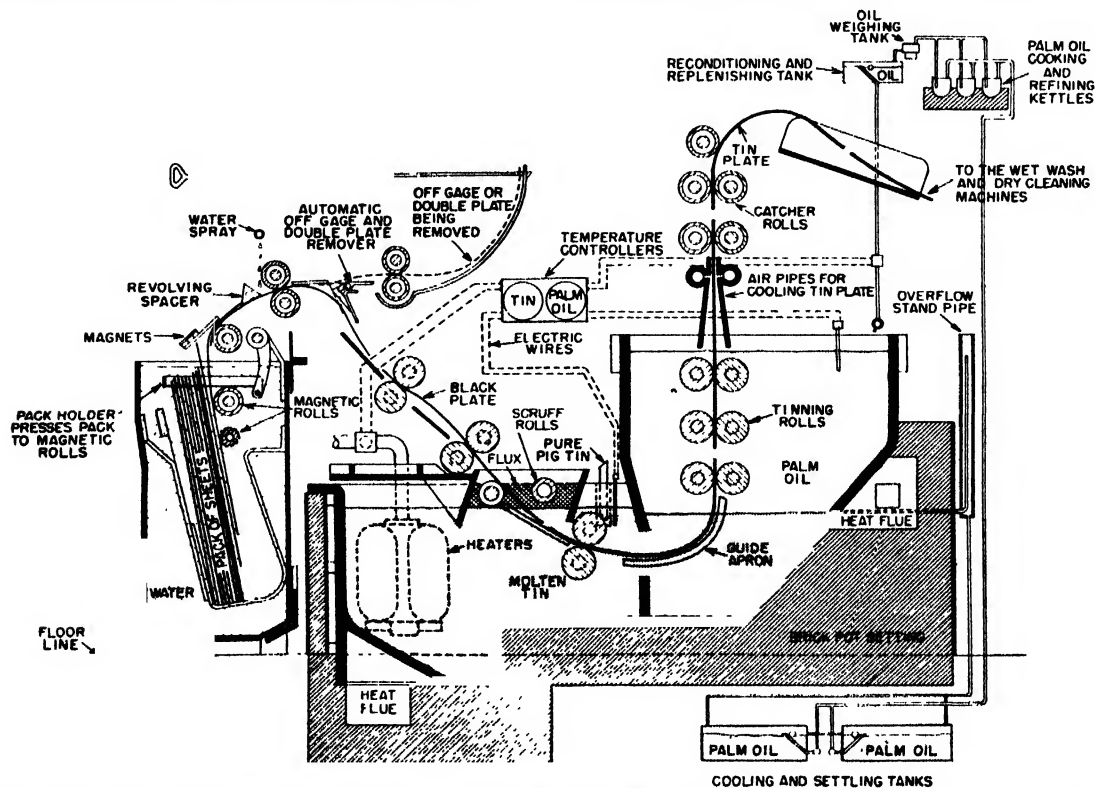


FIG. 37-2. Schematic diagram of a tin pot and tinning machine. The feeding unit at the left is known as a Poole feeder. The brushes on the tinning rolls have been omitted.

the 64-inch and the 75-inch machine. These designate the length of the roll bodies in the tinning machines. The 64-inch normally operates as a two-way machine (two sheets of tin plate pass side-by-side through the molten tin bath) whereas the wider 75-inch machine may be used as a three-way machine on narrow product.

The feeding mechanism varies in design from plant to plant. Feeding may be done manually or by automatic feeding machines. The Poole feeder (Figure 37—2) is a common automatic type, in which individual sheets are lifted by a magnetized roller from the feeder tank to the supplementary tinning machine equipment.

This tinning equipment consists of a series of guides and rolls to convey and guide the sheets being coated from the feeding mechanism downward through the molten flux and tin baths, then upward out of the molten tin and through a bath of hot oil onto the conveyor, as shown in Figure 37—2. The axes of each set of rolls lie in the same horizontal plane. Three pairs of rolls are positioned in the hot oil so that the lowest pair is slightly above the molten tin level and the top pair is just below the top level of the oil. The tinning machine rolls normally range from $3\frac{1}{2}$ to 4 inches in diameter and are of high-carbon steel heat treated to high hardness. Their surfaces are very accurately finished by machining and grinding. The tin pot, which holds from 8,000 to 17,000 pounds of commercially pure molten tin at about 620° F, is filled to a point approximately 4 inches from the top of the entry side. The pot is made of plate steel, enclosed in a brick chamber, and generally is heated by gas or oil burners or by an immersion heater. The flux box is a long narrow box with its sides converging downward but without any top or bottom. It is located on the entry side of the tin pot and the lower edges extend downward into the molten tin bath. The flux bath, approximately 3 to 4 inches in depth, floats on the surface of the molten tin bath. The space between the bottom of the flux box and the opening in the partition is equipped with top and bottom guides. In addition, either an apron or guides are provided in the tin bath chamber to control the path of the plate from the flux box into the nip of the bottom rolls in the tinning machine (see Figure 37—2). The flux in common usage is essentially a water solution of zinc chloride which may contain some ammonium chloride.

The tinning rolls in the palm oil are equipped with wipers or brushes, whose function it is to aid in the control of the tin coating. These brushes are, of course, adjustable from outside the actual oil chamber. They commonly are constructed of either soapstone or asbestos or of laminated combinations of these materials. Directly above the exit set of rolls, approximately 18 inches away, is the catcher. This mechanism consists of a frame supporting guides and rolls which feed the coated and oily sheets from the exit side of the pot past a jet of cooling air to the conveyor.

The palm-oil bath on the exit side of the tin bath, in addition to protecting the surface of the molten tin bath from oxidation, tends to keep the tin coating on the sheet in a molten condition and thus allows the tinning-machine rolls and brushes to control the distribution and thickness of the tin layer. Temperature of the palm oil is maintained as low as is consistent with satisfactory operating conditions, usually about 465° F. Oil generally is recirculated through a central reservoir. Excessive oil temperatures may cause the formation of oxide films of various colors, resulting in unsightly tin plate. Tin oxide films are discussed more fully later under "Metallurgical Aspects." The viscosity of the palm oil is also of prime importance because excessive oil is carried out of the bath if it is too viscous.

After being transferred by the catcher to the conveyor, the plate travels through the wet washing machine (Figure 37—1). The purpose of this unit is to remove the excess of palm oil from the surface of the coated sheet. The washer contains a hot water solution of soda ash or trisodium phosphate at a concentration of 0.10 to 0.15 per cent total alkalinity. The solution in most plants is circulated from a central storage tank which supplies the entire tinning shop. The inflow of solution is controlled automatically and the overflow is collected in a drainage system or sump. The palm oil is removed at this point for future use.

The washed sheet is now relatively free from oil. Since the sheets are unsatisfactory for ultimate use without some oil being present, the sheet is next passed into a branner where a light oil film is distributed across the sheet surface. The branner unit consists of tandem sets of cleaning rolls composed of thousands of cotton flannel discs about 4 inches in diameter tightly compressed on a long square steel mandrel. Bran or middlings (a mixture of wheat and rye) are fed into the first set of rolls to absorb the moisture, and distribute uniformly the oil that remains on the plate surface from the washer. The last rolls of the machine, together with a combination of vacuum and pressure hoods, are used to minimize the presence of bran on the sheets. From these hoods the product passes down the conveyor to the piler, which automatically stacks the plate in neat piles without scratching or otherwise marring the tinned surface.

Coke Tin Plate—Tinning Roll Practice—The most commonly supplied grades of hot-dipped tin plate are Standard and Common Cokes. On Standard Cokes the mills aim for an average tin coating weight of 1.35 pounds per base box of plate and for the Common Cokes the aim is an average tin coating weight of 1.10 pounds per base box. In order to obtain the best possible uniformity of coating, the middle and top sets of tinning-machine rolls in the exit side of the pot usually are grooved spirally. The depth of grooving varies with the results desired for a given tin-house practice, but in general the grooves on the top set of rolls are shallower than on the middle set. The heavier the coating desired, the deeper is the grooving. As previously mentioned, the roll brushes also play an important part in the control of coating weight and coating distribution.

HOT-DIPPED TIN PLATE COATING WEIGHTS

The following is a quotation from American Iron and Steel Institute Steel Products Manual "Tin Mill Products," July 1954, with the table number altered to conform to present text:

"There are several classes of hot dip tin plate which are designated by terms that have been used since the early days of tin plate manufacture. The distinction between classes is based upon a difference in the weight of tin used to produce a base box of tin plate. This weight, by custom, is referred to as tin pot yield and reflects the total weight of tin used in the manufacturing operation. The actual average weight of tin coating on the plate is less than the pot yield for each class due to unavoidable losses of tin in the tinning operation.

"On hot dip tin plate, the coating is mechanically distributed over the plate surface by tinning rolls. Due to the nature of the hot tinning operation and the elevated temperature of the tin metal it is not possible to control, within close limits, the tin coating weight on individual plates and from plate to plate.

"Coke Tin Plate, other than Best Cokes and Kannars Special Cokes, did not carry a specified coating weight prior to the second World War. The necessity for the conservation of tin caused the United States Govern-

ment to issue restrictions on the amount of tin to be used on tin plate. Federal Material Orders were promulgated which limited the amount of coating on hot-dipped tin plate on the basis of pot yield, and later orders stated, 'coating shall be determined on the basis of pot yield, in the case of hot dipped tin plate.' The orders limited the pot yield, or the amount of tin consumed in the manufacture of hot dip tin plate, requiring that the over-all average must not exceed 1.25 pounds per base box for 1.25 pound coke tin plate and 1.50 pounds per base box for 1.50 pound coke tin plate, now customarily designated as Common Cokes and Standard Cokes, respectively.

"The tin coating weight test values shown in Table 37—II are the minima which one may expect to encounter when the plate is tested according to the procedure recommended under Sampling. Experience has shown that the average tin coating is substantially higher than these values. Typical tin coating weight distribution data are contained in Report of Industry Survey of Coating Weights of Hot Dip Tin Plate, Contributions to the Metallurgy of Steel, No. 39, April 1952, published by the American Iron and Steel Institute. Production control test records indicate that over an extended period the average coatings on Common Cokes and Standard Cokes will approach 1.10 and 1.35 lb. per base box, respectively. Due to limitations of the process, these averages cannot always be maintained in the production of specific items.

"The coating weight in localized areas on occasional plates, as determined by spot samples selected in the manner described under the recommended sampling procedure, varies as much as 0.25 lb. per base box under the values shown in Table 37—II. Exceptional instances of even lower coated areas sometimes occur.

"Other classes of coke tin plate carrying heavier coatings, as shown in Table 37—II, are Best Cokes and Kanners Special Cokes.

"Charcoal Tin Plate—The distinguishing characteristic of charcoal tin plate is higher luster, due to heavier coatings, and these in the order of their increasing weights are designated: 1A and 2A. Coating weight test values for these classes are shown in Table 37—II. Charcoal tin plate is used in the manufacture of articles, the appearance of which is an important consideration, and where greater resistance to corrosive conditions is requisite.

Table 37—II. Coating Weights of Standard Grades of Tin Plate

Class Designation	Minimum Average Coating Weight Test Value (Pounds per base box)
Common Cokes (1.25 lb. per base box pot yield)	0.85
Standard Cokes (1.50 per base box pot yield)	1.05
Best Cokes	1.19
Kanners Special Cokes	1.40
1A Charcoal	1.80
2A Charcoal	2.30
3A Charcoal	2.80
4A Charcoal	3.50
5A Charcoal	4.20
Premier Charcoal	4.90

"Silver Tin—Silver tin has a dull surface, the base plate of which has been mechanically roughened and which carries a heavy tin coating of no specific weight.

"Sampling—When coating weight check tests are conducted on a shipment of any specific item of hot dip tin plate, based on experience the following procedure is recommended.

"One plate is taken at random from each 50 packages, with a minimum of three sample plates, each taken from different multi-package units from any one item of a specific shipment. Each sample is spot-tested at three positions taken on a diagonal across the plate. Spot samples are customarily discs or squares, each of 4 sq. in. area. In order to secure representative samples they are taken so that the outer edge of the sample is at least one inch from any edge of the plate. The average value of all spot tests on all plates tested represents the average coating weight test value.

"Experience has revealed significant variations in test results. The variations arise in part from the use of different tin coating determination methods, and they are also affected by the personnel and techniques of the testing laboratories. The significance of test results and comparison of them with check test results depend also on close control of sample locations on the plates tested. For such reasons great care should be exercised in making the tests as well as in the interpretation of the results of those tests. (End of quotation)."

Heavy-Gage Tin Plate—For the heavier gages, a longer sweep is necessary in passing through the pot to avoid pronounced "bowing" or "breaking" of the plate and for this purpose, a "hillside" machine is usually used. In this design, the tinning rolls are positioned at an angle (approximately 45 degrees) rather than one of them being directly over the other, and more liberal radii of curvature are employed on guides and aprons.

Manufacture of Charcoal Tin Plate—In order to produce the heavier coated charcoal plate, it is necessary to modify the tinning equipment in such manner that the tinning rolls carry more metal. This is accomplished in a conventional machine by replacing the brushes on the top or top-and-middle rolls with pans. Tin is supplied to these pans by ladling or pumping molten metal into them; the bottom of the rolls revolving in the tin. These rolls have grooves up to 0.008 inch deep.

Assorting of Hot-Dipped Tin Plate—After the tinning operation, the coated sheets must be inspected and classified. Individual standards for a given grade and application may vary somewhat, but the general classifications are (1) primes, (2) seconds, (3) menders, and (4) waste-waste. These classifications are defined as follows:

1. **Primes** are tin plates free from defects readily observed by the unaided eye.
2. **Seconds** are tin plates having imperfections in moderate degree or extent.
3. **Menders** are those tin plates having imperfections in coating, which product, upon recoating, will produce either a "prime" or "second" quality sheet.
4. **Waste-waste** are tin plates which, either through imperfections of coating or base metal cannot be recoated to produce a "prime" or "second" quality sheet.

The plates from the tin stack pilers are inspected visually on both sides for various base-metal surface defects, tinning defects and handling damage. During the assorting operation, the product is checked for individual sheet weight and commercial flatness. The sorters soon

develop a high degree of skill in detecting slight differences in weight and recognition of surface defects as they handle individual sheets. In recent years some semi-automatic assorting machines have been installed. By the latter method, the individual plates travel on belt conveyors by an inspection station and are automatically turned over to permit two-side inspection. In the manual assorting operations, the same end is accomplished but the assorter must turn over every plate by hand and

properly classify it at the same time. Each assorter maintains separate bins for menders and waste-waste, but most hot-dipped tin plate orders permit the inclusion of primes and seconds together. Hence, these two quality classification types are placed in the same pile after classification. Following the assorting operation, the prime and seconds plates are "reckoned" or counted individually, packaged in containers as specified by the customer, tagged, wired and shipped.

SECTION 6

ELECTROLYTIC TIN PLATE

Introduction—Prior to 1937, all tin plate produced commercially was manufactured by the hot-dipping process. While the electro-deposition of tin on steel had been a known process for many years, its application to the production of tin plated sheets could not be made to compete economically with the hot-dipping process due to much higher equipment and labor costs. However, with the introduction to the sheet and tin plate industry of continuous cold-reduction mills in the early 1930's, the possibility of continuous high-speed electro-tinning became obvious. As early as 1935, small experimental units were designed and constructed capable of continuously electroplating tin on steel strip moving at relatively high speeds. From the results of these studies, electrolytic tin plate appeared on the market as a commercial item in 1937.

The early development of electrolytic tin plate was given a major impetus by the requirements of the dry package market for a light-coated product which could not be produced by the hot-dip process. The precipitous necessity to conserve vital tin during World War II resulted in a phenomenal expansion of this development so that by 1948 over half of the tin plate produced was light-coated electrolytic tin plate and by 1955 it was 80 per cent.

The superiority in tin coating uniformity which electrolytic tin plate exhibits as compared to hot-dipped tin plate, together with the close control of tin coating weight which is obtained, has resulted in large savings of tin.

Basic Principles of Electroplating—To the famous English scientist, Michael Faraday, belongs the credit for placing electrochemistry on a quantitative and orderly basis. In 1833, he postulated certain laws which today bear his name and which can be summarized as follows:

1. In any electrolysis, the quantities of materials liberated at the electrodes bear a direct relationship to the quantity of electricity passed through the system.

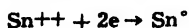
2. An equal number of equivalents of substances are set free by the same quantity of electricity.

It can be seen from a study of these laws that tin plating can be controlled when the electrode reactions are known. Electroplating can be accomplished with acid or alkaline electrolytes.

In the acid processes the anode reaction consists of direct oxidation from metallic tin to the bivalent stannous ion with the liberation of two electrons as follows:



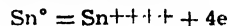
The reaction is usually 100 per cent efficient, thus making the quantity of tin driven into solution directly proportional to the electrical energy used. The reverse reaction takes place at the cathode,



thus causing metallic tin to be deposited on the cathode which, in electrolytic tinning lines, is the moving steel strip. The cathode reaction may be 100 per cent efficient or it may be somewhat less. The latter condition is desirable since the resulting tin build-up in the solution eliminates the necessity for additions of tin salts when solution losses occur. The use of organic addition agents in acid baths appears to be essential to the production of dense adherent tin deposits. While such agents may lower the operating cathode efficiency somewhat, the effect is slight.

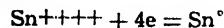
There are several successful acid baths used for electroplating, among them being the phenol-sulphonic acid bath, the fluoride-chloride bath, the fluoborate bath and the sulphate bath. In each of these the electrode reactions are the same. Additions of acid are commonly made to raise the conductivity of the electrolyte and thus lower the plating power consumption.

The electrode reactions which take place in the alkaline baths are somewhat complex. The over-all effects are well known, however, thus permitting satisfactory control of the deposition. The net reaction at the anode is to oxidize tin from its metallic state to the quadri-valent stannic condition, as follows:



It will be noted that four electrons must be liberated to drive an atom of tin into solution in this process as compared to two electrons in the acid process. This means that twice as much electric current is required to dissolve one pound of tin. It must be emphasized that the anode reaction shown above is an over-all effect since the actual mechanism at the anode indicates the formation of intermediate tin compounds during the transfer process. This complexity of the tin anode reaction in the alkaline stannate process limits its operating current density and thus necessitates the use of large plating areas to achieve high operating speeds.

At the cathode (steel strip) the tin is reduced again to the metallic condition with the absorption of four electrons per atom:



When operating temperatures are kept above 200° F, the electrode efficiencies of the alkaline processes are quite close to 100 per cent.

Commercial Electrolytic Tinning Equipment—The equipment used for the production of electrolytic tin plate in no way resembles the more common electroplating equipment seen in most plating shops. The complex and bulky machinery required to handle the heavy steel coils and the fast moving steel strip itself is much more expensive and usually takes up more space than the plating unit proper.

Figure 37—3 is a schematic diagram of the equipment used in United States Steel Corporation units. An actual

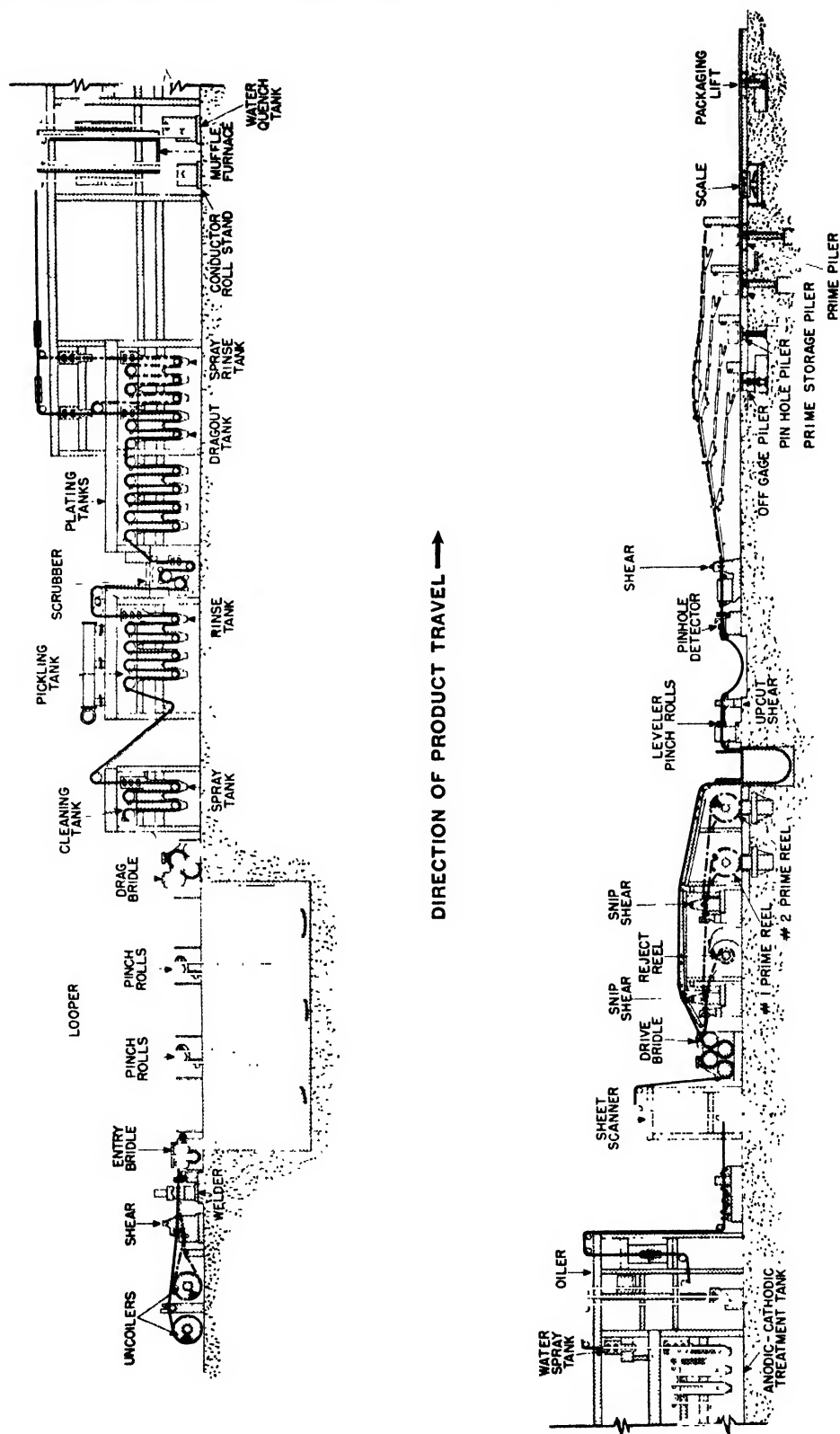


FIG. 37—3. Schematic arrangement of the handling and processing units comprising a sulphonic acid electrolytic tinning line.

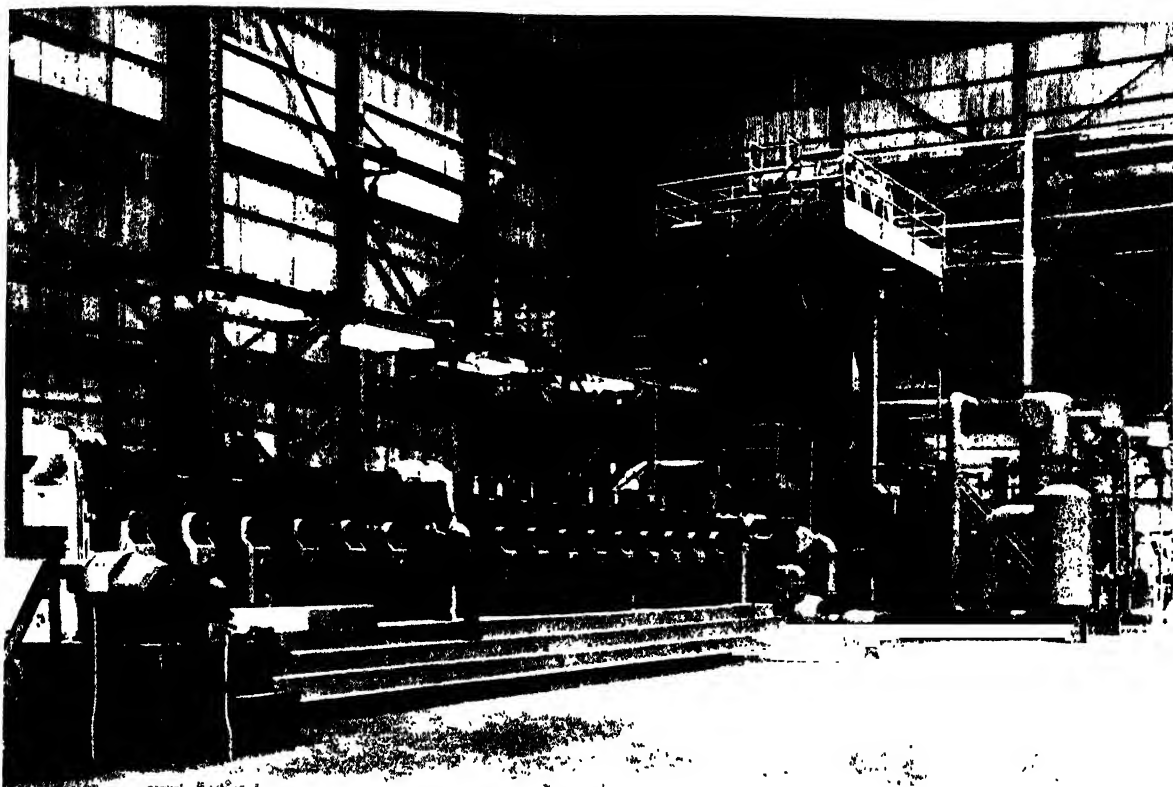


FIG. 37—4. Close-up view of the plating unit of a sulphonic acid electrolytic tinning line, with melting tower immediately behind in right center background.

installation is shown in Figures 37—4 and 37—6 through 37—9. The electrolyte used is a solution of stannous sulphate and phenolsulphonic acid, capable of carrying current densities up to 400 amperes per square foot or more at cathode efficiencies of 95 per cent or better; such operating conditions permit the use of relatively small plating tanks. The alkaline stannate units, also in wide use, are basically similar in design but require considerably larger plating units. This difference in design is necessary because of the low current density limit of the alkaline bath, which is usually a solution of sodium stannate and sodium hydroxide. The alkaline electro-

lytes in general operate at current densities up to 60 amperes per square foot.

The third common type of electroplating line makes use of an acid halogen type electrolyte and is so constructed that the bottom side only of the strip is plated in horizontal baths during its travel through the unit. The plating section of the line is composed of many individual plating tanks with tin anodes resting on the bottom of each. The horizontal tanks are set one above another. As the strip leaves a lower tank and passes over a roller guide to reverse its direction of travel and enter the tank at the next higher level, what was the bottom

FIG. 37—5. Side-trimming unit that prepares cold-reduced, annealed and temper-rolled coils of flat steel for electrolytic tinning by removing excess material from side edges to produce strip of exact desired width. An air-operated shear following the side trimmer cuts off the crop ends of coils preparatory to seam welding of successive coils together to form larger single coils weighing up to 30,000 lb.





FIG. 37-6. Entry end of an electrolytic tinning line, with one uncoated coil being paid off into the line and another in reserve position. The trailing end of the coil being fed will be welded to the leading end of the reserve coil (after squaring in the shear) to provide a continuous feed to the line.

surface of the strip becomes the top surface. Thus, the strip is reversed on each plating level to permit even deposition on each surface of the strip. Current densities vary from 50 amperes per square foot upward.

Regardless of the type of plating unit used, the steel strip for manufacturing electrolytic tin plate is similarly prepared. The strip itself is manufactured as described elsewhere; that is, it is hot rolled from slabs, continuously pickled, cold reduced, electrolytically cleaned, box or continuously annealed and temper rolled. In box annealing, a protective deoxidizing atmosphere is used to insure a uniform, bright surface and the temper rolling operation is carefully controlled to maintain this condition. At this point in the processing, strip for electro-tinning takes a different course than that for hot-dipped tin plate as the retention of a long, uninterrupted unit of strip is needed for the success of the operation. Accordingly, the coiled temper-rolled strip is usually delivered to the side trimming units (Figure 37-5) where it is uncoiled, run through pairs of rotary knives and recoiled. The rotary knives are adjustable in such a manner that the strip can be very accurately side trimmed to the width ordered by the customer. It is general practice to adjust the slab selection and hot-rolling practice to obtain about $\frac{1}{2}$ inch protective over-width on the process strip which is removed at these slitters. In some instances, this strip side trimming is done after tinning in a separate shearing operation. In many plants where the strip is trimmed before coating, lap welding equipment is included in the side trimmers to permit welding several coil units together and thus provide a continuous section of steel strip for plating, up to six or seven miles long. In other plants, the side-trimming operation is performed in the electrolytic tinning unit itself.

The entry end of an electrolytic line is usually so designed as to provide two uncoilers in line (Figure 37-6). This permits the operator to "pay off" from one uncoiler while charging a coil into the other. Electrolytic tinning units do not require any special type of unreeling equipment and either the conventional cone or expanding

types are in use. All of the auxiliary uncoiler equipment such as brakes, forward and reversing drives, hydraulic lifts, and strippers are usually to be found on all units.

In preparing a coil for processing, the lead edge of the strip is manually engaged in a set of small pinch rolls which can be opened and closed by air pressure and which are usually motor driven. The function of these rolls is to permit the operator to advance the lead edge of a new coil into the welding assembly. This welding assembly consists primarily of an up-cut shear, a lap welder, and a set of large pinch rolls.

It is desirable to maintain a high strip speed in the plating baths, so facilities are provided to join fresh coils to the strip without reducing line speed. As the coil in process is being unrolled, the operator takes care that the maximum amount of strip is contained in the looper located just after the large welder pinch rolls. When the last several wraps of the coil unwind from the uncoiler in use, the operator stops the large welder pinch rolls, "trues-up" the tail end of the coil with the up-cut shear, moves this tail end into welding position with the head end of the new coil, welds the two together and starts the welder pinch rolls into motion. All this must be done before the strip previously stored in the looping unit has been completely used. The welder pinch rolls are then run at some speed greater than line operating speed to refill the loop at which point automatic electrical devices slow these rolls back to synchronism with the rest of the unit. Since these looping units are usually designed to accumulate 300 to 500 feet of strip, it is obvious that the welder operator must act rapidly in order not to slow a line operating at 600 feet per minute. These loopers may be a "tower" type, a "pit" type, or a combination of the two. The "tower" type looper, as its name indicates, is constructed predominantly above ground and usually consists of fixed and movable sets of rolls over which the strip is passed. The top set of rolls moves down or up, depending on whether strip is being expended from or accumulated in the equipment. The "pit" type looper consists of a deep pit in which hangs a long catenary of strip.

From the looper the strip enters the main process section of the line. In the acid lines, the process section usually consists of a dynamic tension device (often called a tension bridle), an alkaline electrolytic cleaner, a rinsing unit, a pickler, another rinsing unit, a plating unit, a third rinsing unit, a fusion unit, a quench tank, a chemical treating unit, a fourth rinsing unit, a drying unit, an oiling unit, a drive or pull-through bridle, and finally a set of recoilers or a shear or both. The alkaline lines have the same sequence of units except that the alkaline cleaner is not part of such lines inasmuch as the alkaline plating bath itself does sufficient cleaning.

The function of the tension bridle is to produce sufficient drag on the strip to maintain a positive strip tension throughout the line. It consists of a series of rolls, some of which may be pinch rolls, through which the strip passes out of the looper. These rolls are usually geared together and to a generator. By controlling the field voltage on this generator and "shorting out" the armature through a resistance, a controllable drag can be applied to the strip through the geared rolls.

In the acid electrolyte units, the strip passes from the drag bridle to the alkaline electrolytic cleaners. In some lines these cleaners are of conventional design, with horizontal electrodes as described in the section on the electro-cleaning of cold-reduced strip. In other lines, vertical units are used. Current densities in these units vary from 50 to 300 amperes per square foot. The clean-

ing solutions are alkaline detergents. Strip polarity may be either anodic or cathodic since it is found that both conditions yield good cleaning performance.

The strip passes from the alkaline cleaner into a rinsing and scrubbing unit. Its function is to remove all alkali from the strip in preparation for the pickling operation. This rinsing unit is usually comprised of water sprays playing on both sides of the strip and of rotary bristle brushes which rotate vigorously against the strip. There is a trend toward the use of high-pressure water sprays and the elimination of brushes in such units.

The strip pickling units, which are used on all electrolytic tinning lines, may be of the hot immersion type or of the cold electrolytic type. The immersion type usually consists of a large rubber- or brick-lined tank through which the strip passes vertically or horizontally. These tanks are filled with hot sulphuric acid of a strength varying up to 12 per cent and the pickling time is regulated by the operating strip speed. The electrolytic picklers are usually small units as the control of pickling is maintained by regulation of the electric current. These units are built similar to the alkaline cleaning tanks and the electrical circuit is also similar; the strip may be alternately anodic and cathodic, assuring effective surface cleaning. After pickling, the strip is again rinsed in a unit similar to the one used after the alkaline cleaner and enters the plating tank.

As explained earlier, the main difference in the vari-

FIG. 37-7. Tin anode being placed in the plating tank of an electrolytic tinning line, with other anodes lined up awaiting placement.



ous electrolytic tinning units lies in the type of electrolyte used; hence, the type of plating tanks used. The unit used in phenolsulphonic acid lines is designed for operation at high current densities and consists of four vertical compartments in each of which the strip passes over metal contact rolls and down into the electrolyte between banks of tin anodes. (A fifth compartment in line with the four plating tanks and identical in appearance plays no part in the plating operation but merely collects the solution dragged out of the plating system.) Thus, the current can be considered to pass from the tin anodes through the solution to the strip and up the strip to the metal deflector rolls which act as the negative contact of the system. By such a circuit, the tin is deposited from the solution onto the strip and is also equally driven into solution from the anodes. The tin anodes consist of tin bars which, for example, may be 3 inches by 4 inches in cross section and approximately six feet long, weighing close to 200 pounds each (Figure 37-7). The life of the individual anode depends on the quantity of electric current passing through it. The electrolyte is constantly recirculated through the plating tanks after passing through a settling tank and several water coolers. The temperature of the bath is generally maintained at about 100° to 120° F. All of this equipment must of necessity be constructed of corrosion-resistant materials and care must be taken to provide sufficient insulation in the system to prevent electric current leakage. Generating equipment capable of developing 45,000 amperes at 8 volts has been a typical installation.

The alkaline-type plating bath, while essentially of the same basic design, requires much greater floor space than the acid type. Inasmuch as alkaline stannate baths are operated at temperatures in excess of 200° F, no recirculation of the electrolyte for cooling is necessary.

These plating tanks are fabricated as a single large unit with contact rolls placed at the top of the tanks and rubber deflector rolls placed in the bottom. The tin anodes used in the alkaline units are usually very large slabs of tin hanging under and between the contact rolls. These slabs are large enough to allow several weeks of operation before replacement is necessary. Alkaline lines are provided with up to 90,000 amperes at 10 volts, with current densities of about 45 amperes per square foot permitting strip speeds up to 600 feet per minute on half-pound coatings.

The third type of unit (halogen) consists of a series of small cells, each with its own circulation system, contact roll and anode bank. These tanks are so designed that the strip is barely immersed in the electrolyte and is plated on the bottom side only. After passing through a number of these units, the strip is deflected upward and backward so that the original top of the strip now becomes the bottom. It then passes through another series of similar plating cells until an equal amount of tin is deposited on this side of the strip. The halogen type electrolyte used exclusively in these units is constantly recirculated through the cells, cooled and filtered during its circulation. The tin anodes used in the individual cells resemble regular pigs of tin and rest on side supports just under the strip pass line. Halogen lines have been designed for strip speeds greater than 2000 feet per minute. Generator capacity exceeds 100,000 amperes at a voltage of from 8 to 10 volts. The current density used is approximately 300 amperes per square foot.

The plated strip, regardless of plating process, is now freed from the dragout and rinsed in pure water or condensate. The electrolyte which is dragged into this wash water is all or in part returned to the plating tank. To accomplish this, drag-out recovery systems varying

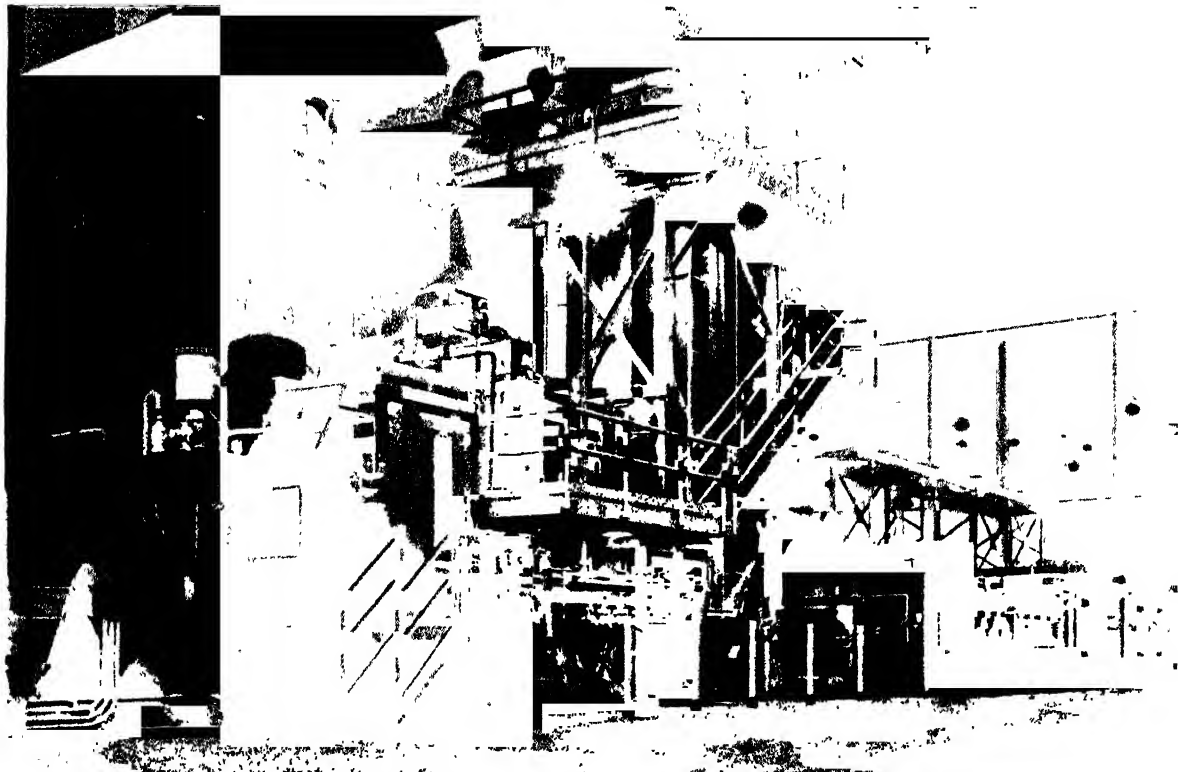


FIG. 37-8. General view of an electrolytic tinning line, looking toward the entry end and showing the melting tower in the center foreground.

from complex recirculation and evaporation systems to simple counter-current rinsing with partial recovery are used. Too much emphasis cannot be placed on the necessity for efficiently recovering the dragged-out electrolyte since solution losses from this source can be enormous at high speeds, reaching as much as 30 gallons per minute.

The tin coating, as it emerges from the plating bath, is grey-white and semi-lustrous. It does not in appearance resemble tin plate as it is commonly known. While some attempts have been made to plate bright tin coatings, no commercially successful developments of this type are at present practiced. Instead, it is universal practice to melt and quench the electrodeposited tin which gives it the brilliant luster typical of hot-dipped plate (Figure 37-8).

There are three types of units in which this tin fusion is accomplished. They vary only in the source of heat and not in basic construction. One of these melting units makes use of electrical resistance heating. The strip is run in a vertical loop between two contact rolls, the second of which is partly immersed in water. These two contact rolls form the terminals of an alternating-current circuit in which the strip is the closing resistance. Thus, by regulation of the current flow through the strip (or applied voltage) it is possible to bring the plated strip up to 450° to 455° F just prior to passage into the water. The maximum temperature to which this strip can be heated falls in a very narrow range because excessive heating causes discoloration of the product and/or high iron-tin alloy, whereas insufficient heating naturally results in failure to melt. A second type of melting unit utilizes high-frequency induction for melting. In this unit, the strip passes down and through a water-cooled copper coil on which is impressed a high-frequency voltage. The induced eddy currents in the tin plate strip cause it to heat up with resulting fusion of the tin coating. Control is again exercised by voltage variations on the induction coil terminals. The third type of melting unit in commercial use is gas-heated and equipped with special ceramic burners which radiate controlled amounts of heat to the strip. Control on such units is exercised by lateral movement of the burner banks, closer to or farther away from the strip, depending on whether more or less heat is required. Further limited control is available by regulation of the gas supply to the unit. As in the other units, the strip is

quenched directly after fusion of the tin is accomplished.

The fused and quenched coating is now given a **filming** treatment which may be either chemical or electrochemical, as described later in this section. After such treatment, the tinned strip is rinsed with clean water and is dried either by blasts of hot air or high-pressure steam.

Unlike hot-dipped tin plate, the electrolytic plate is not oily as it emerges from the coating operation; hence, it is necessary to deposit a controlled oil film on the product in order to improve its handling properties in succeeding operations. The oil used is usually cottonseed (synthetics such as dioctyl sebacate, DOS, are also finding favor) which is applied in several ways. Two oiling methods now widely used on electrolytic tinning units are the emulsion and electrostatic processes. The first of these consists of passing the strip through a bath or a spray of unstable oil emulsion. Control of the resulting oil film is exercised by emulsion composition and operating temperature. After exposure to the emulsion the strip passes through wringer rolls, is rinsed and dried and passes into the pull-through bridle ready for recoiling or shearing. The electrostatic method of oiling strip makes use of a high potential between the strip and a fixed electrode which creates a powerful electrostatic field around the moving strip. Into this electrostatic field is allowed to rise vaporized palm or cottonseed oil. Since such vapors consist essentially of charged particles, they are deflected onto the strip by the proper adjustment of strip polarity. Despite the relative complexity of the equipment, these oiling units operate very satisfactorily and economically.

The strip next enters the unit which supplies tractive power to the strip to pull it entirely through the electrolytic line. This piece of equipment is called the "pull-through" or drive bridle. It actually does the pulling of the strip through the process section and is the basic unit with which the plating current and both entry and delivery ends are synchronized. The equipment itself is usually quite similar to the tension bridle described elsewhere in this section. However, instead of a drag generator being geared to the rolls, they are coupled with a powerful motor which is sufficiently large to pull the strip through the whole process section at high speeds. A tachometer generator attached to this motor provides the impulse to regulate the plating and melting currents in required relation to strip speed.



FIG. 37-9. Discharge end of an electrolytic tinning line, showing sheared-plate pilers (left) and inspector.

A great many of the tinning units currently operating are provided with loopers and two recoilers on the delivery end into which the strip passes from the drive bridle. Such an assembly performs in a reverse manner to that of the entry end previously described. The looper acts strictly as a strip accumulator. It normally operates with a minimum length of strip between rolls so that it is capable of accumulating strip at line speed while recoiler switching is underway. As soon as this switch is made, the operating recoiling unit speeds up until the loop is absorbed, at which time it automatically falls into synchronism with the processing section of the line. The coil of coated product is discharged from the inoperative recoiler and sent to the conventional shearing units where it is sheared to size, oiled (if not so treated in the electrolytic unit itself), assorted, counted and piled. At least one modern line is now provided with both flying shears and quick-change coilers. The latter need to be used only for product which is shipped to the customer in coils, and are not preceded by a looping device.

In those units where flying shears only are included in the equipment assembly, no "loopers" are necessary. Instead, a very small catenary is maintained immediately in front of the shear in order that perfect guiding into the shear knives can be accomplished. Failure to guide properly into a shear results in miscutting and "out-of-square" sheets. The flying shears themselves are conventional units for light-gage-strip shearing, as described elsewhere in this chapter. There are some modifications, however, which should be mentioned. The most important ones concern the adaptation of these units for careful inspection, classification and counting of the product (Figure 37-9).

The flying shears are equipped with four piling stations and a system of conveyor belts which allows the deposition of sheets in any of these pilers at the will of the operator.

In lines containing four pilers, all sheets containing

perforations are accumulated in the first piler. Sheets containing bad surface defects and product which is outside the accepted gage tolerances are accumulated in the second piler. The third piler is intended for disposition of sheets which, while not acceptable as prime product, contain only superficial coating defects. Such product is usually referred to as "menders" inasmuch as it is common practice to pass it through a hot-dip unit for conversion to hot-dipped tin plate (thus "mending" the product). The fourth or end piler is the so-called "prime" piler. As the sheet counter is located between the third and this prime piler, the product delivered into it is usually counted into ten-, twelve-, or fifteen-package bundles (1120, 1344 or 1680 sheets) ready for packaging and shipping.

The methods of inspection and classification of electrolytic tin plate on these flying shears are rather ingenious. Located somewhere after the melting unit is a non-contacting thickness gage. When the strip is too thick or too thin to meet specifications, the gage actuates an electronic memory device which rejects those particularly heavy or light sheets at the second piler. Likewise, a photoelectric cell continuously scans the coated strip and causes sheets with perforations to be deflected into the first piler.

Trained inspectors are located at the pull-through bridle and the shear, respectively; they operate contact buttons which allow them to deflect at will any defective product they detect into the second or third piler (depending on product classification) and pass prime product into the fourth piler where the attendant sees that it is properly counted and that the shear operator is kept advised of the product flatness.

Laboratory determinations are made regularly to determine the tin coating and oil film weights on the finished product. Various laboratory controls of cleaning, pickling and plating solution characteristics are necessary.

SECTION 7

METALLURGICAL ASPECTS

General—A fuller appreciation of the metallurgical aspects of tin plate may be had by considering it as a nine-layer sandwich (Figure 37-10), the layers of which consist of:

1. Oil layer
2. Tin oxide
3. Free tin
4. Tin-iron alloy
5. Steel base
6. Tin-iron alloy
7. Free tin
8. Tin oxide
9. Oil layer

Adaptability for a specific purpose may depend on the properties of a given layer. To illustrate, for a pressed part, such as a toy, the drawing characteristics of the steel base may be the controlling factor. In the case of electrolytic tin plate, the chemical filming treatment gives an oxide layer found beneficial for adhesion of lacquers and enamels. Similarly, tin plate must have an oil film adequate both to promote good "feeding" in automatic equipment and to prevent scratching during fabrication and yet not so heavy as to cause difficulty during roller coating with lacquers or printing inks. Thus, each layer requires adequate control for some of the applications. Generally, since the manufacture of tin plate is on a mass production basis, it is most economical to standardize on the optimum quality of each layer for total tin mill production.

The Steel Base—The metallurgical controls of the steel

base are similar to those employed for the production of sheet and strip and, hence, will not be elaborated on further. It is sufficient to point out that mechanical properties of the finished tin plate depend on the composition of the steel, the heat treatments and the rolling opera-

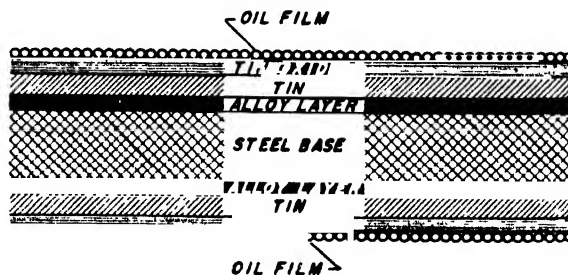
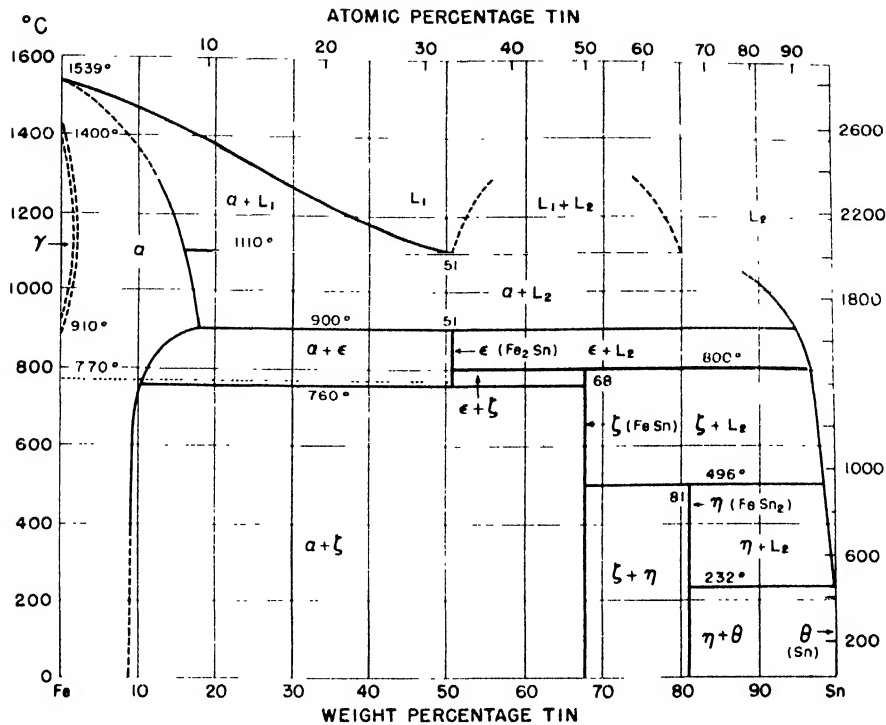


FIG. 37-10. Schematic enlarged cross-section of a sheet of tin plate, showing approximate relative thicknesses of the various "layers." The approximate thickness in inches of the individual layers is as follows:

Layer	Thickness (In.)
Oil film	10^{-7} (0.0000001)
Tin oxide	10^{-7} (0.0000001)
Tin	10^{-4} (0.0001)
Alloy layer	10^{-6} (0.00001)
Steel base	10^{-2} (0.01)



tions. Inasmuch as the steel used is a low-carbon mild steel and as the usual hardening agents are looked upon with disfavor because of their detrimental effect on corrosion resistance, as well as cost, strengthening by variations in composition is limited to addition or control with respect to phosphorus and nitrogen. Phosphorus has been used in amounts up to 0.15 per cent but its use is limited to tin plates employed for non-corrosive food and non-food containers.

Strength, or temper, is further controlled by regulating the amount of temper or "skin-pass" rolling. The tin

plate industry has adopted the Superficial Rockwell Hardness test as a control measure of the temper of the strip. This test is described elsewhere.

As the major portion of tin plate is used for containers, the requirements of this application dictate most of the metallurgical considerations in the processing of the steel base. As will be pointed out more fully in a later paragraph, the corrosion resistance of tin plate to food products is an important consideration—and in this respect the production of steel sheet for tin plate differs from the production of sheets for other applications.

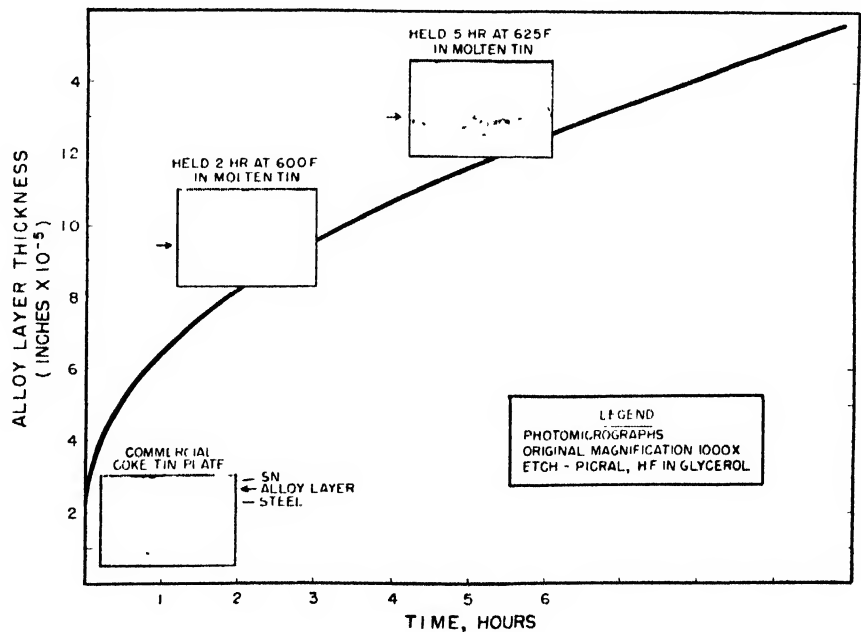


FIG. 37—12. Illustrating the rate of growth (increase in thickness) of the iron-tin alloy layer with increasing time of immersion of the steel base in molten tin at a temperature of 600° F.

Many factors affect the corrosion resistance of tin plate, but quality of the steel base is probably the most important. Besides the composition, the various processes, such as hot-strip rolling and the annealing after cold reduction, are subjected to even closer control than would be necessary from the standpoint of mechanical properties alone.

The various cleaning and pickling operations, previously described, are also closely controlled so as to prepare a suitable surface for subsequent coating with tin.

The Tin-Iron Alloy Layer—When a clean steel surface comes in contact with molten tin, a reaction takes place with the formation of a tin-iron alloy layer intimately bound to the steel surface. This alloy layer is quite thin (about 10 to 15 millionths of an inch), in coke tin plate. Chemical analyses have shown the ratio of tin and iron to correspond to the intermetallic compound FeSn_2 .

The metallurgy of the tin-iron alloy has received considerable attention by various investigators. The phase diagram for the iron-tin system in Figure 37—11 reveals that there are probably at least three compounds of tin and iron.

1. Fe_2Sn , referred to as the epsilon (ϵ) phase, which is stable between 1400°F (760°C) and 1652°F (900°C) but can, in the presence of sufficient tin, react at 1472°F (800°C) to form FeSn or zeta (ζ) phase.
2. FeSn , or zeta (ζ) phase, which is stable at all temperatures below 1472°F (800°C) but which reacts with excess tin below 925°F (496°C) to

form the compound FeSn_2 , known as the eta (η) phase.

3. FeSn_2 , or the eta (η) phase, which is stable below 925°F (496°C) and does not react with tin to form a higher tin compound.

Inasmuch as tinning operations are always carried out at temperatures considerably below 925°F (496°C), it follows that the alloy layer should contain both eta (FeSn_2) and zeta (FeSn) phases. The results of X-ray diffraction studies and chemical analyses indicate that the compound FeSn_2 predominates.

It has been demonstrated that the amount of tin-iron alloy formed is a function of time and temperature. The initial formation is extremely rapid and subsequent growth is much slower (Figure 37—12).

The thickness of alloy can be determined by electrostripping, by chemical stripping, by magnetic testing, or by metallographic cross-sectioning. The most commonly used is the electrochemical stripping method. A 4-square inch sample is attached to a suitable holder and immersed in a 1 N solution of hydrochloric acid containing a carbon electrode and a silver—silver-chloride reference electrode. The tin coating is stripped anodically at a constant predetermined stripping current. The potential difference between the specimen and the reference electrode is continuously measured by a potentiometer recorder during the stripping operation. The slope of this potential-time curve will change as the stripping proceeds from the surface, to the alloy layer, to the base

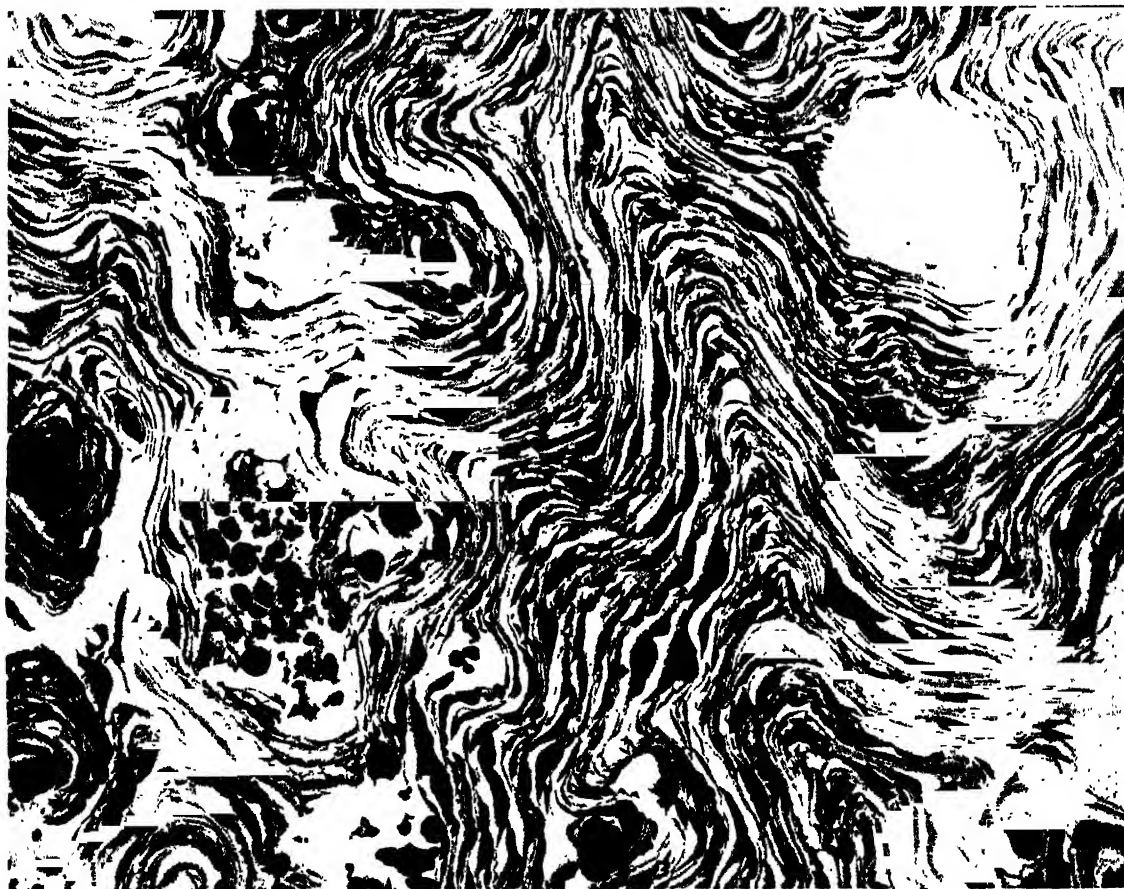


FIG. 37—13. Flux pattern on the surface of the steel base (interface of iron-tin alloy and steel) of commercial coke tin plate having a coating weight of 1.49 lb. per base box, after removal of both the tin and the iron-tin alloy. Magnification: 3X.

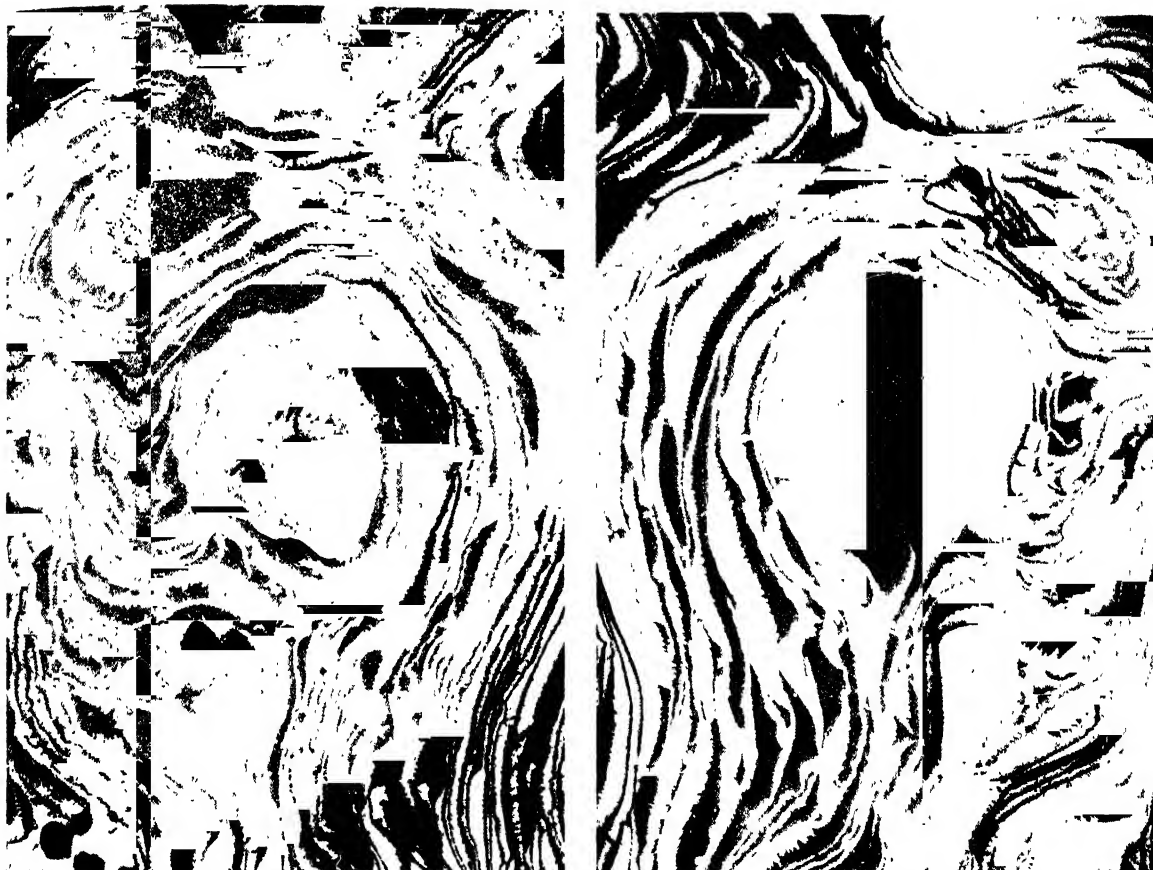


FIG. 37—14. Reversal of flux pattern in corresponding areas on opposite sides of a sample of commercial coke plate (coating weight, 1.32 lb. per base box), showing steel surface (interface between iron-tin alloy and steel) after the tin coating and iron-tin alloy were removed. Magnification: 5X.

metal. With proper cathode-anode geometry, the changes in slope occurring when the potential changes at the several interfaces are clearly discernible. The distances between the so-called end points, or points where the slope changes (measured parallel to the time axis), are measures of the thickness of the free tin and alloy layers.

The alloy layer on tin plate has a very definite pattern according to the manufacturing practice. Hot-dipped tin plate produced with the use of a zinc chloride flux has a distinctive mottle shown in Figures 37—13 and 37—14. This pattern is the result of variation in thickness and crystal size of the alloy, the dark areas being thinner and finer grained than the light areas (see Figures 37—15 and 37—16). Electrolytic tin plate (melted) has a smooth fine-grained alloy layer similar to the dark areas in coke tin plate alloy layers (Figure 37—17). Matte finish (unmelted) electrolytic tin plate does not have an alloy layer detectable by ordinary metallographic or chemical procedures. Heating matte finish plate at temperatures below the melting point of tin will result in formation of tin-iron alloy, the amount formed being dependent on the time and temperature.

The alloy layer is important with respect to tin-plate quality, especially in 0.25 lb. per base box electrolytic tin plate, since an increase of thickness of the alloy layer results in a decrease of thickness of the tin layer and may result in soldering difficulties.

Physically, the compound FeSn_2 is very hard and brittle and of itself cannot stand much bending. In com-

mercial tin plate the alloy layer is very thin and is covered with a much thicker layer of ductile tin so that even sharp bending does not expose much of the base steel because the tin will bridge gaps in the alloy layer. With thick layers of FeSn_2 , bending will cause breaks through the tin. The crystallites in thick layers of alloy are produced only with prolonged heating at temperatures above the melting point of tin (450°F).

FeSn_2 crystals are also formed in the tin stacks by reaction of the molten tin with the equipment. This accumulation is known as tin dross which is periodically cleaned out of the pots. The tin dross is heavier than the molten tin and settles to the bottom but, because of the agitation by sheets passing through, small particles may become imbedded in the tin coating, resulting in coating defects known as scruff.

The Tin Layer—As seen in the photomicrographs, the tin layer is considerably thicker than the alloy layer. It is customary to estimate the thickness of the tin layer in terms of weight of tin per unit of area, rather than in terms of thickness measurements. Thus, coke tin plate will have on the average of 1.0 to 1.5 lb. of tin per base box, equivalent to 1.0 to 1.5 lb. spread over both surfaces of 31,360 square inches of steel. From the specific gravity of tin it is readily calculated that 1.0 lb. per base box of tin represents an average thickness of 0.00006 inch on a surface. The actual thickness at specific areas may vary considerably from the average.

The weight of coating is usually estimated chemically by dissolving samples of known area in hydrochloric

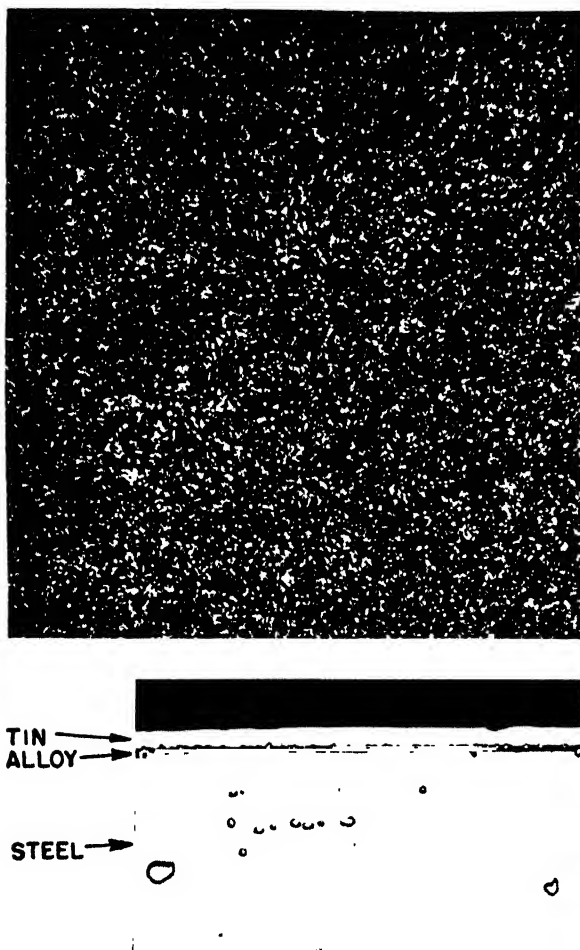


FIG. 37-15. (Above) Representative surface of iron-tin alloy layer on tin plate in dark areas, representing the interface between the iron-tin alloy and the tin of the coating, after removal of the tin portion of the coating. (Below) Representative cross-section through a dark area similar to that shown above, etched to show the tin coating and the iron-tin alloy layer. Magnification of both photomicrographs: 1000X.

acid under non-oxidizing conditions and titrating the stannous tin to stannic tin with standard iodine or iodate solutions. Analytical textbooks give details of the procedure. Gravimetric stripping methods are also used but to lesser extent. Recent testing methods incorporate electrolytic methods for dissolving the tin followed by standard volumetric titrations. The tin plate from each producing unit is sampled periodically for tin coating weight.

Electrolytic tin plates are produced to much closer tolerances with respect to tin coating weight than are possible for the hot-dipped tin plates. At present, commercial grades include 0.25, 0.50, 0.75, 1.00 and 1.00-0.25 lb. per base box and the spread in tin coating weight is usually within 10 per cent of the average. Electrolytic coatings are capable of fairly precise control, variations being due to fluctuations in current flow or variations in distance of the strip (cathode) from the individual tin anodes.

Tin plate usually has a bright, lustrous, mirror-like surface. The luster varies somewhat with the surface finish of the steel base. Rough base finish will result in

poorer luster than smooth base finish. The luster is also affected by the thickness of the tin coating. Heavy tin coatings, as on charcoal tin plates, have bright luster irrespective of base metal roughness. The smoothness of the tin deposit on electrolytic plate also affects the luster.

Mild etching of the tin coating reveals a spangle which may take a variety of shapes. This spangling is evident on the interior of all non-lacquered cans which have contained fruits or vegetables. The pattern can also be seen on unetched surfaces when viewed under polarized light. The size of the spangle is a function of the rate of quenching. Slow cooling produces a massive grain or spangle whereas rapid cooling, as by water quenching in the electrolytic plating lines, produces a very small spangle. This is illustrated in Figure 37-18. The size of the spangle has no bearing on the quality of the tin plate.

The tin coating, even when as heavy as 6 lb. per base box, is not continuous but has microscopic areas of exposed base metal. These areas of exposed base metal are known as pores. The porosity of tin plate has been the subject of considerable research and excellent methods for indicating the degree of porosity have been devel-



FIG. 37-16. (Above) Representative surface of iron-tin alloy layer on tin plate in light areas, representing the interface between the iron-tin alloy and the tin of the coating, after removal of the tin portion of the coating. (Below) Representative cross-section through a light area similar to that shown above, etched to show the tin coating and the iron-tin alloy layer. Magnification of both photomicrographs: 1000X.

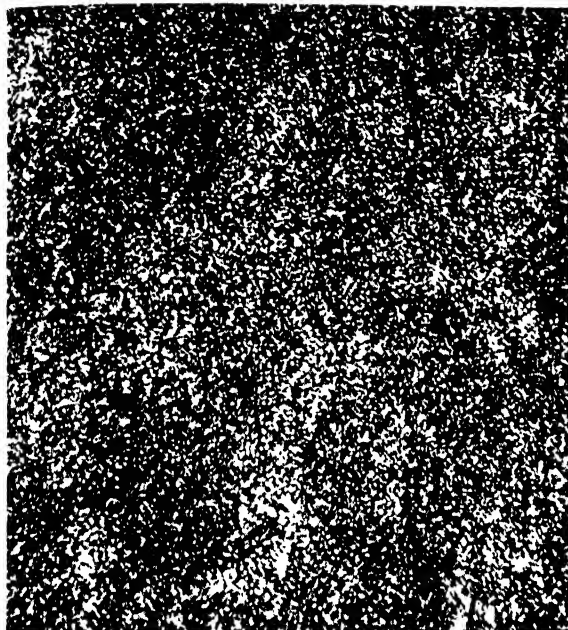
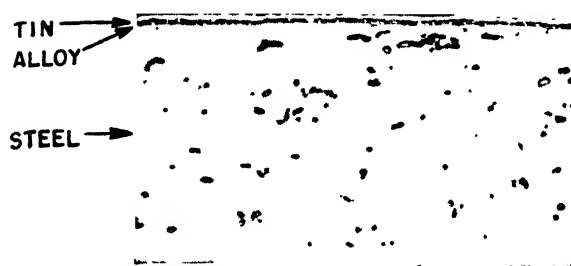
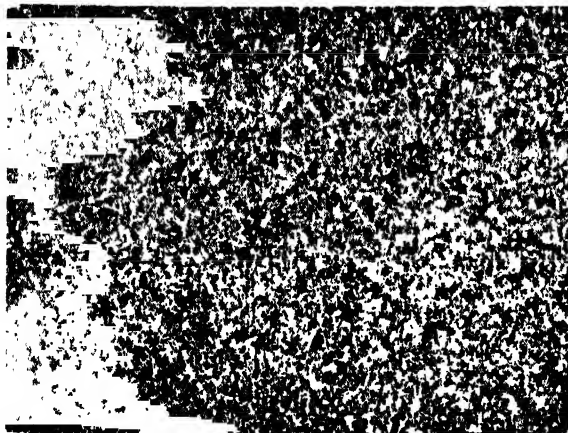


FIG. 37—17. (Above) Cross-section representative of commercial melted electrolytic tin plate (coating weight, 0.56 lb. per base box), etched to show iron-tin alloy layer. (Below) Unetched surface of iron-tin alloy layer (interface between iron-tin alloy layer and tin coating) after removal of the tin portion of the coating shown above. The surface of the iron-tin alloy has a pebbly appearance, indicating that it is not of uniform thickness. Magnification: 1000X for both photomicrographs.



oped: details of the procedures are summarized in International Tin Research Institute Bulletin, Series A, No. 7.

W. E. Hoare reports that the total area of iron exposed in pores is only about 0.7 sq. mm. per square meter of surface. The satisfactory commercial service of tin plate for fruit containers shows that despite these minute pores, the tin coating has excellent protective properties. This will be discussed more fully under resistance to corrosion.

The Tin Oxide Layer—An extremely thin oxide film is developed on tin plate immediately upon contact with the air as it emerges from the tin pot. At this stage the film is invisible. Depending upon the atmospheric conditions during storage this oxide film grows during long periods of storage. Experience has shown that areas of high humidity, such as adjacent to the Gulf of Mexico and on the West Coast, are the most critical in regard to expediting the natural growth of the tin oxide film. In the most severe cases the film becomes sufficiently thick to be visible. However, in the majority of instances the film is not visible until the baking operation which follows lacquering or lithographing. The visible films are very objectionable as they cause poor enamel adhesion and adversely affect solderability.

Just as temper colors are produced on steel surfaces by oxidation, so are varying colors developed on tin surfaces. Initially, tin surfaces show only their characteristic metallic luster, but as the thickness of the oxide film increases, interference colors develop, beginning with yellow. As oxidation proceeds, the color changes from yellow through brown and purple to blue. Under ordinary conditions, the various shades of yellow and brown are the only colors noted. Occasionally, under severe baking conditions, purple may be noted. When such colors appear on tin plate, it is said to be "discolored" and is likely to be unsatisfactory for some purposes. Inasmuch as the yellow color is most common, such films are generally known as "yellow discoloration" or "yellow stain." A further distinction is sometimes made, in that discoloration arising from baking operations is called "baking discoloration," whereas discoloration developed during storage is known as "warehouse discoloration."

Another objectionable type of surface film is the "sulphide discoloration" or "process discoloration" which develops on the interior surface of tin cans as a result of the action of sulphur-containing foods. This discoloration is evidenced by brown, purple, blue or black areas, and the color is often imparted to food in

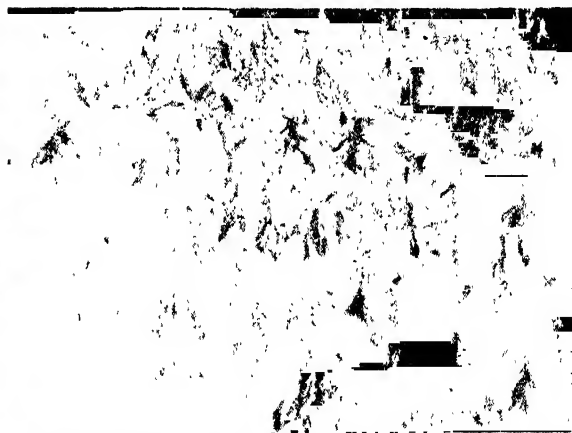


FIG. 37—18. Effect of cooling rate on the comparative size of spangles produced on tin plate. Small-grain spangle at left resulted from rapid cooling (water quenching); larger spangle at right resulted from cooling at a slower rate (quenching in oil).

contact with the discolored surfaces. In the severest cases of sulphide discoloration, there are formed gelatinous masses of yellow sulphide. Such masses often contain black iron sulphide formed by reaction of food sulphur with the steel base exposed at pin holes or dented areas. Sulphide discoloration can be prevented by coating the inside surface of the can with zinc oxide impregnated enamel.

Tin surfaces, along with surfaces of most common metals, begin to oxidize immediately upon exposure to air. Attempts have been made to generalize on the fact that the thin oxide films on some metals effectively inhibit further oxidation of the base metal while others do not. Pilling and Bedworth (see "Bibliography" at end of chapter) concluded with considerable accuracy that an oxide may be protective when the ratio of the specific volume of the oxide to the specific volume of the metal is greater than unity. In the case of tin, however, with an oxide-metal volume ratio of 1.34, the classification fails when the service requirements for the normal oxide film on tin plate are considered. It is possible, nonetheless, to treat tin surfaces in such a manner as to produce oxide films which effectively inhibit further growth of the oxide. Such films also provide considerable resistance to the growth of sulphide films.

The treatments for providing protective oxide films on tin plate may be divided into two classifications. First, there are the chemical treatments in which the plate to be treated is immersed in or sprayed with a treating solution. Such treatments generally require that the plate be in contact with the hot treating solution for periods of time in excess of five seconds. If only resistance to baking discoloration is needed, a short treatment is satisfactory. When the plate must resist yellow discoloration throughout long periods of storage or is expected to resist sulphide discoloration, treatments of 30 seconds or more may be required. The commercial solutions used for chemically treated tin plate contain chromic acid or an alkali salt of chromic acid. The chromic-acid treatment is the most common of the two and utilizes hot chromic acid at a concentration range of 0.50 to 1.00 per cent.

The second type of treatment makes use of electrochemical action. Cathodic treatments of one second or less in a dilute solution of sodium hydroxide, sodium carbonate, trisodium phosphate or the like usually offer only slightly improved resistance to yellow discoloration. Cathodic treatment followed by anodic treatment in the same solutions provides further slight protection against yellow discoloration, but greatly enhances resistance to sulphide discoloration. Cathodic treatment in an acid or alkaline solution of a chromate produces a thin oxide film which is very resistant to baking discoloration but which may lack adequate resistance to warehouse discoloration and sulphide discoloration. Cathodic-anodic treatment in a chromate solution permits the formation of films which are very resistant to all types of discoloration and requires a treating time of less than one second for each step. The anodic portion of this

last treatment must be conducted at a low anodic current density, if good lacquer adherence is to be obtained.

The Oil Film—It has been found necessary for tin plate to have a thin film of oil to permit feeding of sheets to fabricating equipment and to prevent scratching and abrasion during fabrication on automatic equipment. Too much oil, however, causes trouble if the tin plate is to be lacquered, because globules of oil immiscible in the lacquer film produce thin spots and uncoated areas. The oil, of course, must be edible since it comes in contact with food products. Palm or cottonseed oils are normally used, but recent trends are toward the use of synthetic oils such as dioctyl sebacate. Satisfactory oil films usually are about 0.10 to 0.30 gram per base box in weight. This means an oil film less than ten molecules in thickness. To illustrate the minute thickness of this oil film, imagine spreading one teaspoon of salad oil uniformly over a wall 10 feet high and 2000 feet long. That is the equivalent of 0.10 gram per base box.

In the hot-dipping method, the tin plate is initially covered with oil which is later mostly removed in the wet washers. The final oil film thickness on the sheets is controlled by the branner. The oil content of the wheat bran used as a polishing medium is maintained at the desired level by periodic addition of "oil mids," which is bran to which has been added fresh palm or cottonseed oil. In some mills the oil content of the bran is periodically determined by extraction with solvents and weighing the oil residue. The oil content of the bran required to deposit the required oil film on tin plate varies with the speed of tinning, as well as with the efficiency of cleaning of the wet cleaners.

Application of Oil Films to Electrolytic Tin Plate—Electrolytic tin plate does not require cleaning as does hot-dipped tin plate but it has been found necessary to apply an oil film. Several methods have been tried commercially, including:

1. Branning machines
2. Spray oiling
3. Emulsion oiling
4. Electrostatic oiling

The first two methods have been largely supplanted by the emulsion oiling method and the electrostatic oiling method. Conventional branning machines did not operate satisfactorily with the high line speeds in use. Emulsion and electrostatic oiling were discussed earlier, but will be briefly described here.

In the emulsion oiling process, an acid-stable emulsion of either palm or cottonseed oil stabilized with an amine is sprayed onto the moving strip. Squeegee rolls spread the emulsion over the tin plate strip. The oil is deposited on the tin plate and the water phase is removed by the squeegee rolls.

The electrostatic method utilizes a high electrical potential between a fixed electrode and the moving strip, which creates a powerful electrostatic field around the strip. Vapors of oil are allowed to rise into this field and they are deflected onto the strip by proper adjustment of strip polarity.

SECTION 8

CORROSION RESISTANCE

To a large degree the successful application of tin plate for containers depends on its corrosion resistance. Food products react with the tin coating and the base metal but the rate of reaction is generally sufficiently slow to permit fairly long storage or shelf life. The type of corrosion that takes place depends on the nature of the

corrosive media. Thus, the corrosion problem can be divided into:

- A. Atmospheric Corrosion—rusting.
- B. Discoloration of the Interior of the Container, as by etching due to food acids, or blackening by food sulphides.

- C. **Hydrogen-producing corrosion** by contents of cans giving rise to "swells" or hydrogen springers in which the pressure of the evolved gas bulges the ends of the cans, thereby making them unmerchantable. In some cases the cans may perforate.

Atmospheric Corrosion—Tin plate is very durable in dry air but all tin plate will rust eventually in the atmosphere, especially when moisture is present. The amount of external rusting depends, in large measure, on the porosity of the tin coating and the resulting area of steel base exposed. Under conditions involving moisture and oxygen, the exposed iron behaves as an anode and the tin as a cathode (see Chapter 35, "Theory of Corrosion"). Increasing the thickness of the tin coating reduces the rusting, by reducing porosity. Rust resistance of tin plate can be somewhat controlled also by the nature of the tin oxide film. Treatment of tin plate in solutions of chromates and phosphates have been shown to minimize rusting but to date such treatments have not been developed sufficiently to permit commercial usage, at least in the United States. The composition of the steel base also has some bearing on rust resistance under certain conditions. Copper-steel base tin plate (low metalloid) showed less tendency to rust under certain conditions. Silicon above 0.02 per cent in the steel has been shown to make tin plate very susceptible to rusting.

Hartwell has shown that silicon causes the formation of small cracks in the surface of the tin plate when it is fabricated into cans (see second Hartwell reference in "Bibliography" at end of chapter).

Discoloration of the Interior of Cans—When foods rich in sulphur-containing proteins are packed in plain tin cans (that is, without lacquer), it is generally noted that the inside of the can is stained purple, brown or black. This stain is tin sulphide and is in no way harmful. The usual method of preventing this type of corrosion is by lacquering, especially with zinc oxide impregnated lacquers. The sulphur reacts with the zinc oxide to form zinc sulphide which is white and not noticed in the can. Sulphide blackening may also be prevented by a surface filming treatment as mentioned under the section discussing the tin oxide layer in an earlier part of this chapter.

In some mildly corrosive foods such as evaporated milk, localized detinning or dark staining may occur if the tin-oxide film is not soluble in the food product. For evaporated milk cans, some users prefer electrolytic tin plate either with no chemical treatment (NCT) or plate filmed cathodically.

Non-chemically treated plate has limited storage stability because of susceptibility to form yellow stains on aging. Cathodically treated plate has better storage stability.

The etching of the tin plate by food products is no indication of spoilage. All food products react with tin, some very slowly and others more rapidly. Spinach and other greens, rhubarb and squash are rapid de-tinners. These products are frequently packed in lacquered containers.

Hydrogen-Producing Corrosion—The usefulness of tin plate for food-container manufacture depends largely on its resistance toward the formation of hydrogen "swells" or perforations.

The corrosion of tin plate containers by food products is a complex problem and the state of our knowledge is still largely empirical. There are many factors affecting the rate at which tin plate corrodes. Major factors include:

1. Type of food product.
2. Packing procedure used in canning.
 - a. The initial vacuum.
 - b. The headspace volume.
 - c. Use of inhibitors.
3. The storage conditions.

1. **Food Product**—Food can be roughly divided into three classes with respect to corrosiveness:

Most Corrosive—This group includes the highly colored fruits and berries. They are generally packed in lacquered containers because tin has a reducing action on the anthocyanin pigments which results in bleaching of the color. The acidity and pH (intensity of acidity) of the food product is no criterion of its corrosivity. In fact, corrosivity can often be reduced by adding an organic acid, as by adding lemon juice to dried prunes in syrup.

Mildly Corrosive—This group includes the bland fruits such as peaches, pears, apricots and the citrus juices from grapefruit and oranges. The mildly corrosive fruits are generally packed in plain (not lacquered) tin cans.

Slightly Corrosive—This group includes the vegetables and meat products which normally do not produce hydrogen springers in the time required to merchandise the products.

2. **Food Packaging Procedure**—The canning procedure is carefully controlled to provide maximum shelf life. Cans are sealed under a high vacuum to assure maximum exclusion of oxygen because the presence of oxygen within the can markedly accelerates corrosion. A high vacuum is obtained either thermally or mechanically. In the former procedure the cans are sealed at as high a temperature as feasible, normally above 165° F. At this temperature, water has an appreciable vapor pressure so that the atmosphere immediately above the liquid level of the can is largely water vapor which displaces the air that would normally be present. Upon cooling the can, the contents shrink, increasing the headspace volume and thereby producing the vacuum within the can. Mechanical methods of vacuumizing involve evacuating the chamber of the sealing machine in which the cans are sealed by a vacuum pump or sucking the air out of the headspace of the filled can with a jet of steam passing over the end of the container at the moment of attaching the lid. Mechanical vacuumizing eliminates preheating of the contents.

The headspace volume also affects the rate of corrosion by acting as a hydrogen reservoir. Overfilled cans require much less hydrogen to dissipate the vacuum and build up a pressure within the can than do cans with normal headspace. Slack filled cans, however, are prohibited by Federal statutes.

Sulphur in very small amounts markedly accelerates corrosion of tin cans by certain products. Spray residue on the fruit or contaminated sugar have been found to be the source of sulphur responsible for rapid corrosion failures in some instances.

3. **Storage Conditions**—The temperature of storage of packed cans profoundly affects the corrosion rate as, indeed, temperature affects the rate of most chemical reactions. The higher the storage temperature the shorter is the pack life. Cans of fruit which would normally last four years at 70° F might last only a year or less at 100° F sustained storage.

Characteristics of Tin Plate Affecting Its Corrosion—The resistance of tin plate to corrosion by food varies considerably according to the methods of manufacture.

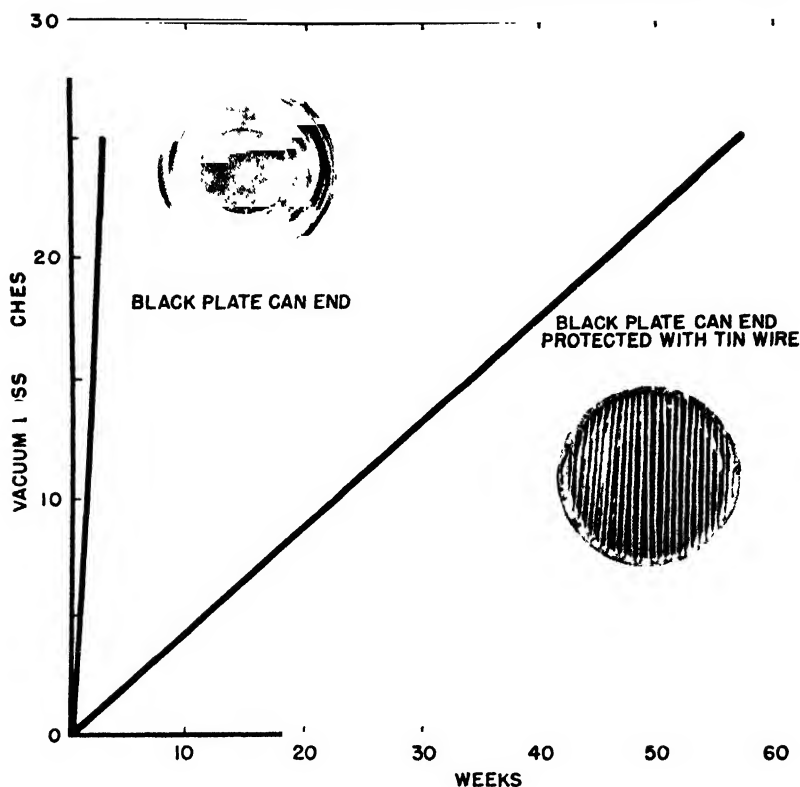


FIG. 37-19. Chart comparing results of prune-pack test on (upper left) can with black-plate end, and (lower right) similar can with black-plate end protected with tin wire, illustrating cathodic protection of steel by tin under the conditions of the test.

Extensive study by numerous investigators has helped to clarify many misconceptions as to the nature of the corrosive attack. Fundamentally, the corrosion of a tin can is the corrosion of the tin-iron couple. Tin of itself is quite corrosion resistant in weak organic acids in the absence of oxygen, either gaseous or in the form of oxidizing substances. Iron, on the other hand, is much more readily corroded by weak organic acids even in the absence of oxygen. Experiments in which tin is coupled to iron and immersed in weak acids in the absence of oxygen reveal that the corrosion of the iron is inhibited whereas the corrosion of the tin is accelerated. The tin protects the iron under such conditions. In the presence of oxygen or oxidizing agents the cell may be reversed and the corrosion of the iron is accelerated.

It has been shown previously that the tin coating on tin plate is not continuous but contains numerous pores. In the region of these pores, local electrolytic cells are formed when the tin plate is in contact with an electrolyte. Depending on the electrolyte and the environment, either the exposed iron in the pores or the surrounding tin may behave as the anode. With normal conditions existing in a can, closed so as to have a minimum of oxygen, the tin generally behaves as an anode, protecting the exposed iron. Were this not so, cans would perforate very rapidly at the exposed iron areas. Actually, tin cans have a remarkably long shelf life.

An interesting experiment to illustrate the protective efficiency of tin consists of fabricating cans with the body and cover lacquered and the bottom end made of black plate. Such a can packed with dried prunes in syrup, sealed hot to eliminate oxygen, will withstand storage at 100° F for about four weeks before bulging due to hydrogen evolution. A similar can, to which a

pure tin wire grid so spaced as to have $\frac{1}{8}$ inch spacing is tack-tinned to the black plate bottom, will withstand up to sixty weeks at 100° F before bulging (see Figure 37-19).

The degree of protection given by tin varies with the nature of the steel base and also with the nature of the food product.

Characteristics of the Steel Base Affecting the Corrosion of Tin Plate—The composition and methods of manufacture of the steel base are probably more important in controlling the corrosion resistance of the resulting tin plate than any other factors.

Hartwell, by means of experimental packs of corrosive fruits in cans from many lots of tin plate, has shown that phosphorus, copper and silicon may have pronounced effects on corrosion resistance, whereas, within the limits usually found in tin plate steel, the effects of carbon and sulphur are not of commercial significance. Phosphorus and copper have a profound effect on the service life of enameled (lacquered) cans and silicon is more effective with plain cans. Increasing amounts of phosphorus and silicon are detrimental to the service life of containers, while copper increases or decreases the corrosion resistance, depending on the product packed. The detrimental effects of copper usually appear more serious than the benefits derived from its presence. Therefore, tin plate made from a steel base low in phosphorus (0.015 per cent maximum), copper (0.06 per cent maximum) and silicon (residual) is desirable for the strongly corrosive products.

Container manufacturers order tin plate according to the type of food product packed. The following grades are in general use:

Type L—Cold-reduced open-hearth steel containing:

Element	Per Cent (Max.)
Carbon	0.12
Manganese	0.60
Phosphorus	0.015
Sulphur	0.050
Silicon	0.010
Copper	0.06
Chromium	0.06
Nickel	0.04
Molybdenum	0.05

This is used for the most corrosive foods.

Type MR—Cold-reduced open-hearth steel similar to Type L except that the phosphorus limit is increased to 0.020 per cent, the copper limit is increased to 0.20 per cent and no tolerances are specified for nickel, chromium and molybdenum. This grade is used for mildly corrosive products.

Type MC—Cold-reduced open-hearth or Bessemer steel similar to Type MR except that phosphorus limits are increased to between 0.04 and 0.15 per cent. This grade is used for noncorrosive products where strength of the container is the main consideration.

There is increasing evidence that the manner of preparation of the steel strip for tinning may be as significant in controlling ultimate corrosion resistance as composition of the steel. Fairly rapid quenching after hot reduction and thorough annealing in dry, low-carbon and low-oxygen content atmosphere favor the highest corrosion resistance.

In a tin can, the hydrogen evolved during pack storage results primarily from corrosion of the iron and not from the corrosion of tin. Analyses of various food products stored in plain unlacquered cans show up to 120 parts per million of tin after about a year's storage at room temperature. Tin pickup from lacquered cans is very much lower, averaging 10 to 30 parts per million, according to Hirst and Adam (see "Bibliography" at end of chapter). The iron content of cans which have a good vacuum even after prolonged storage periods is invariably low, less than 20 parts per million. In springers or hydrogen swells the iron content invariably approaches or even surpasses the tin content.

Controlled experiments by a United States Steel Corporation research laboratory on rate of tin and iron pickup by dried prunes in water in plain cans have shown that at constant temperature the tin is corroded at a constant rate, or linearly with time. The rate of iron pickup, however, is not constant. During the early stages of storage there is little or no corrosion of the iron. After this induction period the iron goes into solution at an increased rate and the can becomes a springer. The vacuum-loss curve of the can is similar in shape and displacement to the iron pick-up curve. The length of the induction period varies with the corrosion resistance of the tin plate. Fast-corroding tin plates have a short induction period while slow-corroding tin plates have a long induction period.

In essence, the metallurgical problem of improving corrosion resistance of tin plate resolves itself into methods of slowing down the attack on the steel base. This can be accomplished by:

1. Improved quality of steel base.
2. Heavier tin coatings.
3. Coating with proper type of lacquer in certain cases.

Effect of Tin Coating Thickness on Corrosion of Tin Plate—Increasing tin coating weights results in increased corrosion resistance in plain cans but not necessarily in lacquered cans. The corrosion of lacquered

containers will be discussed in more detail later. The relationship between pack life and tin coating weight of hot-dipped tin plate has been difficult to establish quantitatively because of the variation in coating as well as the variation in behavior of different portions of the same sheet. With the commercial development of electrolytic tinning, it was possible to prepare test lots of accurately controlled tin coatings in various coating weights on adjacent portions of the same coil. Fruit pack tests (dried prunes in syrup) on such specially prepared lots show that above a certain minimum coating weight an increase in tin coating linearly increases the corrosion resistance within the range tested.

The slopes of the straight portions of the curves obtained for a large number of steels plated from different electrolytes are essentially equivalent. Similar curves for hot-dipped tin plate have been obtained by electrolytically stripping areas to reduce the relatively thick hot-dipped coating to predetermined coating thicknesses comparable to those on electrolytic tin plate.

Micro variations in tin coating thickness apparently do not affect pack life of plain cans packed with fruits. This could be expected from the cathodic protection afforded by tin to the steel.

Lacquered Tin Plate—Lacquered cans are normally fabricated from tin plate which has been lacquered in sheet form. In the United States the trade refers to the lacquers as enamels. The lacquers vary in composition but are either organic solutions of resins or mixtures of resins and vegetable oils, the resins being either naturally occurring or synthetic. They may be clear or pigmented.

Lacquered cans are normally used to prevent the bleaching of highly colored fruits or to prevent the sulphide staining caused by sulphur-bearing foods. Considerable success in the reduction of hydrogen swells has, however, attended the use of lacquered cans for certain food products. During World War II, considerable quantities of 0.50 lb. per base box electrolytic tin plate, and even black plate, protected by lacquered coats, gave satisfactory service life for meat and vegetable products. The use of lacquered 25 plate has increased substantially, so that currently it accounts for a major portion of cans.

The corrosion process in lacquered cans is considerably more complex than in plain cans. The type of lacquer and methods of application may be as important as the quality of the tin plate used. Generally, improved service life may be obtained by more continuous coverage, as by spray or flush lacquering the fabricated cans.

Cracks in the lacquer coating expose minute areas of tin and iron. At these areas the tin tends to protect the iron but, because of the limited amount of tin exposed, it is soon used up and corrosion of the tin plate may be found in areas which have no apparent discontinuities in the lacquer film. Indeed, lacquered cans of fruit, which have abnormally long service life, frequently show pronounced etching of the tin beneath the lacquer indicating that lacquer films may be acting as semiporous membranes and the corroding tin beneath the lacquer is protecting the iron cathodically.

Trial packs are used to determine whether pack life can be improved by the use of a specific lacquer on the tin plate. As yet there are no laboratory tests which will determine the suitability of a specific lacquer system for a specific food product.

Bibliography

Adam, W. B. and D. Dickinson, The substitution of black-plate for tinplate in cans for fruit and vegetables. *Iron and Steel Institute Journal* 152: 195-205

(continued on next page)

Chapter 38

LONG TERNE SHEETS AND TERNE PLATE

Long Terne Sheets—Long terne sheets, or long ternes, are steel sheets that have been coated by immersion in a bath of terne metal, which is an alloy of lead and tin. Sheets coated with terne metal are duller in appearance than sheets coated with tin alone, and it is this feature of the product that gave rise to the name "terne"—meaning "dull." Long terne sheets fall within the dimensional ranges covered by the uncoated sheet commodities. In commercial practice, they seldom are manufactured thinner than 0.014 inch, thicker than 0.125 inch, or wider than 48 inches. Long ternes occupy a relatively minor position tonnage-wise in the sheet business, but offer a combination of properties that make them very suitable for such applications as air cleaners, fuel tanks and burial caskets.

Long Terne Coatings—The composition and weight of long terne coatings were drastically restricted in 1941, when the normal supplies of tin were cut off by World War II. Prior to that period, it was customary to coat with terne metal containing approximately 80 per cent lead and 20 per cent tin, although this composition was not mandatory and could be varied for specific applications. Since the object of terne coating is to apply to the steel base an inexpensive, corrosion-resistant coating of lead, the percentage of tin used need only be sufficient to obtain a smooth, continuous coating. Lead alone does not alloy with iron, so that it is necessary to incorporate a certain amount of another element, in this case tin, which alloys readily with the steel base and forms a solid solution with the lead. It was formerly thought impractical to coat with terne metal compositions containing much less than 15 per cent tin but experience gained during the war period proved otherwise, for during this time, when government restrictions limited the amount of tin in long terne metal to 10 per cent maximum, terne coatings of entirely satisfactory quality were produced. It is to be noted, however, that decreasing the amount of tin in the coating alloy necessitates higher pot temperatures and lessens the alloying or "wetting" properties of the terne metal.

Prior to 1946, the coating weights of long terne sheets were designated by pounds per double base box, a term used in the tin plate and short terne industry, and never logically applicable to long ternes. With the exception of roofing long terne sheets, which are still produced to coating weights of 20, 30, or 40 pounds per double base box in specific gages and sizes, this designation is no longer used. Long terne sheets are now ordered to a coating weight specified in ounces per square foot, with weights ranging from "commercial" (no minimum amount) to 0.55 ounces per square foot. The designation of terne coating weights in terms of ounces per square foot follows the practice used in designating coating weights of galvanized sheet, as long ternes are commonly made on equipment located in galvanizing shops. Table 38—I lists the various weights of coating commonly available and the applicable sheet gage numbers. Commercial coating is furnished unless a special coating

weight is ordered. Long terne sheets with commercial coating are well-coated sheets but are not subject to any minimum coating test.

The thickness and nature of the coating are the most important factors governing the corrosion resistance of long terne sheets. Both lead and tin are highly corrosion resistant, and their combinations as used on long terne coatings also resist corrosion well. However, both lead and tin are cathodic to iron under most conditions of exposure, so that terne metal will actually accelerate corrosive action on the steel base when any portion of the steel surface becomes exposed to attack. Since a light coating has a greater tendency to form pinholes or other discontinuities than does a heavier coating, exposure of the base metal is minimized and corrosion resistance is increased as the thickness of the coating increases. However, even the most heavily coated ternes should be well-painted when maximum protection against corrosion is desired.

Table 38-I. Ordered Coating and Minimum Coating Weight Limits for Long Terne Sheets.

Ordered Coating	Weight (Oz. Per Sq. Ft.)		Applicable Sheet Gage Numbers	Nominal Coating, Lb. Per Double Base Box
	Minimum Coating By Sheet Weight Test	By Triple Spot Test		
Commercial	No Min.	No Min.	10 to 30	Commercial
0.35	0.30	0.26	16 to 30	9.00
0.45	0.38	0.34	18 to 30	12.00
0.55	0.47	0.41	20 to 30	15.00

Composition and Preparation of Steel Base—The steel base of long ternes may be provided on either a "commercial quality" or a "drawing quality" basis. In the former case, rimmed, capped, or semi-killed steel is generally used; in the latter, specially selected rimmed or aluminum-killed steels are provided, because of their superior ductility. All of these steels usually fall within the following composition limits:

Carbon (%)	Manganese (%)	Phosphorus (%)	Sulphur (%)	Silicon (%)
0.03-0.10	0.25-0.50	0.025 max.	0.050 max.	0.020 max.

Selection of the composition and treatment of the steel to be used for the base of long terne sheets is governed by the end use of the products to be fabricated from them and hence must take into consideration a number of factors, important among which are the customers' fabricating practices. Factors normally considered in

addition to the usual requirements for flatness, finish, and weight of coating, are such forming hazards as breakage in deep drawing operations, fluting, stretcher strains, and the conditions under which the material is to be used. In this latter connection, for example, a copper content of 0.20 per cent minimum is usually recommended for applications involving corrosion hazards, such as may be encountered when used for roofing or for burial caskets.

The treatment accorded the steel base used in manufacture of long terne sheets, depends upon the desired combination of gage, surface finish, and physical properties. It is basically identical with the treatment which would be given a cold rolled sheet or a hot rolled sheet manufactured for the same type of customers' fabricating operations. Product 18 gage and lighter generally is rolled on cold reduction mills while 16 gage and heavier usually is rolled on a continuous hot strip mill; some overlapping in method of rolling occurs, however, in the range 18-14 gage.

The continuous cold reduction process involves the use of lubricating solutions containing oil, the removal of which from the surface of the steel prior to annealing is of vital importance in producing a base metal which is to be terne coated. Any oil which remains upon the sheet or strip will decompose when heated to the annealing temperature. The relatively inert, carbonaceous residue which results can interfere seriously with the application of a smooth, continuous coating of terne metal, since these residues are not attacked to any appreciable degree by the pickling solutions commonly employed for descaling and cleaning the steel prior to terne coating. As a result, they prevent the terne metal from "wetting" or alloying with the steel base, causing rough coating or entirely bare areas where the film is particularly heavy. Some manufacturers make use of an electrolytic cleaning operation prior to annealing in order to prevent the formation of the "carbon smudge"; others cold reduce the base metal in thin mixtures of volatile water-soluble oil.

Cold reduced product is usually box annealed since it is more economical than normalizing and the sheets or coils may be kept scale-free by using a deoxidizing atmosphere, thereby reducing the iron loss due to pickling. The temper rolling operation may be eliminated on certain gages and applications wherein it is known that the customer's fabrication involves a deep-drawing operation in which fluting or straining of the base metal is not objectionable. Final preparation of the base metal prior to coating may involve shearing coil product to sheet form on a flying shear unit, with or without surface inspection, and in these operations it is important that the sheets be kept free of oil or grease, for the reasons previously stated.

When the base metal is rolled to gage on a continuous hot strip mill, the hot rolled coil is sheared to sheet form before any further processing takes place. Sheets intended for deep-drawing and stamping generally are normalized, while material for less severe applications may receive no more than a flattening operation after shearing, because in these heavier gages the steel strip is sufficiently soft for moderate forming operations.

The pickling practice prior to coating is essentially the same as that used in the production of galvanized sheets and the operation is performed in the same shop or department. Hydrogen embrittlement of the base metal, especially cold reduced product, is an ever-present hazard and close control of the pickling operation is required to avoid deleterious effects.

Long Terne Coating Application—There are two methods generally used for applying terne metal in the

manufacture of long ternes. The chief objective of both of these methods is to bring the properly prepared surface of the steel base into contact with the molten terne metal, and to remove the coated sheets from the pot, under conditions that will assure a uniform distribution of the coating. In both methods the aim is to produce a continuous coating of terne metal as free of discontinuities as possible.

The flux process, as the name indicates, employs a flux of molten (or a water solution of) zinc chloride, or a solution of zinc chloride in hydrochloric acid, to remove any oxides of iron that may be present and also effect a rapid drying of the sheets. This latter feature makes it possible to use a coating machine similar to that used in the tin plate industry. In fact, the flux process is quite similar to the methods used in tinning, with the exception that the temperature of the terne pot is higher than that of the tin pot because of the higher melting points of the terne metal compositions. The terne pot temperatures commonly employed range from 620° to 680° F. The process is carried out in a "rigging" or machine which carries the sheet through the several steps of the process as follows: The pickled sheets are fed from water boshes or piles into rubber pinch rolls, which conduct them through a hydrochloric acid wash tank. The object of this acid dip is to remove the iron hydroxide which may have formed on the sheets subsequent to pickling. Another set of rubber pinch rolls squeezes excess acid from the sheet, and the entry rolls of the coating rigging carry the sheet into the flux box, where it is cleaned and dried. The sheet then passes downward through the molten terne metal, where the coating is applied, then upward through a bath of oil (palm oil, fish oils, mineral oils, or combinations thereof) floating on top of the metal, being conveyed through the oil by the oil rolls. The oil bath on the exit side of the metal bath, in addition to protecting the surface of the molten metal bath from oxidation, tends to keep the coating on the sheet in a molten condition and thus allows the oil rolls to control the distribution and thickness of the coating. As the coated sheets leave the oil bath they are conducted to the conveying system of the cooling and cleaning train.

In the combination process, a coating machine using the flux process is followed by another machine of the same type but with oil substituted for the zinc chloride. This type of arrangement permits complete control over the coating conditions within the entire range of coating weights commonly available. Since most long ternes are ordered in the lighter coating weights, the bulk of the material produced is coated on a single unit by the flux process.

After the coating operation, the freshly coated sheets must be cooled sufficiently to permit cleaning without injury to the soft coating. Terne plate, because of a heavier steel base and higher temperature of the coating bath, requires more time than tin plate to cool to the proper temperature for cleaning, therefore special equipment is needed. In general, this equipment consists of a hooded conveyor against which blasts of cool air are directed from a distributor pipe. Control of the volume of air through the distributor main and branch tubes is effected by varying the speed of the blower and regulating the flow through the individual tubes by a system of dampers. The conveyor system carries the sheets in a horizontal position with both surfaces exposed to the cooling action of the air blast. This conveyor consists of a series of deeply hollowed rolls, sometimes referred to as double-cone sets, because each roll is so deeply concave that it resembles two cones mounted on a shaft with their apexes abutting. When a

sheet passes over the conveyor only its edges are in contact with the rolls; and with the air distributors suitably arranged, both surfaces are equally exposed to the cooling action of the air blast.

Regardless of the coating process used, the terne plate emerges from the coating operation covered with oil. Part of this oil drips from the sheets as they pass over the cooling conveyor and is collected in a grease pan located under the conveyor, but the remainder must be removed with special cleaning equipment. Long terne sheets are produced with a "dry" or "bright" finish, and an "oil" finish. Sheets leaving the cleaning equipment practically free of oil are designated as dry finish, but if they carry an appreciable amount of oil, or are subsequently coated with oil, they are designated as oil finish.

The residual oil present on the surfaces of long terne sheets after coating is usually removed by a branner similar to that used on hot-dipped tin plate. Contrary to tin plate practice, however, the long terne branner is preceded by a pair of cloth squeezer rolls, which remove much of the heavy oil film before the sheet enters the regular branning operation. Middlings or sawdust may be used in the branner. Steel rolls thinly wound with flannel may be used at the exit end of the branner to exert a smoothing action on the surface when a bright finish is desired. For best results in cleaning, two branners are normally employed, one following the other. When oil finish is specified, oiling rolls may be inserted in the line to apply the desired amount of oil on the surface, or it may be accomplished as a separate operation in a regular oiling machine. Upon completion of cleaning, the sheets are ready for inspection by the inspectors stationed at the end of the line.

Inspection and Testing of Long Terne Sheets—The inspection of long terne sheets normally takes place when they have completed their travel along the conveyor system, through the branners, and are being removed for piling into lifts. All but a small percentage of the sheets coated are primes, i.e., sheets free from any injurious defects recognized as such in the trade. However, even with the best practice and control of processing variables, a certain percentage of the sheets from the terne pots have defects which are of such a nature or magnitude that they are classified as rejects. **Recoats** are sheets containing defective areas which may be reconditioned to prime grade by rerunning them through the terne pot a second time. **Scrap sheets**, which are those totally unsuitable for reconditioning or salvaging, are normally the result of cobbles in the rigging of the terne pot itself or in the branners. The terne coating from scrap sheets is reclaimed by placing them in palm oil, heating to the proper temperature, and allowing sufficient time for most of the coating to run off and collect in the bottom of the container.

The defects of long terne sheets may be classified into three general types, depending on the basic origin of the defect. These types are as follows:

- (1) Those due to an imperfection in the steel base.
- (2) Those due to faulty processing of the material prior to the coating operation.
- (3) Those due to faulty coating practice.

The major defects encountered in category (1) are blisters and seams, the latter sometimes being designated as "skin laminations" or "slivers." As in galvanized sheet production, the blister defect occurs most frequently. The basic causes and the method of formation of blisters and seams, as discussed in the chapter on galvanized sheets, apply also in the case of long terne sheets.

Defects which may be classified in category (2) include rolled-in scale, pits, gouges, scratches, under-pickled areas, and other imperfections of a minor nature which may be eliminated by making corrections in preceding processing operations. Another defect falling into this category is the previously mentioned "rough coating," which results when the terne metal fails to "wet" or alloy with the steel base because of a film of reduced iron or carbonaceous residue which has not been satisfactorily removed in pickling. In this connection, it is to be noted that the cleanliness of the base metal is much more critical in the coating of long ternes than in the case of hot-dip zinc coatings.

Defects comprehended in category (3) include such items as bent corners, uncoated areas, oil stains, branner marks, and other minor items which may be controlled by the pot operators. In addition to the inspection for the above enumerated defects, the sheets are also checked for gage, size and flatness by the pot inspectors and are stamped with the appropriate identification marks.

The tests to which long terne sheets are subjected after completion of the coating operation may be classed as chemical, microscopic and mechanical. It is also necessary to have some type of immediate control test to determine the amount of coating being applied to each lot as it is being coated, and the sheet weight test is used for this purpose. The sheet weight test is made on a lot of from one to ten sheets, depending on the gage. The test lots are weighed after the pickling, washing and drying operations, and then weighed again after coating. The weight of coating in ounces per square foot is calculated by dividing the difference in weight so obtained by the total area of the sheets comprising the test lot. This type of test is conducted frequently for each coating unit so that an adequate control of the coating weight may be maintained and adjustments made to the coating machine as the occasion arises. For accurate determination of the weight of terne coating on the sheet and to determine conformance to specifications it is customary to use the **triple spot test method**. Where the product is coated in sheet form, three specimens exactly 2.25 inches square, or discs of equivalent area, are cut from each test sheet, one being cut from the center and the other two from diagonally opposite corners. The end spot tests are cut not closer than 4 inches from the ends and 2 inches from the sides, in order to avoid the heavier coating on these areas. In testing product made by the continuous-strip coating process, since the coating-weight difference for the back and front of the test sheet do not exist, it is acceptable to take all three spot tests across the width of the sheet. After each specimen has been weighed on an analytical balance the coating is dissolved by immersion in an appropriate solution. The specimen is then washed, dried and reweighed, the difference in weight in grams being numerically equal to the weight of coating in ounces per square foot. From time to time, and particularly when certain specifications governing the composition of the terne coating must be met, the coating on a test sample is chemically analyzed to determine the percentages of tin and lead present.

A microscopic examination of a properly polished and etched cross-section is made to determine the ferrite grain size of the steel base. Since the adherence of the coating to the base metal is never a problem in the production of long terne sheets, adherence testing is not required as it is for galvanized sheets. **Mechanical property tests** are used primarily as a guide in determining the ductility of the coated sheet and its suitability for the intended application. These mechanical tests include

Rockwell hardness, Olsen cup, bend, fluting and tension test. Hardness tests are normally made on the base metal after the coating has been removed since the soft coating would give erroneous values if the hardness impression were made through it. Besides providing a measure of the ductility, the Olsen test is a valuable tool for detecting hydrogen embrittlement of the base metal. In the Olsen cup test the material normally "necks" at the periphery of the base of the cup, but when absorbed hydrogen has embrittled the base metal, a ragged fracture will occur on the side of the cup or, in extreme cases, at the base of the cup. Fluting tests are made by forming a suitable test piece into a cylinder and observing the presence or absence of flutes or creases.

Continuous Strip Production of Long Ternes—The advantages of the continuous terne coating of strip have long been recognized. Constant improvement, development and adaptation of existing terne coating equipment have now made continuous hot-dip coating possible. The first major step was the development of completely synchronized drives, so that long coils could be run without gain or loss in the loop between rolls. Other developments included the perfection of automatic welding equipment and vertical branners. Continuous coating lines may be of the semi-continuous or the fully-continuous type. The semi-continuous line has only one payoff reel and one tension reel, which necessitates threading each individual coil through the line. The fully-continuous line is equipped with double payoff reels, welder, and looping towers or pits, which permit a continuous flow of strip through the coating unit.

The base metal for coating on continuous lines usually consists of cold reduced, box annealed and temper rolled coils, although hot-rolled, pickled coils may be used. The treatment of the base metal prior to coating is identical to that given sheet product, with the exception that all the operations are accomplished in coil form. The pickling operation is performed in the line prior to the actual terne coating operation.

The equipment contained in a continuous coating line may vary from shop to shop, depending on numerous operating conditions and the preference of the manufacturers concerned. Essentially, the equipment and process involved are as follows: A coil holder, followed by a payoff reel, feeds the strip into a pinch-roll unit, which in turn is followed by a squaring shear and a welder if the process is fully continuous. The strip then enters an acid pickling tank, which may consist of a single-dip tank, wherein normal chemical pickling is effected, or which may be an electrolytic pickling unit; if the latter, it may be followed by a short non-electrolytic hydrochloric dip. In either case, the object of the operation is to provide a chemically clean surface for the effective application of the terne coating. If desired, provision may be made for an alkaline cleaner unit equipped with suitable rinsing and scrubbing facilities just ahead of the pickling tank. From the pickling unit the strip then enters a pinch roll unit which conveys it into the terne coating pot. Upon emerging from the pot, the strip is cooled and conveyed to the branner (vertical in modern installations) which removes the residual oil. At this point in a continuous line, provision may be made for shearing the strip into cut lengths, using a gage table, shear, and piler or, if the material is to be shipped in coil form, a shear for cutting out the weld and a tension reel are available to recoil the strip.

The process of continuous terne coating has several unique advantages which suggest that its use will become more widespread. The first of these is the elimination of the intermittent operation characteristic of sheet

pots, whereby a space of from 1 to 3 feet is left between each sheet as it passes through the line. The next distinct advantage is the uniformity of the coating that can be realized by having a continuous flow of the base metal through the coating rigging with no break in the operation. As has been indicated, the sheet method results in a heavier coating on the front and back ends, "entry" and "list" ends respectively, so that continuous coating makes more efficient use of a given quantity of coating metal. A disadvantage of the continuous process lies in the fact that it is less adaptable to producing small, varied types of orders since it is most efficient when used to produce coils of one coating weight. The operation of a continuous coil terne line is quite similar to a sheet unit and the principles of the coating technique are identical. Since it is not possible to make use of the sheet weight test as a routine check on coating weights, the operator must rely on a determination made by electrical or chemical means. The final product may be shipped to the customer in coil form or may be sheared to sheets.

Recent developments in continuous terne coating have pointed toward the elimination of oil as a part of the coating operation. The use of oil entails a number of operating disadvantages, among which is the necessity of including a cleaning apparatus to remove excess oil from the coated strip. This operation can apparently be avoided by dispensing with oil finishing rolls and substituting partially submerged exit rolls, using a molten zinc chloride flux with ammonium bromide or ammonium chloride dissolved in it. The zinc chloride acts as a solvent for the ammonium bromide or ammonium chloride to prevent its dissipation by volatilization and makes the entire process quite similar to conventional galvanizing.

Applications of Long Terne Sheets—The major portion of the long terne sheet production is used in the manufacture of gasoline tanks for the tractor, truck and automotive industries. Automotive gas tanks are usually drawn in two halves, top and bottom, and seam welded around the perimeter. Truck and tractor tanks are sometimes drawn in the same manner, but are also fabricated by forming the sheets into cylinders by using bending rolls and then lockseaming or soldering the ends. The use of long ternes for these applications is dictated by the resistance of the coating to corrosion by gasoline, water and air. Other automotive parts manufactured from long ternes include water distributor tubes, radiator parts, mufflers, oil pans and air cleaners. The latter use is considered a minor corrosion hazard but manufacturers of this item prefer to draw the shapes involved from long ternes because of the lubricating value of the coating in deep drawing operations. Other uses are for roofing, hand fire extinguishers, outboard-motor gasoline tanks and burial caskets. The latter application involves hand rubbing and polishing to a high luster for a pleasing appearance. An important wartime use was the manufacture of expendable gasoline tanks for aircraft.

Terne Plate—Terne plate, a tin mill product of relatively minor importance, is black plate that has been pickled and coated with terne metal. Known occasionally as **short ternes**, this commodity is made on hot-dipped manufacturing equipment. Its approximate sheet thickness is expressed as basis weight and its coating weights are expressed in terms of pounds per double base box. The terne metal used for terne plate generally contains a maximum of 15 per cent tin; the balance of the coating is lead.

Roofing ternes are commonly ordered in the standard sizes, 20 by 28 inches or 14 by 20 inches, the standard

basis weights of 107 lb. and 135 lb., and the standard coating weights of 8 pounds, 20 pounds and 40 pounds per double base box. The base metal is copper-bearing steel.

Firedoor ternes are ordered in the standard size 20 by 28 inches with a basis weight of 107 lb. or heavier and a coating of 20 pounds per double base box.

Manufacturing ternes are short ternes of any tin-mill size produced by using approximately 6 pounds of terne metal per double base box. **Heavy-coated manufactur-**

ing ternes are short ternes produced in coating weights of 8, 15, 20 and 40 pounds per double base box. **Special-coated manufacturing ternes** are short ternes customarily produced with no specified amount of coating but carrying the lightest practical coating consistent with thorough coverage of the plate.

Bibliography

"Steel Products Manual", Sections 11 and 14. American Iron and Steel Institute, New York.

Chapter 39

PRODUCTION OF GALVANIZED SHEET AND STRIP

SECTION 1

GENERAL

Production and Uses of Galvanized Sheet and Strip— The importance of providing protection against corrosion for steel articles having a light section is obvious, and coating the steel with zinc is a very effective and economical means of accomplishing this end. Zinc coatings are commonly applied by dipping or passing the article to be coated through a molten bath of the metal. This operation is termed "galvanizing," "hot galvanizing" or "hot-dip galvanizing" to distinguish it from zinc electroplating processes which are termed "cold" or "electro-galvanizing."

The present discussion is limited (1) to **hot-dip sheet galvanizing** which is the hot-dip galvanizing of cut-length sheets by passing them one by one and in close succession through the molten zinc, and (2) **continuous (strip) hot-dip galvanizing**, in which material in coiled form from the rolling mills is uncoiled and passed continuously through the galvanizing equipment, continuity of operation being achieved by joining the trailing end of one coil to the leading end of the next.

The word "strip," as used in this chapter, denotes the *physical form* of the material in process; i.e., a continuous strand of steel of sheet gage and width. Commercially, the meaning of the term "strip" has the limitations outlined in Chapter 33.

Of all the common metals used for protective coatings, zinc enjoys the lowest cost per pound. With the single exception of tin, it is used to protect a greater area of steel than any other coating metal. In 1954, 398,000 tons of slab zinc were consumed by the galvanizing industry. Of this amount, 181,000 tons were used for the galvanizing of sheet and strip. The balance of the zinc was used largely in wire galvanizing, pipe galvanizing, and in hot-dipping prefabricated articles such as range boilers, wash tubs, garbage cans, pails, etc. The reported shipments of galvanized sheet and strip in the United States during 1954 amounted to over 2.5 million tons.

In 1954, about 37½ per cent of the galvanized-sheet production was used to satisfy construction-industry requirements. Construction of grain-storage bins, stock tanks, feeding equipment, farm machinery and buildings in the agricultural industry consumed 10.4 per cent of the production. The electrical industry consumed about 2½ per cent of the total, and about 34 per cent was shipped to warehouses for miscellaneous uses. Substantial tonnages were used by the appliance and container industries. The railroads consumed around 1 per cent and the export trade about 6½ per cent of the total.

Specific uses for galvanized sheets include roofing and siding, eave troughs, conductor pipe, air ducts, heating furnaces, air-conditioning equipment, metal

lath, corner beading, switch boxes, chimney flues, metal awnings, railroad-car roofs, garbage cans, oil cans, ash cans, pails, tubs, coal hods, refrigerators, deep-freeze units, kitchen equipment, laundry tubs, automatic washers and dryers, minnow buckets, storage tanks, beverage coolers, well casings, culverts, poultry equipment, manure spreaders, hay loaders, tractors, and wagons.

FACTORS INFLUENCING EFFECTIVENESS OF GALVANIZED COATINGS

The amount of zinc on a galvanized sheet is stated in terms of ounces per square foot of sheet; since the sheet is coated on both sides, the stated weight of coating is twice the average weight of coating per square foot of either side.

The effectiveness of a protective coating depends to a considerable extent upon the character of the environment in which it is used. In general, galvanized coatings are subjected to atmospheric and liquid corrosion and, less frequently, to soil corrosion. Their effectiveness in the atmosphere depends mainly upon the amount of acidic contaminants present. Thus, in the relatively pure air of rural districts, the life of zinc coatings is four to ten times that of the same coatings in industrial areas.

For any specific set of exposure conditions, the thickness of a zinc coating is the most important factor in determining its effectiveness. Coatings applied to sheets by the hot-dip process range, in general, from 0.6 to 2.5 oz. per sq. ft. of sheet, and the 2.5 oz. per sq. ft. coating can be expected to have approximately four times the life of the 0.6 oz. per sq. ft. coating under similar exposure conditions. For this reason, uniformity of coating is also an important consideration. As a practical matter a completely uniform coating is not produced in a commercial hot-dipping operation; however, no area or spot of a hot-dipped sheet should carry a lesser weight of coating than the trade-accepted minimum for the grade. These minimums vary with the grade or class ordered and are considerably less than the nominal weight of coating, i.e., coating class, specified; see "Coating Weight Requirements" below.

One of the advantages of zinc as a protective coating is its anodic relationship to iron in the electro-motive series. Due to this relationship, the protection afforded is extended to small areas of coating discontinuity adjacent to areas actually coated. Thus, lack of coating at a sheared edge or minor damage to the coating during fabrication or use of a galvanized article does not seriously impair its service life, provided, of course, the uncoated areas are not too large.

Table 39—I. Galvanized Sheet Gage Numbers with Equivalent Unit Weights

Galvanized Sheet Gage Number	Ounces Per Square Foot	Pounds Per Square Foot	Pounds Per Square Inch	Thickness Equivalents (Inches)
8	112.5	7.03125	0.048828	0.1681
9	102.5	6.40625	0.044488	0.1532
10	92.5	5.78125	0.040148	0.1382
11	82.5	5.15625	0.035807	0.1233
12	72.5	4.53125	0.031467	0.1084
13	62.5	3.90625	0.027127	0.0934
14	52.5	3.28125	0.022786	0.0785
15	47.5	2.96875	0.020616	0.0710
16	42.5	2.65625	0.018446	0.0635
17	38.5	2.40625	0.016710	0.0575
18	34.5	2.15625	0.014974	0.0516
19	30.5	1.90625	0.013238	0.0456
20	26.5	1.65625	0.011502	0.0396
21	24.5	1.53125	0.010634	0.0366
22	22.5	1.40625	0.0097656	0.0336
23	20.5	1.28125	0.0088976	0.0306
24	18.5	1.15625	0.0080295	0.0276
25	16.5	1.03125	0.0071615	0.0247
26	14.5	0.90625	0.0062934	0.0217
27	13.5	0.84375	0.0058594	0.0202
28	12.5	0.78125	0.0054253	0.0187
29	11.5	0.71875	0.0049913	0.0172
30	10.5	0.65625	0.0045573	0.0157
31	09.5	0.59375	0.0041233	0.0142
32	09.0	0.56250	0.0039062	0.0134

COATING WEIGHT AND GAGE REQUIREMENTS

Galvanized sheets are manufactured to several general specifications, the requirements of which have been found by usage and test to be the most suitable for the applications involved.

The so-called **Commercial Coatings** generally include those that are produced on sheets for direct weather-exposure applications. Galvanized Formed Roofing and Siding, and Galvanized Flat Sheets for Roofing and Siding, are produced to ASTM Specification "Steel Roofing Sheets" A361-55T with a 1.25 oz. per sq. ft. coating class, 0.90 oz. per sq. ft. check limit by triple-spot test, or 0.80 oz. per sq. ft. check limit by single-spot test. (Tests for galvanized sheets are described in Section 5 of this chapter.) Galvanized Flat Sheets for other than roofing and siding products, but for an intended equivalent corrosion hazard, may be produced to ASTM Specification "Steel Sheets, Coil and Cut Lengths" A93-55T with a similar 1.25 oz. per sq. ft. coating class.

For applications in which the above Commercial Coatings might cause some difficulty in forming because of their thickness, or in which extreme corrosion resistance is not a consideration, galvanized sheets may be produced to ASTM Specification A93-55T, Light Commercial, with no minimum check limits specified.

Heavier than Commercial Coatings are supplied when required for specific applications. The American Zinc Institute "Seal of Quality" roofing sheet is specified as 2.00 to 2.25 coating class with a minimum 1.80 oz. per sq. ft. check limit by triple-spot test and a minimum 1.50 oz. per sq. ft. check limit by single-spot test (see Section 5). Culvert sheets are required to carry a minimum 2.00 oz. per sq. ft. coating when tested by the triple-spot-test method.

Galvanized sheets are produced to weights per unit of sheet area. The Galvanized Sheet Gage, established by custom, is based on the United States Standard

Gage, each Galvanized Sheet Gage weight being 2.5 ounces per square foot heavier than the gage weight of the same United States Standard Gage number, regardless of the coating weights. For example, the equivalent unit weight of a 20-gage galvanized sheet is 26.5 ounces per square foot regardless of the weight of coating. Adjustments are made in the weight of the base metal to compensate for heavier or lighter weights of coating. Galvanized sheet gage numbers and their weight equivalents are given in Table 39—I. Thickness equivalents are also shown in this table. Table 39—II shows the coating class in ounces per square foot and the minimum coating check limits for coatings heavier than Commercial Galvanized Coatings.

GENERAL QUALITY DESIGNATIONS

Sheets of "Commercial Quality" are suitable for bending and moderate forming requirements. They must be capable of withstanding standard bend tests.

Table 39—II. Coating Class and Minimum Coating Check Limits for Galvanized Sheets (Oz. Per Sq. Ft.)

Coating Class	Minimum Check Limit by Triple-Spot Test*	Minimum Check Limit by Single-Spot Test*
2.75	2.35	2.00
2.50	2.10	1.80
2.25	1.85	1.60
2.00	1.65	1.40
1.75	1.40	1.20
1.50	1.15	1.00
1.25 (Commercial)	0.90	0.80
Light Commercial	—	—

* Test procedures are described in Section 5 of this chapter.

The adherence of the coating must be sufficient to withstand bending without flaking when tested in accordance with ASTM Specification A-93.

When "Drawing Quality" steel is supplied, the producer assumes responsibility for selection of steel, control of processing, and performance of the material within established limits. There is no standard of coating adherence for drawing operations, so this item is subject to agreement between producer and user. Drawing Quality sheets are made from rimmed steel.

For critical forming requirements, "Drawing Quality—Special Killed Steel" should be specified. The limitations described above for "Drawing Quality" also apply to this grade of material.

Temper rolling may be employed to produce an "Extra Smooth" sheet for applications involving critical end uses when galvanized sheets are required to have one smooth, clean surface of a higher degree of smoothness than is normal for regular commercial quality, or drawing quality product. Examples are for sign and panel work.

To obtain satisfactory performance in lock-forming operations, "Lock-Forming Quality" sheets should be employed.

For superior resistance to atmospheric corrosion, copper-bearing steel should be employed.

SPECIALTY PRODUCTS

Roofing and siding are produced either flat or in the form of "Corrugated Sheets," "V-Crimped Roofing," "Corrugated Roll Roofing," or regular "Roll Roofing."

Standard 2½-inch corrugated sheets have 10 corrugations to a sheet of siding and 10½ corrugations for a sheet of roofing for a covering width of 24 inches. The width of each corrugation is actually 2.66 inches and the depth is ½ inch. In siding, both edge corrugations are turned the same way and the overall width of the formed sheet is 26 inches. In roofing, the edge corrugations are turned in opposite directions and the total sheet width is about 27½ inches. Lengths of 5, 6, 7, 8, 9, 10, 11, and 12 feet are commonly available, but special lengths are sometimes supplied. Galvanized roofing and siding sheets are produced from 29-gage and heavier sheets.

The standard 1¼-inch corrugated sheets have 20½ corrugations, ¼ inch deep, to a sheet. With one edge each way, they have a formed width of 26 inches and a covering width of 24 inches. Stock lengths are the same as those for the 2½-inch corrugated stock and the gages supplied are similar.

V-crimped roofing is produced by forming 2, 3 or 5 longitudinal inverted "V" crimps in the sides or in the sides and centers of sheets. V-crimped sheets are supplied with the 24-inch covering width and the 5- to 12-foot lengths used for the corrugated material. Various modifications of the standard "V" crimp are used for roofing and siding sheets.

Various accessory fittings for use in assembling a corrugated roof usually are supplied by galvanized-sheet producers. Included in these are ridge roll, ridge cap, gambrel joint, side-wall flashing, end-wall flashing, lean-to filler, and overhang section.

SECTION 2

METALLURGICAL FEATURES OF THE HOT-DIP GALVANIZING PROCESSES

Hot-dip coatings are produced on steel objects by passing the properly prepared base metal through a bath of molten coating metal. Tin, zinc and lead are the common coating metals used. As in all coating processes, the surface of the base metal must be clean to insure satisfactory coating continuity. In addition, successful hot-dipping requires the surface of the molten bath to be kept clean, particularly at the entrance to the bath, since oxide films of the coating metal at this point will be picked up by the surface of the entering base metal to interfere with the wetting of the latter by the molten coating metal.

In the hot-dip galvanizing of steel, the cleanliness of the base metal is somewhat less critical than in electroplating operations since molten zinc is highly reducing to iron oxide and possesses a strong tendency to form intermetallic compounds with iron. Excessive contamination of the surface with any material, of course, cannot be tolerated and must be removed by acid pickling, alkaline cleaning or other preparatory treatment prior to introduction in the zinc bath. Satisfactory cleanliness of the steel and zinc is maintained by (1) the use of suitable fluxes either carried on the surface of the bath or applied to the surface of the steel just prior to entry into the bath, or (2) in the case of continuous operations which include an annealing or other heat-treating step immediately ahead of the galvanizing pot, by keeping the heated strip under the protection of the furnace atmosphere, preferably a reducing atmosphere, until it passes below the surface of the molten zinc.

For many years, galvanized sheets were produced by passing single sheets of steel in close succession through a coating pot; the operations are described in

detail in Section 3 of this chapter. However, with the development of wide strip mills capable of rolling to sheet gages, there has been a gradual shift to processing sheet products in strip form. In such operations, the steel is unwound from coils and passes as a continuous strand through the processing equipment. The product may then be recoiled or cut into individual sheets as desired. Processing in strip form is sometimes referred to as "continuous processing" to distinguish it from similar operations involving the treatment of individual sheets. Continuous processing permits many operations formerly conducted on separate pieces of equipment to be combined into one continuous processing line. In addition, certain problems inherent to operations on individual sheets disappear when the steel is processed in strip form. For example, in the hot-dip coating of sheets the necessary gap between successive sheets causes a considerable variation in the weight of coating applied to the leading and trailing ends of the sheet. The absence of this gap in operations on material in strip form therefore results in a more uniform coating and consequently affords better utilization of a given quantity of zinc. Continuous-type operations, however, require a very high investment in equipment. A detailed description of the application of continuous practices in hot-dip galvanizing of steel is given in Section 4 of this chapter.

At and above its melting point, zinc readily forms intermetallic compounds with iron. In hot-dip galvanizing, these alloys form at the interface between base metal and coating metal and ultimately constitute an appreciable portion of the finished coating. After solidification, the coating consists of an outer layer approaching the metal of the coating bath in composition, and



FIG. 39-1. Typical spangles on a "Tight-Coat" sheet galvanized with aluminum-free spelter. Average coating weight: 1.10 oz. per sq. ft. Actual size, unetched.

several inner layers, generally termed "alloy layer," consisting of intermetallic phases of iron and zinc, the latter becoming successively richer in iron with depth.

The outer layer of a hot-dip galvanized coating solidifies as a cast crystalline structure starting at scattered nuclei and developing into a more or less regular "frost-flower" or "spangled" pattern (Figures 39-1 and 39-2). The size of the spangles is influenced by the composition and prior treatment of the base metal, by the duration of the solidification period, and by the composition of the coating bath. The shape and general pattern of the spangles are largely influenced by the latter.

The alloy layer has been the subject of considerable investigation. It has been considered to be the bond between the outer layer of relatively pure zinc and the

steel base; but, paradoxically, many of the failures of galvanized coatings by flaking during fabrication have also been attributed to its presence. Actually the exact role of the layer in adherence of the coating has not been clearly established.

It has been known for a number of years that the addition of aluminum in small amounts (0.10 to 0.25 per cent) to the zinc bath suppresses the alloy formation and produces coatings of increased tightness. However, upon metallographic examination, coated samples made with a nominal 0.15 aluminum spelter sometimes show a relatively thick alloy layer and yet pass severe forming tests. Moreover, although the presence of a heavy alloy layer on steel sheets galvanized with aluminum-free spelter generally portends bad flaking, sheets galvanized in the same manner but galvanized, i.e., the coating



FIG. 39-2. Typical spangles on a sheet continuously galvanized with aluminum-bearing (nominal 0.15 per cent Al) spelter. Average coating weight: 1.32 oz. per sq. ft. Actual size, unetched.

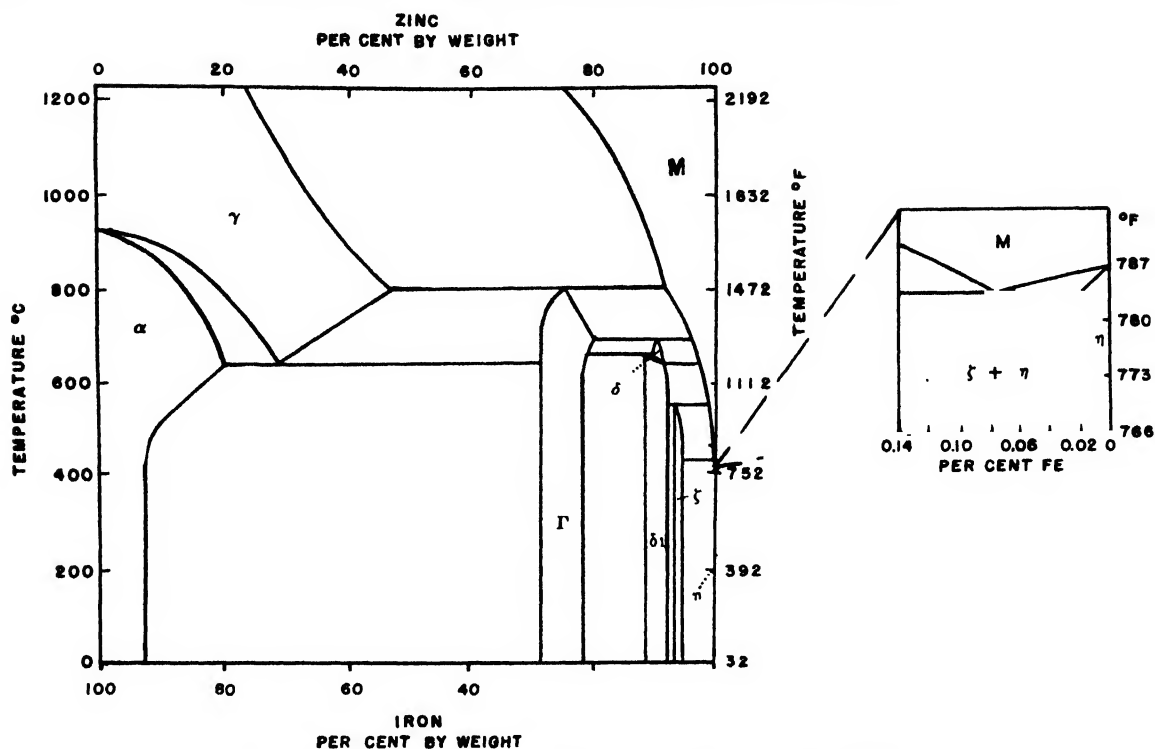


FIG. 39-3. The iron-zinc constitution diagram, according to J. Schramm

converted entirely to alloy, will exhibit excellent adherence. Thus there are indications that the composition of the layer may be significant and that the thickness of the layer is a factor; however, any generalizations must be accepted with reservation.

The equilibrium relationship between iron and zinc at different temperatures has been studied by various investigators. The constitution diagram as developed by J. Schramm (1938) is shown in Figure 39-3. The composition and lattice structure of each phase indicated on the diagram are shown in Table 39-III.

Intermetallic compounds present in hot-dipped galvanized coatings produced under a variety of conditions are indicated in Figures 39-4 through 39-7. This series of illustrations shows that the clarity of definition and the relative amounts of the various constitu-

ents of the alloy layer can be varied by varying time and temperature of immersion. The composition of the steel also, particularly the silicon content, can have a noticeable influence on the layer. The presence of tin, antimony and cadmium in the usual amounts found or used in spelter have been reported to have very little effect on the character of the layer; however, aluminum in excess of 0.10 per cent has a decided effect. Figure 39-8 is a photomicrograph at 1000 diameters showing the cross-section of a sheet galvanized coating of aluminum-free spelter.

Figure 39-9 is a similar photomicrograph of a strip galvanized coating of aluminum-bearing (nominal 0.15 per cent Al) spelter. It will also be noted that almost all the coating in the latter instance consists of the eta phase.

Table 39-III. Phases in the Iron-Zinc Constitution Diagram

Phases	X-Ray Formula	Limits of Composition, (Per Cent Fe)		Space Lattice
		Atomic	Wt.	
Alpha (α) Fe	FeZn		80-100	Body-Centered Cubic
Gamma (γ) Fe	FeZn		55-100	Face-Centered Cubic
Capital Gamma (Γ)	Fe ₃ Zn ₂₁ or Fe ₂ Zn ₁₀	23.2-31.3	20.5-28	Body-Centered Cubic
Delta ₁ (δ ₁)	FeZn ₇ (?)	8.1-13.2	7-11.5	Hexagonal (?)
Delta (δ)	FeZn ₇ (?)	8.1-11.5	7-10	Monoclinic
Zeta (ζ)	FeZn ₁₈	7.2-7.4	6-6.2	Hexagonal
Eta (η)	Zn		Max. 0.003	Close Packed

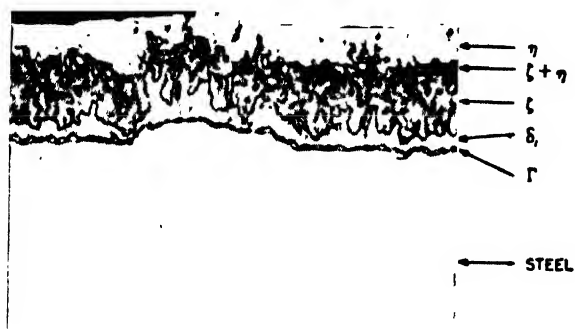


FIG. 39-4. Photomicrograph of commercial sheet galvanized coating. Average coating weight, 0.63 oz. per sq. ft. Magnification: 1000X.



FIG. 39-5. Photomicrograph of experimental galvanized coating formed by immersing sheet steel for one minute in molten zinc at 840° F (450° C). Magnification: 500X.



FIG. 39-6. Photomicrograph of experimental galvanized coating formed by immersing sheet steel for five minutes in molten zinc at 840° F (450° C). Magnification: 500X.

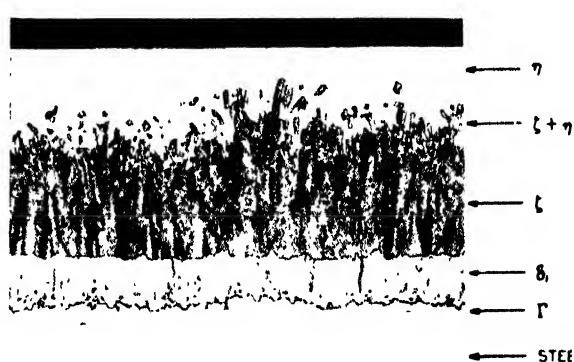


FIG. 39-7. Photomicrograph of experimental galvanized coating formed by immersing sheet steel for ten minutes in molten zinc at 840° F (450° C). Magnification: 500X.

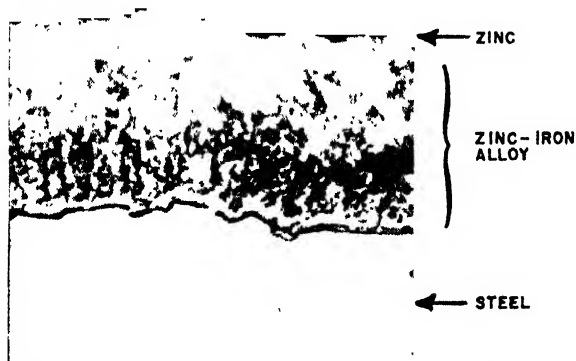


FIG. 39-8. Photomicrograph showing thickness of the alloy layer in the coating on a sheet galvanized with aluminum-free spelter. Average coating weight, 1.13 oz. per sq. ft. Magnification: 1000X.

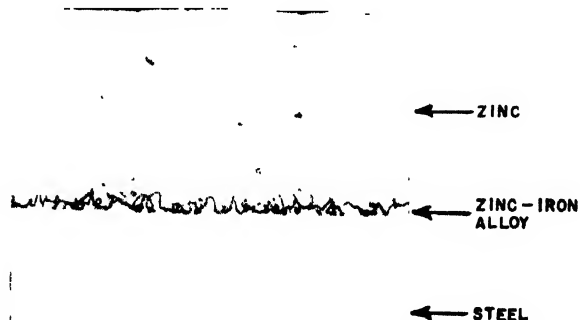


FIG. 39-9. Photomicrograph showing thickness of the alloy layer in the coating on a sheet galvanized with aluminum-bearing spelter (nominal 0.15 per cent Al). Average coating weight, 1.17 oz. per sq. ft. Magnification: 1000X.

COATING METAL USED IN HOT-DIP GALVANIZING

There are a number of commercial grades of zinc or spelter available for hot-dip galvanizing. A list of the different grades is given in Table 39—IV.

Table 39—IV. Composition of the Different Grades of Zinc and Spelter

Grade	Composition, Maximum Per Cent of			
	Pb	Fe	Cd	Pb + Fe + Cd
Special High Grade ..	0.010	0.005	0.005	0.01
High Grade ..	0.07	0.03	0.07	0.10
Intermediate .	0.20	0.03	0.50	0.50
Brass Special .	0.60	0.03	0.50	1.00
Prime Western.	1.60	0.08

In either sheet or continuous-strip galvanizing, it is the general practice to add small quantities of other metals to the coating bath to control the appearance and properties of coatings. The general effects of elements that are either found in or are added to molten commercial grades of zinc are discussed in the following paragraphs:

Lead, in certain percentages, is necessary for the production of a spangled finish. If present in amounts exceeding its solubility in zinc (around 1.00 per cent at coating temperatures), the excess lead settles to the bottom of the coating pot and serves as a cushion to support a bed of zinc-iron alloy or dross that forms during the coating operation.

Antimony is not present in commercial spelter grades and it may be added, if desired, either as an alloy of zinc or as the metal itself. In small amounts it assists in producing an attractive low-relief spangle.

Cadmium, which is present in most virgin splatters, assists in producing a frosty low-relief spangle finish.

Tin additions of from 0.3 to 1.50 per cent have been used for many years to produce a frosty spangle finish in conventional galvanized coatings.

Aluminum additions of between 0.10 and 0.25 per cent increase the adherence of galvanized coatings. The addition cannot be used in conjunction with practices utilizing any of the common chloride fluxes floating on the surface of the zinc, since the aluminum is rapidly removed from the bath by the flux as $AlCl_3$, which is highly volatile at galvanizing temperatures.

A small amount of iron is always present in a hot-dip galvanizing bath. The excess above its low solubility limit gradually settles to the bottom of the pot as dross.

STEELS USED FOR HOT-DIP GALVANIZING

Hot-dip galvanized coatings may be successfully applied to a variety of ferrous metals, ranging from cast iron to low-carbon and alloy steels. In the coating of sheet, the bulk of the production is from low-carbon basic open-hearth steel. The following is a general classification of the grades used in galvanizing:

- A. Rimmed Steel
 - 1. Regular low-carbon
 - 2. Pure iron, copper-bearing
- B. Capped Steel
 - 1. Mechanically capped
- C. Killed Steel
 - 1. Aluminum killed
- D. High-Strength Steel
 - 1. High-strength low-alloy

Copper may be added, if desired, to any of the above grades to improve atmospheric corrosion resistance. Steels containing between 0.20 per cent and 0.30 per cent copper are designated as "copper bearing."

Chemical requirements for the various steel grades, as specified by a majority of the producers, are shown in Table 39—V.

MILL TREATMENT OF STEEL PRIOR TO HOT-DIP GALVANIZING

Sheets used for sheet galvanizing may be processed by hand-mill rolling operations (now practically obsolete) or by continuous rolling processes. Hand-mill sheets are hot rolled, box annealed, cold rolled, and pickled to prepare them for hot-dipping. Continuous-hot-rolled sheets may be either hot rolled in strip form or hot rolled and pickled in strip form before shearing to cut lengths and pickling for the galvanizing operation. Shot blasting after shearing is sometimes used to facilitate the production of heavy coatings on heavy-gage material. Cold-reduced sheets are either hot-rolled, pickled, cold-reduced, and recoiled, or they are hot rolled, pickled, cold reduced, recoiled, and sheared to cut-length sheets before box annealing. Alkaline cleaning is often used after cold reduction to remove rolling lubricants. Material that is box annealed in coils is temper rolled and sheared before pickling for galvanizing, and material that is box annealed in sheet form is temper rolled before pickling for galvanizing.

Coils for continuous galvanizing may be prepared in a number of ways. Thus, they may be hot rolled and pickled; hot rolled, pickled and cold reduced; or hot rolled, pickled, cold reduced, box annealed and temper rolled before processing. Again, alkaline cleaning may be used after cold reduction to remove rolling lubricants. Because of the inherent properties of the base metal, coated sheets made from cold-reduced coils have been found to possess ductility superior to that of coated sheets made from handmill products.

SPECIAL FINISHES

In addition to the regular spangled finish produced in sheet or strip galvanizing, provision may be made in the processing facilities to accommodate the production of special finishes. **Galvannealed sheets** may be produced by passing the material, after it leaves the coating pot, through a heated chamber that maintains the coating metal molten until it alloys completely with the base metal. Galvannealed coatings possess a silvery matte finish of relatively low reflectivity. They have good paint-adherence properties without additional surface preparation and are capable of withstanding moderate forming.

A **bonderized finish** may be produced by processing in line or as a separate supplementary processing operation. In the bonderizing process, a thin film composed largely of zinc phosphate is formed on the sheet surfaces. Bonderized sheets are outstanding with regard to their paint-adherence properties.

Extra-smooth finishes may be applied after galvanizing, galvannealing, or bonderizing, by temper rolling with shot-blasted or with smooth rolls. On bonderized

Table 39—V. Chemical Composition of Sheet and Strip Galvanizing Steels

Grade	Carbon (%)	Manganese (%)	Phosphorus (%)	Sulphur (%)	Silicon (%)	Copper (%)	Comments
Regular Rimmed	0.05-0.10	0.25-0.50	0.030 Max.*	0.050 Max.	Residual	Residual	Any of these Steels May Be Supplied with Copper Content of 0.20% Min.
Mechanically Capped	0.05-0.10	0.25-0.50	0.030 Max.*	0.050 Max.	Residual	Residual	
Aluminum Killed	0.05-0.10	0.25-0.50	0.030 Max.*	0.050 Max.	Residual	Residual	
Silicon Killed	0.05-0.10	0.25-0.50	0.030 Max.*	0.050 Max.	0.15-0.25	Residual	
High Strength	0.05-0.20	0.30-1.30	0.030 to 0.11	0.050 Max.	Residual to 0.80	Residual to 0.60	<div> <div>Nickel</div> <div>Residual to 1.00</div> <div>Chromium</div> <div>Residual to 1.00</div> </div>
Pure Iron, Copper Bearing (Culverts)	L.A.P.**	L.A.P.**	0.015 Max.	0.040 Max.	Residual	0.20 Min.	
Copper Iron (Culverts)	0.015 Max.	0.040 Max.	Residual	0.20 Min.	<div> <div>Sum of 1st Five Elements—0.10 Max.</div> </div>
Steel, Copper Bearing (Culverts)	0.05-0.10	0.25-0.40	0.030 Max.	0.050 Max.	Residual	0.20 Min.	<div> <div>Sum of 1st Five Elements—0.25 Max.</div> </div>

* Phosphorus contents may be varied up to 0.10 per cent maximum to facilitate pack rolling or to improve the adherence of the zinc coating.

** L.A.P. = Low as possible.

material the extra-smooth finish may, in some instances, be applied before bonderizing.

Metal sheets shipped and stored in piles or coils are subject to white or variously colored stains when water is allowed to remain in contact with surfaces in the interiors of piles and coils for any appreciable period of time. This condition is usually designated as "wet storage stains." Common sources of moisture may be rain water, melted snow, or condensation, the latter

resulting from sudden temperature changes. Various chemical films and oil films have been found to be effective in retarding the appearance of the discoloration, but, even with filming treatments, wet storage stain is a hazard when sheets or coils get wet.

Good packaging, handling and adequate warehousing facilities have been found to be essential requirements for preserving the original finish until the material is used.

SECTION 3

HOT-DIP SHEET GALVANIZING

Pickling for Sheet Galvanizing—Pickling for sheet galvanizing is usually conducted as a batch operation in stationary tubs provided with an agitating means. This operation may sometimes be conducted as a continuous process in equipment provided with a sheet conveyor and means for electrolytic acceleration.

Very light pickling requiring only a short-time exposure to the pickling solution has been found suitable for products, such as roofing and siding, that require little mechanical deformation. Deep etching of the base metal has generally been found to be necessary when forming requirements are severe.

Following the pickling operation, a thorough water rinse, with or without an alkaline neutralizing dip, is employed to remove iron salts and residual acid. A prolonged immersion in boiling water may be used, when necessary, to minimize hydrogen embrittlement.

Equipment for Sheet Galvanizing—The equipment used for sheet galvanizing (Figure 39—10) consists of mechanical facilities for transporting cut-length sheets successively through acid washing, fluxing, hot-dipping, and cooling operations. The coating bath itself is contained in a heated low-carbon steel vessel or pot. A framework or rigging with suitable entry feed rolls, sheet guides, driven bottom pinch rolls, and driven

exit rolls, is suspended in the bath in such a manner as to completely submerge all but the entry rolls, part of the exit rolls, and part of the supporting framework. A baffled section or flux box at the entry end contains a floating fused-chloride flux prepared from sal ammoniac and the zinc of the bath.

General Arrangement and Operation of a Sheet-Galvanizing Line—A typical sheet-galvanizing line consists of a feeding table, a muriatic (hydrochloric) acid tank, a coating pot, a spangle and cooling conveyor, a water rinse tank, a dryer, a roller leveler and an inspection table (see Figure 39—11). In operating the line, pickled sheets are either fed dry from a feeding table or wet from water boshes directly into a set of driven rubber pinch rolls. The sheets pass forward through a shallow muriatic-acid bath and a second set of rubber pinch rolls into steel intake rolls that force them downward into the flux box of the coating pot. The sheets then pass into the coating bath downward to the driven bottom rolls, and upward through exit rolls, the latter being located at the bath surface. As they emerge from the exit rolls the sheets may have blasts of air, sulphur-dioxide fumes, or powdered sal ammoniac directed against their edges to produce the surface finish desired. To control the amount of coat-

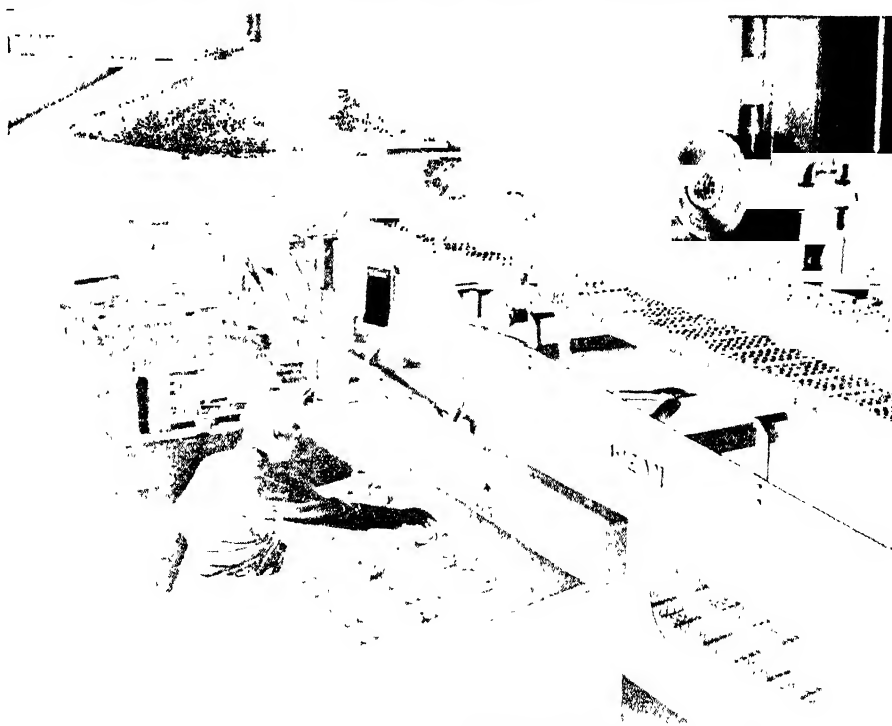


FIG. 39—10. A sheet galvanizing line in operation, showing the hooded setting which contains the galvanizing machine immersed in the pot containing molten zinc, the spangle conveyor, and control station.

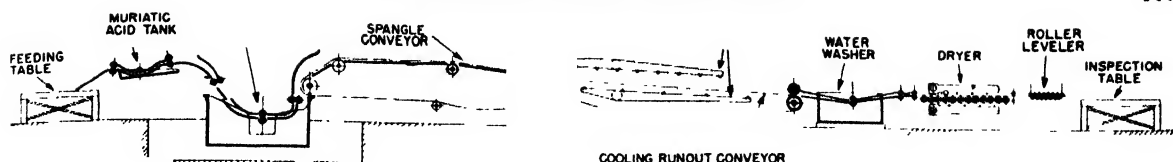


FIG. 39—11. Schematic side elevation of a conventional sheet galvanizing line.

ing applied, the level of the bath may be adjusted or the pressure against the exit rolls may be regulated.

Endless belts for conveying coated sheets away from a galvanizing pot are generally made of coarse woven-wire netting. The meshes of the netting contact the sheet in such a manner as to cause rapid solidification at regularly distributed points. These provide nuclei for crystallization and serve, to some extent, as a means for spangle control. A cooling run-out conveyor, usually provided with air blasts, follows the spangle conveyor in a typical installation. Cooled sheets from

the conveyor are roller leveled in line before inspecting and piling. In some coating lines, a water-rinsing and drying operation precedes the leveler. At the delivery end of the coating line the sheets are inspected, counted, stenciled, and stacked into lifts, after which they may be branded and bundled or otherwise prepared for shipment.

If a special finish, such as a bonderized finish, is to be applied to the galvanized sheets, they may be delivered from the galvanizing line to further processing units before shipment.

SECTION 4

CONTINUOUS (STRIP) HOT-DIP GALVANIZING

General Arrangement and Operation of Continuous Galvanizing Lines—The processing of steel in the form of continuous lengths unwound from coils requires more elaborate equipment than the processing of cut-length sheets, but the grouping of several manufacturing steps in one operation makes possible certain economies in production and an overall improvement in control of product quality.

Several designs of continuous hot-dip galvanizing lines are in commercial use. In the simplest arrangement a continuous strip of box annealed and temper rolled steel is passed from an uncoiler through a long muriatic (hydrochloric) acid pickling tank into a galvanizing pot. The strip enters the pot through a layer of flux and regular spelter is used. The coated strip is cooled and sheared on the line to cut-length sheets. It is then inspected and prepared for shipment.

A more elaborate design utilizes the so-called "dry-fluxing" practice. In this modification a thin film of flux in aqueous solution is applied to the strip after it leaves the cleaner and the strip then passes through a drying furnace into the zinc pot. The cleaning, if necessary, can be acid or alkaline. The furnace is operated to deliver heated strip to the coating pot and thus supplies a portion of the heat required to maintain the bath at temperature. Aluminum-bearing spelter is used as the coating metal. The coated product may be re-coiled, or it may be sheared to cut-length sheets.

Another line is characterized by a continuous anneal during which the strip is exposed to a complex gas containing hydrogen chloride. The strip passes through the annealing furnace, a cooling compartment, and a muriatic (hydrochloric) acid bath before entering the flux box of a conventional galvanizing pot. Asbestos wipes are normally used instead of exit rolls on the discharge end of the coating pot. Coatings produced are very thin but are extremely adherent. Regular spelter is used.

Senzimir type lines include controlled oxidizing and reducing steps prior to coating. The cold-reduced strip passes from the uncoilers through an open-flame oxidizing furnace, which also serves as a flame-degreaser, into a reducing furnace where it is annealed or normalized and the oxide film formed in the first furnace is reduced. The strip is cooled in the rearward

zones of the reducing furnace to about the temperature of the coating bath and then passes into the zinc pot through a conduit extending from the end of the furnace to slightly below the surface of the coating bath. Cracked ammonia, introduced into this conduit to flow counter-current to strip movement, provides the necessary reducing atmosphere of the furnace. The degree of oxidation is said to be critical and must be controlled to achieve a uniform film ranging from pale yellow to blue or grey in color. Aluminum-bearing spelter is used.

In the continuous-galvanizing installations at the several United States Steel mills, strip material is alkaline cleaned, rinsed and dried, bright annealed, cooled to slightly above pot temperature and introduced into the coating bath while still protected by the furnace atmosphere. Between 0.12 and 0.18 per cent aluminum is maintained in the coating bath. Figure 39—12 is a schematic side-elevation of a line of this type designed to process 0.010 to 0.050 in. thick light-gage material at speeds up to 300 ft. per min. When coating 28-inch wide 29-gage strip at a speed of 250 ft. per min., the line can produce about 12.5 tons of galvanized sheets per hour. Figure 39—13 is a photograph of the input portion of a line designed primarily for coating heavy-gage material. Operational details are discussed below.

The starting material, which may be, for example, 5,000 to 50,000-lb. coils of full-hard cold-reduced strip steel, is fed from an uncoiler either of the single-mandrel or of the double-mandrel type, directly through pinch rolls to a squaring shear. A leveler may be used to flatten the head end of the strip prior to shearing. From the shear, the strip advances to a welder where the tail end of the coil being processed is attached to the head end of the new coil. Lap, butt or mash welders are used for this operation.

Following the welder, the strip enters the cleaning unit where residual cold-mill lubricant and any mill dirt picked up by the surfaces of this strip in prior processing or storage is removed. Dipping and scrubbing with a hot alkaline solution is used for this purpose on some of the lines; electrolytic alkaline cleaning on the others. Polarity of the strip during electrolytic cleaning, the nature and concentration of the cleaning agent and other cleaning conditions vary. Cleaning is

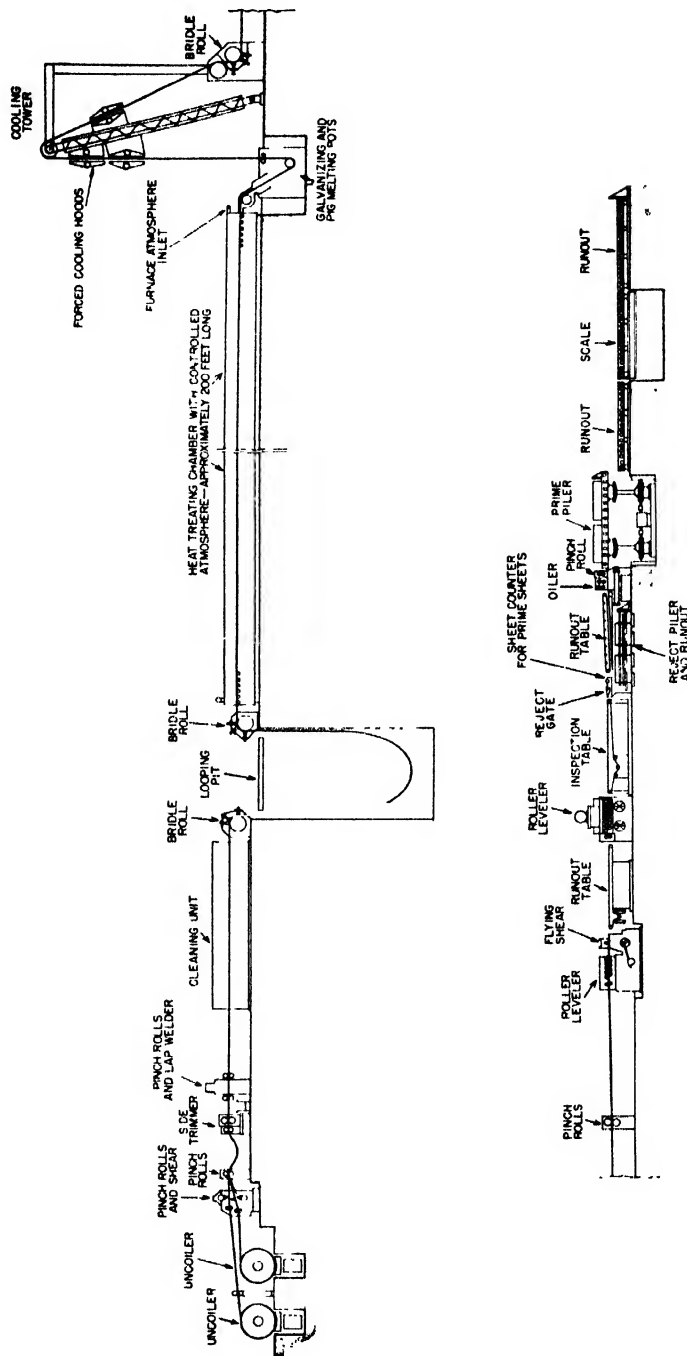


FIG. 39-12. Schematic arrangement (not to scale) of the units comprising a modern continuous galvanizing line for applying zinc coatings to cold-reduced light-gage strip steel. The steel is fed from the uncoilers into the line, successive strips being welded at their ends to provide an uninterrupted feed. The top part of the diagram shows in succession (left to right) the feeding, heat treating and coating stations. The bottom part shows the finishing end of the line.

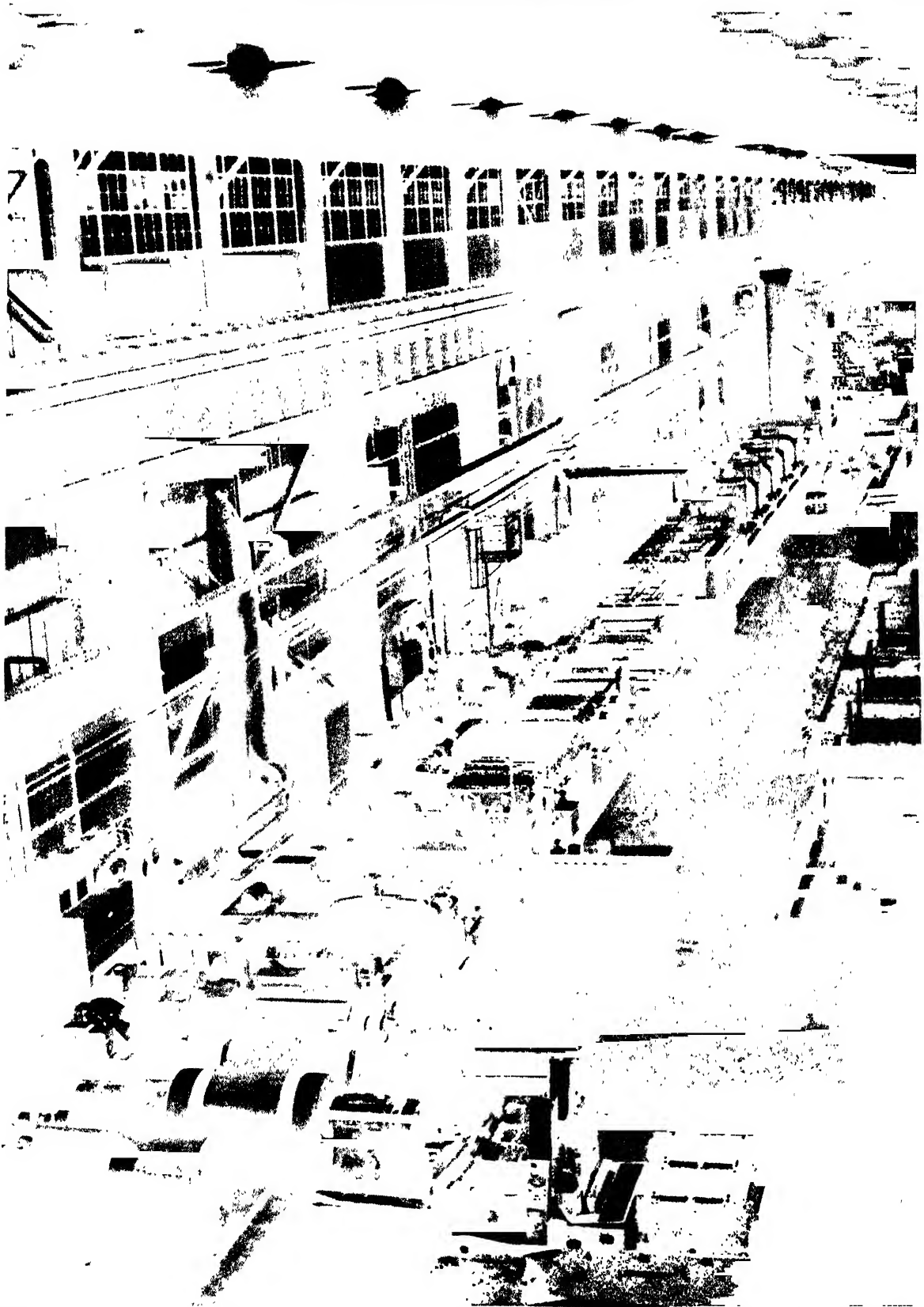


FIG. 39-13. A modern, continuous galvanizing line for applying zinc coatings to cold-reduced strip steel in coil form. This particular line was designed to handle heavy-gage material.

followed by rinsing and drying. The strip is moved through the foregoing equipment by the drive bridle, at the end of the cleaning station, which feeds the clean, dry strip into a slack accumulator. A looping pit, 30 to 90 feet deep, as shown in Figure 39—12 is usually provided for this purpose; where excavation is impractical a tower-type or a horizontal car-looper is used. The speed of the drive bridle is automatically controlled to maintain a desired length of loop in the pit and the slack so stored permits the input end of the line to be stopped for the joining operations without affecting the operation of the other sections of the line.

Strip is pulled out of the looper by a tension bridle and passes into the annealing furnace. The latter is heated either by gas or electrically and is divided into several independently controlled zones that can be operated in such a manner as to permit the strip to attain the desired annealing temperature in the furnace and yet enter the coating metal at but slightly above bath temperature (thus maintaining the coating bath at operating temperature). A mixture of cracked ammonia and NX gas is maintained within both heating and cooling sections of the furnace to prevent oxidation of the strip in process. The discharge end of the furnace extends below the surface of the molten zinc and the gases, which may be introduced at several points along the length of the furnace, flow counter to strip movement to escape at the entry end of the furnace. Furnace atmosphere is maintained at a positive pressure of $\frac{1}{2}$ to 1 inch of water. A considerable portion of the gas is introduced near the end of the last cooling zone to aid in cooling the strip and to insure a non-oxidizing environment for the strip and zinc at their point of initial contact.

When starting operation, the metal in the coating pot is melted either with gas or electric heat. As stated earlier, the pot is kept at operating temperature by heat

that is carried into the bath of molten metal by the strip.

The coating pot contains a sinker roll for submerging the strip in the bath, as well as exit rolls for controlling the thickness and distribution of the coating. The exit rolls are driven and are mounted to operate about half-submerged in the zinc. As in sheet practice, weight of coating is controlled by adjusting the pressure between the rolls and the relative position of the rolls with respect to the level of zinc in the pot.

From the exit rolls, the strip passes vertically upward to a large-diameter roll mounted at the top of a cooling tower. Forced-air blasts are directed against the strip to assist in solidifying the coating before it reaches the tower roll. Air blasts may also be used on the downward pass.

The coated strip, after returning downward from the tower, passes through a drive-bridle assembly that furnishes the necessary tension to pull the material through the annealing and coating units. Space is usually provided immediately ahead of this drive unit for any special treating facilities, provision being made to by-pass or withdraw such apparatus when not in use.

At the finishing end of a continuous galvanizing line, operations are usually of such a nature as to require occasional interruption. For this reason, looping facilities similar to those described for the entry portion of the line generally are provided. For producing coiled galvanized strip, a roller leveler and recoiler may follow the looper. For producing cut-length sheets, a flying shear equipped with suitable pinch-roll and leveling equipment is used after the looper. At the discharge end of the line, a sheet leveler may be used to flatten the sheets before preparing them for shipment.

Various automatic devices are installed in the finishing section to facilitate handling and assorting; these include automatic reject and prime pilers following the inspection table.

SECTION 5

TESTING GALVANIZED SHEETS

The weight of coating on a galvanized sheet is determined by weighing a 2¼-inch square test piece or a 2.539-inch diameter circular-disc test piece, dissolving the coating in hydrochloric acid containing antimony chloride as an inhibitor, and reweighing the washed and dried test piece. The weight loss in grams is numerically equal to the coating weight in ounces per square foot of sheet. Triple-spot tests selected from the center of the sheet and from two diagonally opposed corners may be used for sheet material, and edge-center-edge spot tests may be used for the continuous product. It is customary to obtain edge spot tests two inches inward from the actual edge of the sheet or strip.

Bend tests are sometimes used for determining coating adherence and base-metal formability, especially in heavy-gage galvanized sheets. The bend, which may be made in a vise or punch press, is generally a full angle of 180° around a specified number of sheet thicknesses of the same gage as the test piece.

The **beading test** is commonly used for testing coating adherence in light-gage material. The width, depth, and radii of the beads vary somewhat throughout the industry, although definite standards have been established by individual operators. In conducting the beading operation, the impressions may be made to cross each other or run straight along the edge of the test piece, depending upon the severity of the test desired.

The **lockseam test** is used quite extensively in the sheet-galvanizing industry for testing coating adherence. For sheets that are to be used for seaming, the test also serves as a check on the formability of the base metal.

Tensile properties of the base metal are determined by the standard tension test using an 11/16-in. by 8-in. specimen machined in the center to a 0.5-in. width. In this test the yield point, ultimate strength and percentage of elongation in a 2-in. gage length are measured.

The **Rockwell "B" hardness** of the base metal is determined by subjecting a stripped sheet to the standard Rockwell testing procedure.

The **Olsen cup test** can be used to obtain an indication of the drawing properties of the base metal as well as the adherence of the coating.

Ferritic grain size can be determined by polishing and etching a longitudinal section of the base metal and comparing the microscopic grain structure with that of the proper standard grain-size chart.

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(Continued on next page)

Chapter 40

THE MANUFACTURE OF STEEL WIRE AND STEEL WIRE PRODUCTS

SECTION 1

HISTORY AND IMPORTANCE OF STEEL WIRE

Historical—In general, wire is a term that may be applied to any metallic shred, thread, or filament, or to any exceedingly slender rod or bar of metal having a uniform cross section. Considering the term in this sense, wire is of very ancient origin. Gold wire is mentioned in the Bible in connection with the sacerdotal robes of Aaron, and was used to form part of a necklace, found at Denderah, which bears the name of the Pharaoh who reigned in Egypt about 2750 B.C. Evidences of the use of wire by the Assyrians and Babylonians as far back as 1700 B.C. have also been found.

As to the methods of manufacture used by these ancient peoples, practically nothing is known. The specimens of ancient wire that have been found so far are flat, and it is probable that they were produced solely by hammering.

How, when or of what material round wire was first made is not known. No doubt it was first produced by hammering, but the difficulty of forming a fine round wire by this method probably caused the metalworkers, early in the history of the art, to seek some better method. An old Latin manuscript written by Theophilus sometime during the 8th and 9th centuries describes a method of making wire of an alloy of lead and tin, which was cast into an ingot, hammered into a long slender bar and drawn through holes in a wire-drawing plate.

The draw plate, itself, is described as "a piece of iron three or four fingers wide, smaller at the top and bottom, rather thin and pierced with three or four rows of holes through which wire may be drawn." However, there is reason to believe that the art of wire drawing was known and practiced before this time.

The chain armor used by the knights in the great crusades against the Saracens and Turks is thought by many to have been made of drawn wire. Other authentic records are available to show that wire drawing on a commercial scale was practiced in France in 1270, in Germany in 1350 and in England in 1465. The first wire-drawing mill in this country was built in 1775 by Nathaniel Miles at Norwich, Conn. But, though three other mills, two of which were in Pennsylvania, were started during the next twenty years, none of these appear to have been very prosperous, for in 1820 practically no wire was being made in this country. In 1831, however, the industry was reestablished in this country by Ichabod Washburn of Worcester, Mass., who, with Benjamin Goddard, founded the firm of Washburn, Moen and Company.

Present Importance of Steel Wire Industry—From that date, 1831, the wire industry grew very rapidly. Aside

from the great quantities of wire made from other metals, such as copper and brass, the present annual production of steel wire alone is astonishing. For normal years, the production of steel wire rod now is about 5,500,000 tons. This tonnage represents about one-twentieth of all the steel produced in the country, and if it were all drawn into No. 12 gage wire, which is about average, it would have a total length of more than 70,000,000 miles or enough to girdle the earth about 2,800 times. The causes contributing to this growth were many and cannot be discussed in detail here, but they may be grouped or classified about as follows: (1) The great number of purposes to which wire, particularly steel wire, is adapted. (2) Development of inventions requiring great quantities of wire. (3) Improvements in the methods of producing steel and steel wire. It need not be pointed out that these causes are interdependent.

New demands stimulated efforts to improve the methods of manufacture, and as these improvements resulted in the bettering and the cheapening of the product, new uses were found for it.

Principal Uses of Steel Wire—The adaptability of wire to the manufacture of household articles of common use has expanded until it has become known as the steel product of 150,000 uses. These articles, including such items as pins, needles, brooms, strainers, hooks, egg beaters, toasters, etc., are almost indispensable to our comfort. Of the many inventions developed since 1820 that make use of large quantities of wire, only a few of the most important can be mentioned here. The first of these was wire rope. Rope made of bronze or brass wire is known to be of ancient origin, while iron wire rope was used in Europe as early as 1820. The first wire rope manufactured in this country was made in 1840 at Saxonburg, Pa., by John A. Roebling, founder of the firm of John A. Roebling's Sons Company. In 1844 the first telegraph line, extending from Washington to Baltimore, was completed by Professor Morse, but for more than ten years the growth of the telegraph was hampered because the wire essential in the construction of the lines could be produced only in short lengths, which required much time and labor for splicing.

The use of crinoline wire, a high-carbon heat-treated wire, for hoop skirts has been cited by some as a novel use of the product and as a great boon to the industry, but is better as an example of the unstable character of a business that depends on fashions. Starting about 1860, this business, for the next ten years, consumed about 1500 tons of wire annually, but suddenly collapsed with the passing of the hoop skirt in 1870. This was not to be the last time that feminine fashions would play a part

in the wire industry, for in 1916 some 15,000 tons of wire were being consumed annually in the manufacture of corsets and several hundred tons for hair pins, but these have largely disappeared as did the crinoline wire.

The collapse of the crinoline-wire business was replaced by more staple articles, for in 1868 the first patents were obtained on the use of wire for bale ties and barbed wire for fence. Both these articles came quickly into wide use, the latter article being especially useful in fencing the great unwooded lands west of the Mississippi River, which were rapidly being brought under cultivation about this time. By 1876 the development of the telephone had reached the practical stage, and began to create a demand for more wire of the same type used by the telegraph. It was in 1879 that the beneficial effect of sulphur on machinability was discovered, and the manufacture of high-sulphur screw stock was begun. Chas. H. Morgan also began the manufacture of coiled-wire springs in this year. In 1884 woven-wire fence was first made by machinery. This fence, of which there are now several types, fulfilled all needs that could not be met with barbed-wire or straight-wire fence and possessed all the advantages of any fence that could be constructed of wood, hence it was soon being used in all parts of the country. In 1851 the first American-made wire-nail machine was built by Thomas Morton; in 1875 the first steel-wire nails were made; and in 1888 the production of the wire nails exceeded that of the cut nail for the first time. Today, about twenty times as many wire nails as cut nails are made, and about one-fifth of all the steel wire produced is made into nails. Since 1890 no inventions making such unusual demands upon the wire industry have appeared, so that these items, with the two or three exceptions noted, remain the backbone of the steel-wire industry. However, innumerable items employing steel wire in smaller quantities have been developed, which, added to the rather rapid growth of these principal lines, have contributed to a continual expansion in the industry.

The extent of this expansion can be judged from the fact that the production of steel wire in 1923 was more than twice that for 1903.

Early Method of Manufacture—The methods employed at the time of the introduction of wire drawing into this country were wholly inadequate to meet the demands that were soon to arise. As the Bessemer and open-hearth processes for making steel were then unknown, practically all wire was made from wrought iron, or the softer metals like copper, brass, bronze, etc., though crucible steel in softer tempers was also used. Practices varied somewhat in different localities, and detailed descriptions of the operations are lacking but, from information gleaned from different sources, the process and operations for drawing wrought-iron and steel wire appear to have been about as follows: Starting with a small square bar or billet, the metal was heated and hammered into as small a round as practicable, say $\frac{1}{4}$ to $\frac{3}{8}$ inch in diameter and some 6 to 10 feet in length. One end of this rod was tapered to a size a little less than the hole in the die through which the rod was to be drawn. The rod was then scoured bright with a piece of sandstone or other abrasive. In the case of iron and steel, it was early discovered that this cleaning of the rod was not only essential to good work but was well worth the trouble, for the hard scale, when not removed, soon scored the die. Then, the tapered end of the rod was inserted in the proper hole of a draw plate made of cast iron or hard steel, where it was grasped with a pair of tongs attached to a lever, and the rod was drawn through the hole a few inches at

a time, about 5 inches at each stroke being the common practice. The lever could be operated either by hand or by water power, and the rod, as it passed through the hole, was generally lubricated with butter or other grease, to facilitate the drawing and save the die as much as possible. After the first draft, the operation was repeated through progressively smaller holes in the draw plate until the wire was drawn to the size desired or until it became too hard and brittle for further drawing. If the wire was to be drawn to still finer sizes, it was annealed at a red heat, generally in an oven sealed off from the air, after which the thin coat of oxide thus formed was removed by tumbling with a mixture of sand and water, or by immersing the wire for some time in sour beer or fermented barley water. The machine for drawing this fine wire differed somewhat from that used for drawing the rod. The draw plate was mounted on a table or bench in front of a drum or block that could be revolved by a hand-operated crank and gear drive. After pointing, the end of the wire was inserted into the die and drawn through by tongs, until it could be attached to the drum. Then, by revolving this drum, the drawing of the wire could be performed in one continuous operation, the wire being wound about the drum and thus drawn through the die.

Improvements in the manufacture of iron and steel wire have kept pace with the demands. In connection with this statement, it should be noted that the basic principles of wire making have remained unchanged. As already indicated, the process consists essentially in drawing a rod or a slender piece of the metal through a tapered hole in some harder material, or successively through a series of such holes. It involves the application of force in the simplest manner, a straight pull, and the use of one of the simplest tools, the wire-drawing die. The efficiency of the latter and the simplicity of both these elements have left little room for improvement, and many doubt the possibility of finding a substitute for the process that will do the work so well. The improvements that have been made pertain to the accessory equipment and to the complementary operations, rather than to the process of wire drawing itself. Thus, the need for long lengths was met not by any radical change in the wire-drawing process, but by changing the method of producing the wire rod. As long as the rod was made by hammering, it was too short to coil, or draw with a block; but when the need for long lengths began to be felt, efforts were made to roll the rod. About 1840, Washburn succeeded in adapting the wire-drawing block to coarse-wire drawing. This block was power driven, and provided a means not only of applying a greater pulling force continuously to the rod, but also of storing the wire as it was drawn through the die. This development made it desirable to use very long rods, and this demand was met first by the invention of the three-high mill about 1857, which was succeeded by the Morgan continuous mill about 1875 and the Garrett rod mill in 1882.

Another important development was the substitution of soft steel for wrought iron, made possible by the invention of the Bessemer and the open-hearth processes for steelmaking. This change was inaugurated in this country by the Washburn and Moen Manufacturing Company at Worcester, Mass., in 1871. Important improvements in methods for handling, cleaning, coating, or lubricating the rod, and in heat treating and finishing the wire, were also made.

This brief historical review serves as an introduction to the study of the manufacture of steel wire as it is carried on today.

SECTION 2

CLASSIFICATION OF STEEL WIRE

Bases for Classification—Steel wire products are classified in various ways, as indicated in Table 40—I. From the standpoint of a manufacturer, the factors of greatest interest are the kind and grades of steel; the size, shape and mechanical properties of the wire; and the methods of drawing and treatment. In commercial transactions, classifications based upon mechanical properties of the wire, its treatment and finish, and its commercial applications are of most interest. The following brief descriptions are intended to assist the reader to a better understanding of the classes listed in the table.

Kinds and Composition of Steel Used for Wire—The steel for wire is produced by all the modern processes, including the basic open-hearth, Bessemer, and electric-furnace processes. Some special grades of wire are still made from Swedish wrought iron. Plain carbon steels chiefly are produced by the basic open-hearth, and Bessemer processes; mainly stainless steels and alloy steels for certain special-purpose wires are made in electric furnaces.

By properly selecting and handling the plain carbon steels in drawing and heat treating, it is possible to develop a wide range of mechanical properties. Ordinarily, sulphur and phosphorus are kept within the usual limits for each grade of steel, while the silicon, manganese, and carbon contents are varied according to the mechanical properties desired. Occasionally, the sulphur content will be increased to improve the cutting qualities (machinability) of the steel.

In recent years, there has been a constantly increasing demand for wire possessing properties that cannot be

obtained by cold working and heat treating plain carbon steel. This demand has been met by the addition of alloying elements, such as nickel, chromium, manganese, silicon, vanadium, tungsten or combinations of them. Many of these alloy steels can be produced satisfactorily by the basic open-hearth process. For those that cannot and for the stainless steels containing large percentages of chromium and nickel, electric furnaces of the arc type are commonly used and, to a decreasing extent, the induction type of electric furnace is sometimes used. Each of these processes produces steel having the best properties for certain purposes or kinds of wire.

Wire Shapes—While wire is ordinarily thought of as being round, i.e., with a circular section, it may have any one of an infinite number of sectional shapes, as required by the particular use for which it is desired. After the ordinary round wire of circular section, the most common shapes are square, hexagon, octagon, oval, half-oval, half-round, triangular and flat. Besides these regular, or symmetrical, shapes, wire is also made in various odd and irregular shapes for specific purposes.

The American Iron and Steel Institute (AISI) Steel Products Manual, Section 17, defines a flat wire as a cold-rolled product, with a prepared edge, rectangular in shape, $\frac{1}{2}$ -inch or less in width, under $\frac{1}{4}$ -inch in thickness. "Low-carbon steel flat wire is generally produced from hot-rolled rods or specially prepared round wire, by one or more cold-rolling operations, primarily for the purpose of obtaining the size and section desired and for improving surface finish, dimensional accuracy and varying mechanical properties. Low-carbon steel flat wire can also be produced by slitting cold-rolled flat steel to the desired width. The width-to-thickness ratio together with the specified type of edge generally determine the process which will be necessary to produce a specific flat wire item." Those edges, finishes, and tempers obtainable in flat wire are similar to those furnished in cold-rolled strip.

Sizes of Wire—The size of round wire is determined by its diameter, which is expressed in absolute units, or decimal parts thereof, or by gage numbers. In this country the absolute unit is the inch, and the diameter is obtained by a micrometer capable of making measurements accurate to at least one thousandth of an inch. As stated in Chapter 50 on gages, there are several different gages in effect for the measurement of wire. The size of music wire is expressed in gage numbers of a system known as the Music Wire Gage (M.W.G.), which is the standard for this wire. For iron and steel telephone and telegraph wire, the standard gage is the Birmingham Wire Gage (B.W.G.), while for copper electrical wire the Brown and Sharpe Gage (B.S.G.), also known as the American Wire Gage (A.W.G.), is largely employed. With these notable exceptions the gage used by the American Steel and Wire Division of United States Steel and by many other manufacturers of steel wire, is the United States Steel Wire Gage (U.S. Stl.W.G.), or the Steel Wire Gage (Stl.W.G.), and all unidentified gage numbers used in this chapter will refer to this gage, which is shown in Table 40—II.

The size limits for the product commonly known as wire range from approximately 0.005 inch to under 1 inch for round sections, and from a few thousandths of an inch to approximately $\frac{1}{2}$ inch for square sections. Larger rounds and squares, and all sizes of hexagonal and octagonal shapes are commonly known as bars.

Table 40—I. Kinds and Classes of Steel Wire

Classification, according to:

Method of making the steel—Basic open hearth, Bessemer, basic electric furnace, and acid electric furnace.

Compositions of steel—Low carbon (C1006, C1008, C1010, C1012, B1006 and B1010), medium low carbon (C1013 to C1022, inclusive), medium high carbon (C1025 to C1041, inclusive), high carbon and alloy steel.

Shape—Round, half-round, oval, half-oval, square, rectangular, hexagonal, crescent, triangular, etc.

Method of drawing—Dry-drawn, wet-drawn, single-draft drawn and continuous drawn.

Size—Common wire or coarse wire, fine wire.

Treatment and finish—Bright (hard dry-drawn), bright soft (annealed in process; processed wire), normalized (pre-normalized), spheroidized, extra smooth clean bright, sully coated, annealed, black annealed (pot annealed), strand annealed (lead annealed), bright annealed, lime bright annealed, patented, oil tempered, liquor finished (coppered or bright coppered wire), galvanized, tinned, aluminum coated.

Uses or commercial applications—Armor wire, bale-tie wire, bolt wire, broom wire, cold-heading wire, concrete reinforcement, fence wire, music wire, piano wire, poultry netting, rope wire, spring wire, telephone wire, telegraph wire, tire wire, welding wire, wool wire (for steel wool), special wires, miscellaneous wires.

Table 40—II. American Steel & Wire Gage
(Decimal Sizes and Feet Per Pound)

AS&W Steel Wire Gage No.	Deci- mally	Feet Per Lb.	AS&W Steel Wire Gage No.	Deci- mally	Feet Per Lb.	AS&W Steel Wire Gage No.	Deci- mally	Feet Per Lb.	AS&W Steel Wire Gage No.	Deci- mally	Feet Per Lb.	AS&W Steel Wire Gage No.	Deci- mally	Feet Per Lb.
1—	0.283	4.68	10—	0.135	20.6	19—	0.0410	223.0	28—	0.0162	1129	37—	0.0085	5189
1/4	0.278	4.85	1/4	0.131	21.9	1/4	0.0394	241.6	1/4	0.0159	1483	1/4	0.0084	5313
1/2	0.273	5.03	1/2	0.128	22.9	1/2	0.0379	261.1	1/2	0.0156	1510	1/2	0.0083	5443
3/4	0.268	5.22	3/4	0.121	24.4	3/4	0.0363	281.5	3/4	0.0153	1602	3/4	0.0081	5714
2—	0.2625	5.44	11—	0.1205	25.8	20—	0.0318	309.6	29—	0.0150	1666	38—	0.0080	5858
1/4	0.258	5.63	1/4	0.117	27.4	1/4	0.0310	324.3	1/4	0.0148	1712	1/4	0.0079	6007
1/2	0.253	5.86	1/2	0.113	29.4	1/2	0.0332	310.2	1/2	0.0145	1783	1/2	0.0078	6163
3/4	0.248	6.10	3/4	0.109	31.6	3/4	0.0325	355.1	3/4	0.0143	1833	3/4	0.0076	6491
3—	0.2137	6.31	12—	0.1055	33.7	21—	0.0317	373.1	30—	0.0140	1913	39—	0.0075	6665
1/4	0.239	6.56	1/4	0.102	36.1	1/4	0.0309	392.7	1/4	0.0134	1969	1/4	0.0071	6846
1/2	0.235	6.79	1/2	0.099	38.3	1/2	0.0301	413.8	1/2	0.0136	2027	1/2	0.0073	7037
3/4	0.230	7.09	3/4	0.095	41.5	3/4	0.0294	433.7	3/4	0.0131	2088	3/4	0.0071	7437
4—	0.2253	7.39	13—	0.0915	44.8	22—	0.0286	458.4	31—	0.0132	2152	40—	0.0070	7652
1/4	0.221	7.68	1/4	0.089	47.3	1/4	0.0279	481.7	1/4	0.0131	2185	1/4	0.0069	7874
1/2	0.216	8.04	1/2	0.086	50.7	1/2	0.0272	506.8	1/2	0.0130	2218	1/2	0.0068	8110
3/4	0.212	8.31	3/4	0.083	54.4	3/4	0.0265	533.9	3/4	0.0129	2253	3/4	0.0067	8352
5—	0.207	8.75	14—	0.080	58.6	23—	0.0258	563.3	32—	0.0128	2288	41—	0.0066	8607
1/4	0.203	9.10	1/4	0.078	61.6	1/4	0.0251	595.1	1/4	0.0126	2361			
1/2	0.200	9.37	1/2	0.076	64.9	1/2	0.0244	629.7	1/2	0.0123	2478			
3/4	0.196	9.76	3/4	0.074	68.5	3/4	0.0237	667.5	3/4	0.0121	2561	42—	0.0062	9753
6—	0.192	10.2	15—	0.072	72.3	24—	0.0230	708.7	33—	0.0118	2693	43—	0.0060	10115
1/4	0.188	10.6	1/4	0.070	76.5	1/4	0.0224	747.2	1/4	0.0115	2831			
1/2	0.185	11.0	1/2	0.067	83.5	1/2	0.0217	796.2	1/2	0.0111	3043	44—	0.0058	11145
3/4	0.181	11.4	3/4	0.065	88.7	3/4	0.0211	842.1	3/4	0.0108	3215			
7—	0.177	12.0	16—	0.0625	96.0	25—	0.0201	901.	34—	0.0104	3466	45—	0.0055	12394
1/4	0.173	12.5	1/4	0.060	104.2	1/4	0.0198	956.	1/4	0.0102	3605			
1/2	0.170	13.0	1/2	0.058	111.5	1/2	0.0193	1007.	1/2	0.0100	3749	46—	0.0052	13866
3/4	0.166	13.6	3/4	0.056	119.6	3/4	0.0187	1072.	3/4	0.0097	3985			
8—	0.162	14.3	17—	0.054	128.6	26—	0.0181	1144.	35—	0.0095	4154.	47—	0.0050	14997.
1/4	0.159	14.8	1/4	0.052	138.7	1/4	0.0179	1170.	1/4	0.0094	4243.			
1/2	0.155	15.6	1/2	0.051	144.2	1/2	0.0177	1197.	1/2	0.0093	4335	48—	0.0048	16273
3/4	0.152	16.2	3/4	0.0491	155.5	3/4	0.0175	1224.	3/4	0.0091	4527.			
9—	0.1483	17.1	18—	0.0475	166.2	27—	0.0173	1253.	36—	0.0090	4629.	49—	0.0046	17718.
1/4	0.145	17.8	1/4	0.0459	177.9	1/4	0.0170	1297.	1/4	0.0089	4733.			
1/2	0.142	18.6	1/2	0.0443	191.1	1/2	0.0168	1329.	1/2	0.0087	4953	50—	0.0044	19366.
3/4	0.138	19.7	3/4	0.0426	206.6	3/4	0.0165	1377.	3/4	0.0086	5069.			

Classification of Common Round Wire According to Size—As wire may include a wide range of sizes, its classification has been governed somewhat by manufacturing equipment and methods of handling. It normally is separated into two broad groups known as **coarse** and **fine** wire. The fine-wire classification is usually recognized as including 16 gage and smaller wire, normally produced in 8-inch diameter coils. Coarse wire includes sizes 20 gage and coarser, normally drawn on 16-inch, 22-inch and 26-inch or 30-inch blocks. The distinction is not clearly drawn for 16 to 20 gage, inclusive, as this range is commonly regarded as coarse wire for some end uses, and as fine wire when made by a different manufacturing practice for other end uses. The most commonly used wire sizes are those drawn from a 7/8-inch rod to 8 to 20 gage, inclusive.

Surface Finishes of Wire—A variety of finishes on the wire may be obtained by controlled processing in manufacture. The more common finishes are usually designated as **bright**, **black annealed**, **liquor finished**, **coppered**, **tinned**, **galvanized**, **aluminum coated**, and **painted** or **enameled**. For certain specific uses, a drawn alumi-

num coated or a drawn galvanized finish is sometimes produced, by cold drawing the wire one or more drafts through a die after coating. The bright finish is obtained by dry drawing, liquor finish by wet drawing, black annealing finishes by oxidation of the surface in heat treatment, while the tinned, galvanized and painted finishes are produced by subjecting the wire to a coating process as a final operation. The methods used in producing these different finishes will be developed more fully in the description to follow.

Temper of Wire—In the wire industry, temper is a word used in referring to the hardness, stiffness and strength of wire. The temper is affected by the amount of carbon, manganese and phosphorus, or other alloying element present in the steel, the amount of cold drawing without annealing, and the heat treatment given the wire. Thus, increasing the carbon content of the steel or submitting the wire to drawing operations increases the temper, while heat treating may either harden or soften the metal, according to whether it is heated to the hardening-temperature range and quenched, or annealed. The strength and hardness of low-carbon,

medium-low carbon and medium-high carbon wires are regulated by drawing and annealing operations, and of high-carbon wires by drawing and patenting operations. The tempers of low-carbon wires are usually expressed as hard, medium hard, bright soft, soft and extra soft. Thus the **hard temper** is obtained by drawing through several dies in succession without annealing to obtain considerable reduction, **medium hard** by drawing (usually fewer drafts) to obtain somewhat less reduction or, in some instances, by subjecting hard temper wire to a partial or "slack" annealing, **bright soft** by giving the wire one light pass through the die after annealing, and soft and extra soft by annealing after drawing to size. Thus it will be seen that by properly correlating all these factors—composition, drawing, and

heat treatment—the mechanical properties of steel wire may be varied over a wide range of hardness, strength, toughness, ductility, or softness. As an illustration of the possibilities in varying these properties, there may be cited the strength of steel wires, which are regularly made with tensile strengths varying from 50,000 to as high as 500,000 pounds per square inch for the smaller gages of high-carbon wire. It must be remembered, however, that, with but one exception, whatever is done to strengthen the metal will decrease its ductility, and vice versa.

The exception refers to the special heat treating process known as patenting, which may, under certain conditions, increase both the strength and ductility of the

SECTION 3

ROLLING THE WIRE ROD

The Wire Rod—The slender rods or bars of metal from which wire is drawn are known as wire rod, and, as already indicated, all steel wire rod is now produced by hot rolling. It may be made in various shapes and sizes, but for common wires and fine wires, i.e., No. 8 and finer, $\frac{7}{32}$ -inch round rod may be considered as the standard. This size rod is the smallest round that is practicable to produce on a rolling mill and, while the diameter may vary, it is nominally 0.218 inch. For large wires the rod will, of course, be somewhat larger than the wire to be made from it. As the rod comes from the rolling mill, it is wound into coils. These coils are usually about thirty inches in diameter inside and weigh from 300 to 1200 pounds each. Each coil represents the rod made from a single billet, and its weight, therefore, is determined by the weight of the billet used. While the wire rod represents a finished product of the rolling mills, it constitutes the raw material used in the wire mill and should be considered as the first step in the making of wire, for upon the kind and quality of the rod depend largely the kind and quality or grade of wire drawn from it. Physical defects in the rod will either cause failures in drawing or remain as more minute defects in the wire. While some wire mills obtain their rods from outside sources, many mills, especially the larger plants, roll their own rods.

Types of Rod Mills—After it has been made according to specifications, the steel for wire is cast into ingots and rolled on the blooming mill in the usual manner. In some cases the blooming mill is succeeded by a billet mill, and the blooms are rolled into billets $2\frac{1}{2}$ inches square or less in size. Frequently the product of the blooming mill is a 4-inch by 4-inch billet, which is cut into lengths to give the weight desired, allowed to cool, inspected, reheated, and then rolled directly into rods. As pointed out in the first section of this chapter, the demand for wire in exceedingly long lengths and in large quantities has led to the development of special mills for rolling the rods. These mills are of four different types, known as: (1) the Morgan, or continuous, rod mill, (2) the Garrett, or looping rod mill, (3) the combination, or combined continuous and single Belgian, rod mill, and (4) the double Belgian rod mill. Of these the first two employ fundamentally different principles of roll arrangement, hence will be described somewhat in detail.

The Continuous Rod Mill represents a development which in this country began about 1870. At that time the mill most generally employed for rolling rods was the **single train Belgian, or looping, mill**. This mill, while

it represented a great improvement over the single-stand three-high mill, was comparatively inefficient and was limited as to the length of rod it could roll. Since all the rolls were run at the same speed, the billets could be roughed down much faster than the rod could be finished, and, owing to the loops formed and the length of time required for the piece to pass through the finishing rolls, the rod would become too cold to finish if its length exceeded certain limits. In 1839, however, the Washburn and Moen Manufacturing Company erected at their Grove Street Works, Worcester, Mass., an entirely new type of mill, which had been patented in 1862 by George Bedson of Manchester, England. This mill consisted of a number of pairs of horizontal rolls and an equal number of vertical rolls arranged in a series so that the first, third, and succeeding stands of odd number were horizontal, while the second, fourth, and other stands of even number were vertical. This arrangement of the rolls made the draft in any stand at right angles to that in the succeeding stand, and overcame the necessity of giving the piece a quarter turn between passes, as in the looping mill. Then, by placing the stands as close together as possible, he was able to drive the mill through two long shafts, on each of which was mounted a system of gears, whereby the rolls of each stand beyond the first was made to revolve at a rate definitely faster than the stand preceding. This system of driving was to regulate the speed of the different stands so that the peripheral speed of the rolls in each would be the same, or nearly the same, as the linear speed of the bar, which increases after each pass due to the elongation. In this way the looping of the rod between passes was avoided, and a billet once started in the first stand was rapidly carried through the mill in a straight line, being thus rolled into a rod in one continuous operation. This mill met all expectations as to speed and length of rods rolled, but its output was restricted at first, because the rods could not be coiled and taken out of the way as rapidly as the mill could produce them. But C. H. Morgan, who was then general manager for Washburn and Moen, soon overcame this difficulty by devising a power traction reel for coiling the rods. This was the beginning of the development of the automatic reels, which will be described later.

The Morgan Mill—While the Bedson mill represented a great advance in rod rolling, it possessed certain mechanical features which were objectionable. Its chief faults were found to be due to the vertical rolls. They were not easily kept in adjustment, and the shaft and

gears for driving them were all beneath the mill floor. The scale and water from the rolls fell upon these gears and bearings and caused excessive wear, which, combined with the difficulty of getting to them under the floor, made them a source of much trouble. After a few years' experience with the Bedson mill, therefore, Morgan and his associates developed the twisting guide, by means of which the vertical rolls could be eliminated and horizontal rolls substituted for them. This was a closed delivering guide in which the grooves were cut in a spiral, so that with this guide properly mounted after any pass the piece was forced through it and twisted a quarter turn before it entered the next pass. Besides overcoming most of the disadvantages of the Bedson mill, this Morgan plan presented the additional advantage of more than one rolling line, that is, of rolling two or more rods side by side at the same time and on the one set of rolls. This plan of increasing the capacity of the mill, however, increases the difficulty of keeping such mills in adjustment, and was not taken advantage of until some years after the erection of Morgan's first mill. Even with a single rolling line, the development of the mill to a practical basis of operation presented some difficult problems. It is evident that such a mill requires very fine adjustment of the draft, roll diameter, and speed of rotation, for these factors are brought into close relation. The relations of all these factors can be determined by proper mathematical calculations, but as the draft fixed by the grooves in the rolls is affected by the adjustment of the rolls, the temperature of the bar, the composition of the steel, and changes with the wearing of the grooves and bearings, much difficulty was at first experienced in getting the mill adjusted so that the piece would not loop or jam between the different stands of rolls. This was particularly true in the case of the last few roll stands of the high-speed finishing mill. This difficulty was finally solved by increasing the speed of the rolls enough to keep the rod under slight tension at all times, and by making the bottom as well as the top roll adjustable as explained under the heading of continuous billet mills.

Modern Continuous Rod Mills—Since the time of Morgan's first mill, various improvements have been made, not so much in the mill itself as in the auxiliary equipment. These improvements have made it possible to produce a more nearly perfect rod in greater and greater tonnages. The delivery speed of modern con-

tinuous mills rolling $\frac{7}{32}$ -inch rod ranges from 3700 to 6000 feet per minute. The actual output of a mill depends upon conditions and the number of rolling lines, which was formerly one or two, and in recent years has been increased to three or four. The number of stands of rolls in a mill and, to a considerable extent, the general layout of the mill, depends upon the size of the billets used and rod size to be rolled therefrom. As previously pointed out, there are several standard sizes of billets, $1\frac{3}{4}$, 2, $2\frac{1}{2}$, and $3\frac{1}{4}$ inches square.

Layouts for Rolling Small Billets—From the standpoint of general efficiency, accuracy, and uniformity in size of the product, continuous rod mills are designed, preferably, to roll billets from $1\frac{3}{4}$ to $3\frac{1}{4}$ inches square and approximately 30 feet in length. This length fits the former standard railroad freight car, while the section is about the largest that can be rolled satisfactorily without a reheating or increasing the speed of the mill beyond the limits of safety and practicability. To work these sections into a $\frac{7}{32}$ -inch rod (0.218-inch) without unduly increasing the danger of developing rolling defects, requires from 16 to 23 passes. For straight continuous rolling, that is, where the rolls are all in the same straight line, these passes are split into two groups, known as the roughing and the finishing sets (Figure 40—1). The roughing set consists of 6 to 9 stands of rolls, the first of which is set as near to the discharge door of the heating furnace as possible, usually within 6 to 12 feet. Since the furnace is of the continuous side-discharge type, this arrangement makes it possible, without removing the billet entirely from the furnace to push one end into the rolls, which then withdraw it no faster than it is being rolled. Immediately following the last roughing stand is a flying shear, which permits the cropping of bad ends before the billet enters the high-speed finishing sets. To provide space for these and also for a little slack between the two groups of rolls, so as to make it easier for the roller to keep the mill in proper adjustment, the first stand in the second group is placed about 20 to 30 feet from the last roughing stand. The second group of rolls contains 10 or 12 stands, the last of which is known as the finishing stand. Following the finishing stand, tube-like pipe guides are provided for conducting the rod to the reeling machines which are located some 20 to 60 feet beyond and usually number twice the number of rolling lines or strands. The chief advantages in this arrangement lie in its

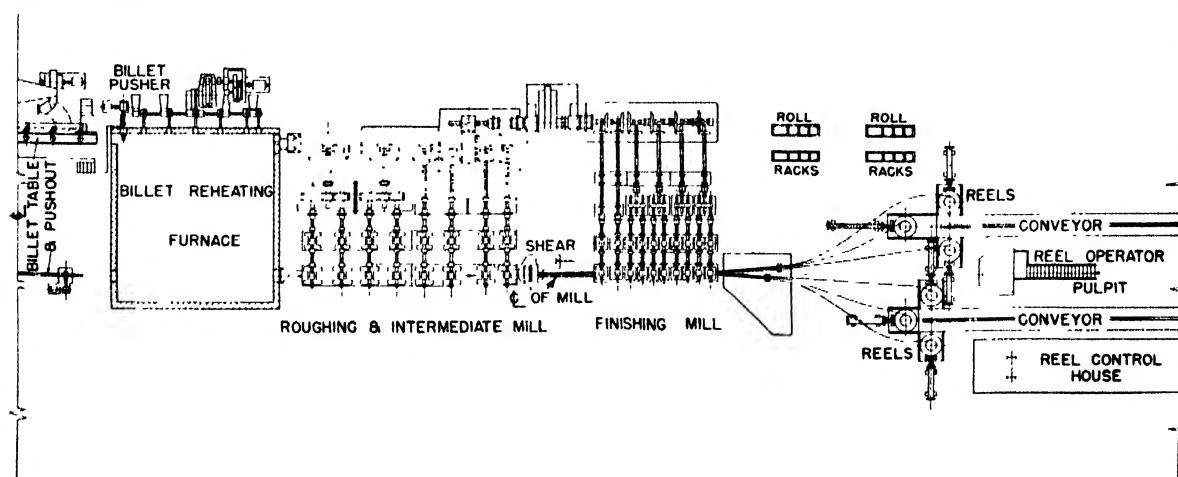


FIG. 40—1. Plan and layout of continuous rod mill for 2-inch billets, including reels and part of the rod bundle conveyor.

compactness, which aids in maintaining uniformity of rolling temperature and provides for a choice of different methods of driving. The control of temperature possible is evident from the fact that at the time the forward end of the rod reaches the reels, about one-half of the billet is still in the furnace. For driving the mill, two motors, one for each group of rolls, may be used, but some mills of this type are driven through two shafts connected by gears to a single motor or steam engine.

The Looping Continuous Mill—To provide greater flexibility and make it easier to keep the mill adjusted, in some recent installations of continuous rod mills, the finishing group of rolls also has been split up into three smaller groups of four, two and four stands each. The two-stand group is both preceded and followed by 180-degree loops. The roughing rolls and the first four intermediates are in the same straight line, and driven in the usual way by a single motor, while the next two groups, driven by another motor, stand in two separate lines parallel to the first line, with the two-stand group a little in advance of the finishing stands. Since the roll speeds are adjusted to the elongation of the piece being rolled, the loops, once formed, increase very little in size, and cooling of the rod through this exposure is very slight and uniform for each unit of its length. Also, this placing permits ironing out some of the effect of tension in the early part of the mill.

Layouts for Rolling 4-Inch by 4-Inch Billets—Although the continuous mill is best adapted to roll small long billets, local conditions frequently make it desirable or necessary to start with a larger size of billet. To meet such conditions, many continuous mills have been designed to use 4-inch by 4-inch billets. In designing these mills, the engineers have developed two plans. In one of these an additional group of roughing or breaking-down rolls is provided, and the rod is rolled directly from the billet, as in the straight continuous plan for small billets. The plan has some drawbacks, the chief of which may be noted as follows: If all the rolls are speeded to correspond to the elongation of the bar, the speed of the first roughing group is very slow. Even with the finishing roll revolving at the rate of 1350 r.p.m., the speed of the first roughing rolls, with a roll diameter of 14 inches, would be close to 3 r.p.m. The long time the piece is thus kept in contact with the cold rolls results in cooling it to a point where it becomes difficult to finish. If the first roughing group is speeded up, then the piece must be held ahead of the first intermediates and is cooled even faster and more unevenly than in the former instance. In some cases a solution to this difficulty has been sought by placing a heat retainer, a long narrow brick chamber heated by gas, between the roughing and intermediate groups. By passing the billet through this chamber slowly some of the heat lost in the previous rolling may be restored before it enters the intermediate sets of rolls. But the best plan involves the use of a reheating furnace between these two groups of rolls. In this plan, which is in effect equivalent to placing a billet mill before the usual straight continuous rod mill, the first group of rolls, consisting of several stands driven by a separate motor or engine, breaks the 4 by 4 by 72-inch billet, for example, down to a more suitable size billet about 30 feet long. This long billet is immediately charged into the continuous reheating furnace, where its rolling temperature is fully restored before being rolled to finished size. From this point the arrangement of the mill does not differ materially from the straight continuous plan already described.

Operation of Continuous Mills—With the mill prop-

erly designed and erected, its efficient operation depends largely upon the skill of the operating crews, headed by the roll turner and roller. For best results the roll diameters for the different stands must be carefully selected and maintained in proper proportions; the roll passes, especially for the finishing group of rolls, must be skillfully adjusted and accurately turned; the rolls and guides must be carefully set in the housings; and thereafter the draft must be regulated through the mill screws to suit the conditions. The uniform heating of the billet is also a matter of importance, as is seen from the following facts: Under a given pressure, a hot bar or rod will elongate more than a colder one. Translated into operating terms, this statement means that if the mill is adjusted to roll the colder rod properly, the hotter rod will buckle between passes and cause cobbles. The mill must, therefore, be adjusted to the hottest bars, and if the draft on the different passes is regulated to keep this bar under slight tension the same adjustment will produce a pull of considerable force on the colder bars. Thus pull is the cause of the longer "finny" ends which are characteristic of rods rolled on straight continuous mills, and it also produces a difference in section between the ends and the middle of the rod, causing it to exhibit a tendency toward flatness in the middle. This matter is especially important in the case of mills rolling more than one line, or strand, at a time. In such a mill, any adjustment of the screws to correct a fault in one pass affects the draft on the corresponding pass of the other strand also, and it is practically impossible to overcome any variation in temperature of the two strands through adjustment of the rolls only.

The Garrett Mill—Up to 1882 the continuous rod mill had no competitor as to speed, length of rod, and tonnage produced. On the Belgian, or looping mill, which was the only other type of mill used for rolling rods, the longest lengths that could be rolled would not exceed 300 feet, the heaviest coils weighed less than 50 pounds, and the tonnage produced daily would not, in case of small rod under 0.2-inch diameter, much exceed 20 tons. On such mills the best rolling practice of that day for steel involved the rolling of the ingots to 4-inch by 4-inch blooms, which were cut into lengths convenient for hand rolling and allowed to cool. These blooms, or billets, were then reheated and rolled on a three-high billet mill to a 2-inch square or its equivalent in rectangular section. These small billets were again sheared into lengths to give the weights required and allowed to cool, when they were reheated and rolled into rods on the looping mill. In endeavoring to overcome the advantages enjoyed by the owners of the continuous rod mill, William Garrett, who was then superintendent of the plant of the Cleveland Rolling Mill Company, conceived a plan whereby the looping mill could be modified to roll No. 5 rod in long lengths and direct from 4-inch by 4-inch billets without reheating. In formulating this plan, he reasoned that, since a large section loses heat less rapidly than a small one, the 4-inch by 4-inch billet could be rolled in the roughing mill to a size suitable for the rod mill in so short a time that it would still retain enough heat to be finished if it were not exposed so long in the looping mill. So he combined the three-high billet mill with the rod mill, and split up the latter into three groups or trains of rolls, which he arranged in echelon and drove at progressively increasing rates, so as to rough at low and finish at high speeds. By this arrangement the speed of the finishing trains could be adjusted to the elongation of the rod, so that not only could a given length be rolled in a shorter period of time, but the size of the loops could be controlled and their length reduced.

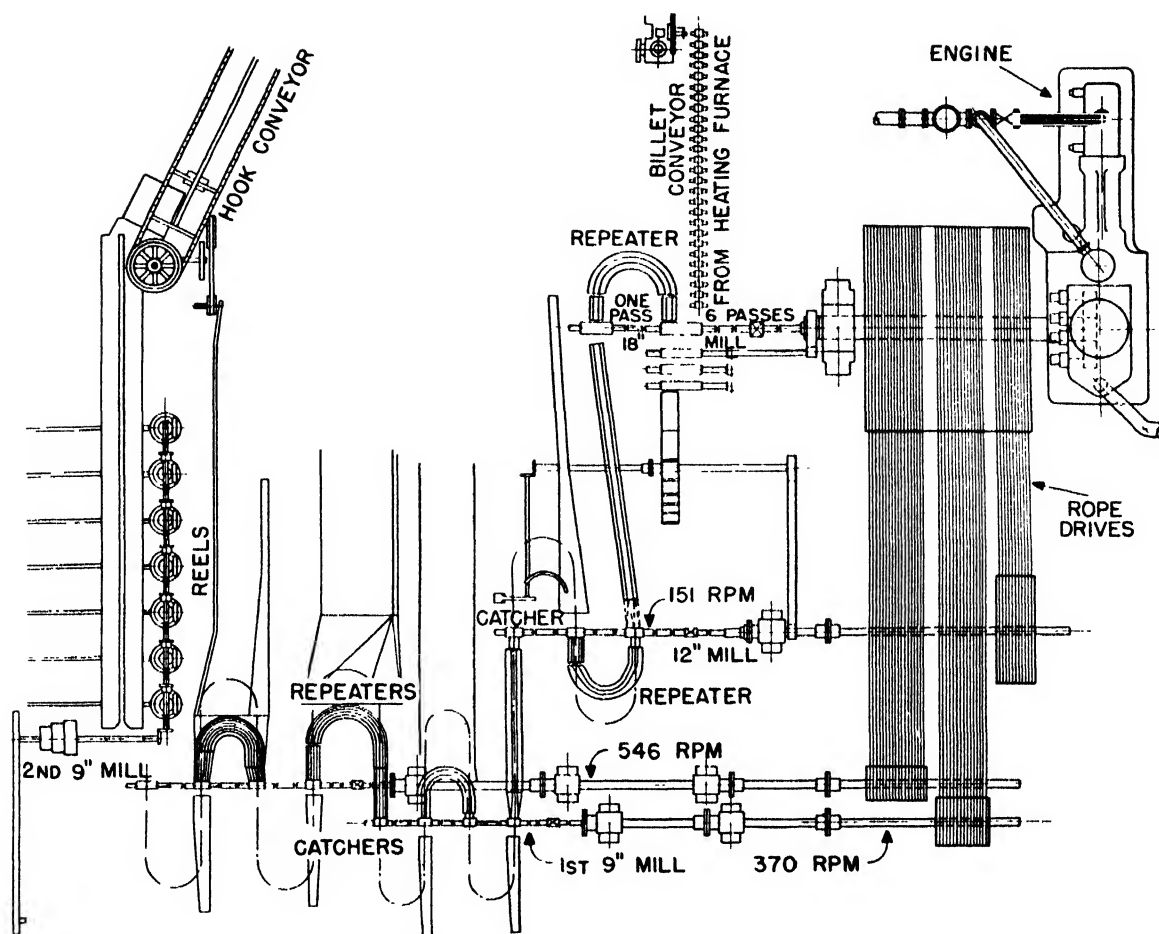


FIG. 40-2. General layout of Garrett rod mill.

Garrett erected his first mill in 1882, and at once he was able not only to roll rod in much longer lengths, but also to double, and more than double, the output of the older looping mills.

Accessories to Garrett's Mill—After the erection of his first mill, Garrett was able, from time to time, to add certain improvements. One of the most important of these improvements was the use of repeaters. As pointed out in connection with merchant mills, these are semicircular trough-like guides placed so as to guide the end of the piece from one pass to the next. As the repeaters are open at the top, a loop once formed is free to rise out of the trough, and enlarge to correspond to the elongation, or otherwise adjust itself to any difference in the speeds of the two passes. A looping mill fully equipped with repeaters is practically as automatic as a continuous mill. In the case of the earlier looping rod mills, however, repeaters did not work equally well on both sides of a mill. In these mills, the passes from the first few roughing stands to the finishing are alternately oval and square, the squares coming out on one side of the mill and the ovals on the other. On entering the square passes, the ovals must be turned on edge, and the use of guides for doing this work generally resulted in the production of much scrap, due to their frequent failures to edge the ovals properly. In some present-day Garrett mills, however, oval repeaters have been perfected and catchers are necessary only at points where cropping must be done.

Number of Strands—Another improvement introduced by Garrett, who was the first to employ the scheme, was the practice of finishing more than one rod at a time. Due to the shortness of the billet, the roughing train was able to break down the billets much faster than the finishing train could roll the rod in a single strand, even when the latter was speeded up to 550 r.p.m., which is the maximum speed at which it was practicable to run this train. Since the rod makes only one pass in the finishing and most of the intermediate stands, there was a great amount of roll space unused in rolling a single strand. By adopting power reels, and reeling each strand on independent reels, Garrett found no difficulty in finishing two strands at once in the same trains and thus doubling the output of the mill. In present-day Garrett mills, four, and even six, strands may be rolled simultaneously. The number of strands rolling in a mill has considerable effect upon the forming of the rod, and it is considered good practice to keep the mill as full as possible.

Floors—Another characteristic of looping rod mills is the sloping floors, which, though a simple device, play an important part in their operation. Since it is necessary to run the rolls in each finishing train at the same rotative speed, and it is not practicable to take up all the elongation of the piece, or equalize the rolling time of the different passes by using rolls of different diameters, it is necessary to allow the loops, which are formed after catching or repeating, to grow upon loop floors. If

these floors are level the frictional resistance soon becomes so great that it stops the forward motion of the loop, when the rod will pile up and become hopelessly twisted. In the old days, boys were employed to draw the loops back with hooks or tongs, but in Garrett mills this frictional resistance is overcome by laying the looping floors at an incline, sloping from the rolls, except on the oval side where a narrow level space is left to provide standing room for the catchers. When more than one strand is being rolled, as in all present-day mills, these floors must be provided with guiding channels and other facilities to prevent the different loops from becoming entangled.

Layouts of Garrett Mills—The arrangement of the roll trains in later looping mills has been subjected to considerable variation. A typical layout is shown in Figure 40—2. As originally planned by Garrett, the three-high billet mill was placed near the heating furnaces, which are now mainly of the continuous end-discharging type with floors sloping downward from the charging end. Two or more of these furnaces may be used to heat the billets, which are conveyed to the mill by roll tables. Following the billet mill, at a distance great enough to permit the piece from the last pass to clear the billet-mill rolls, is the roughing mill which, in present-day mills, may be a looping train of three stands or two or more stands arranged in tandem. Then come, with their looping floors sloping in opposite directions from the narrow level standing, the intermediate and finishing rolls, arranged, usually, in two trains of four or five stands each, lying end to end along two parallel lines in close proximity to each other, and so placed that the first intermediate pass is in line with the last roughing pass and far enough away to give proper clearance for the piece. In some mills especially designed to roll small rods, the finishing and last intermediate stands are coupled together in a third and separate train, which is driven at the highest speeds practicable, to reduce the size of the loops and make it possible to finish the rod at a higher temperature.

Combination Mills—In roughing down the 4-inch by 4-inch billets on the three-high mill, considerable time is consumed in making the necessary five to seven passes back and forth through the rolls. Not only does this delay tend to cool the steel somewhat, but these operations require some heavy hand labor. To overcome these disadvantages as far as possible, some looping mills make use of a group of continuous rolls for roughing purposes. This group usually consists of eight stands, and displaces not only the billet mill but the roughing train as well. With such a group, the work of roughing can be done much faster than the looping trains can dispose of the material even when four strands are rolled at once. Consequently, one continuous roughing mill is used to feed two looping finishing mills, one of which is placed on the right and the other on the left side of the roughing mill. By locating these finishing trains at a suitable distance beyond the roughing mill, the sections from the last roughing stand are carried forward or looped into the first intermediate pass by guides similar to the repeaters previously described. This type of mill is sometimes referred to as the Garrett semi-continuous mill.

The Double Belgian Mill, which is also designated by the name of Boecker's wire mill, represents another attempt to apply the combined continuous and looping methods of rolling, in this case, to the finishing mill. It consists, usually, of seven or nine stands of rolls, six or eight of which are arranged in two parallel lines or trains of three or four stands each, so that the corresponding stands in each line are opposite each other,

thus forming three or four continuous groups of two stands each. The mill is driven from a central shaft lying between the two parallel lines of stands, and from this shaft the power is transmitted to the rolls through three sets of gears. In this way the speed and direction of rotation of the various rolls is so regulated that a rod, passing through the mill, can be looped by catchers or repeaters from one continuous set to another, and from the last continuous set into the single finishing stand, which is two-high. A mill of this design offers some of the advantages of both the looping and continuous types, but also possesses some of the limitations of both. Thus, while it is capable of producing rods that are exceptionally uniform in section, the complicated system of shafting and gearing makes its upkeep expensive and tends to offset its advantages.

Continuous and Looping Mills Compared—A just comparison of the continuous mill and the looping mill is difficult to make. Local conditions and a great number of other factors must all be considered. For example, there is the matter of raw material. The small billet of full standard length, i.e., 30 feet, cannot be rolled on a looping mill because of the long exposure on the looping floor, which cools it below the temperature for hot finishing. On the other hand, the continuous mill is particularly suited to rolling these long billets, and, when combined with the special heating furnace previously referred to, presents a layout by which the rolling can be done at higher and more uniform temperatures, which saves power and minimizes the variation in section of the rod from end to end.

As already pointed out the large, or 4-inch by 4-inch, billet may be rolled on the continuous, the Garrett, or the combination mill. Mills of the Garrett type with looping trains for all except the first roughing, or billet mill, are limited to a billet weight of about 175 pounds for small rod, because, with a larger billet, the last end of the rod becomes so cold that excessive power is required to roll it, and the variation in size of the rod from end to end is too great to be tolerated. Continuous mills and combination mills can roll, without a second reheating, $3\frac{3}{4}$ by $3\frac{3}{4}$ -inch billets weighing up to 1200 pounds. On billets of these weights, the continuous mill requires less power and fewer operators than mills which are chiefly of the looping type; also the repair costs per ton on modern continuous mills are less than on looping mills. The heavier the rod bundle the better, on account of the longer length of the wire that can be drawn from it in one piece, which reduces the handling required and the scrap produced in the drawing and finishing operations.

As to other differences made in the product by these two methods of rolling, these are of such a character as to leave little from which to choose. The tendency exhibited by the continuous-mill rod toward flatness in the middle is offset by the variation in section from end to end of the looping-mill rod when rolled in long lengths. Since the continuous mill cools the rod during the rolling less than the looping mill and uses no hand labor for catching, it permits the use of a greater number of roll stands and passes. This great number of passes makes it possible to use better-shaped passes with smaller reductions from pass to pass, and thus avoid overfills, which frequently cause surface defects in the rod—defects that are of very serious character in many wire products.

Consequently, most mills installed in recent years are either straight continuous mills or employ only two or three loops. However, some of these mills use several motors, to provide more flexibility in rolling a wide range of sizes as well as to keep the loops short.

Rolls for Rod Mills—As to size, the rolls used for rolling rods are subject to considerable variation according to the type of mill and the purpose for which they are used. Thus, in Garrett mills the rolls are long, and vary in diameter from 16 or 18 inches for the first roughing rolls to 9 or 12 inches for the intermediate and finishing rolls, while in continuous mills using $3\frac{1}{4}$ by $3\frac{1}{4}$ -inch billets they vary from about 18 inches for the roughers to about $11\frac{1}{2}$ to 12 inches for the finishing groups. The roughing rolls may be cast steel or cast iron. However, cast steel is only used where it is required for strength because of the poorer pass life obtained. Most mills at present use grain iron rolls in the roughing stands.

In the intermediate and finishing stands, the rolls are either grain iron or chilled iron with the practice being, until very recently, to use chilled iron wherever possible because of the longer pass wear. However, recent development of grain rolls has improved the pass life to the point where they are now being used even in the finishing stands of some mills. As to the roll passes, these are simple in form but of a design to give the necessary reduction without developing defects in the material. In Garrett mills and continuous mills using 4-inch by 4-inch billets, the first roughing passes are rectangular or box passes, or diamond and square passes, but in all other rolls except the finishing, the passes for round rods are arranged to give alternately squares and ovals, etc. For round wire, the last oval feeds the finishing pass which, of course, must be circular. To reduce the number of roll changes otherwise necessary, a number of duplicate grooves are cut in each set or pair of rolls, so that when one set of grooves becomes defective or too badly worn a new set can be made available for use merely by resetting the guides. When all the grooves in a set of rolls have thus been used, they must be replaced by a new set while the worn rolls are being redressed, so that duplicate rolls as well as duplicate grooves are required.

Designing the Passes—Here, the matter of chief concern is the proper regulation of the draft in each pass. The highest drafts are made in the roughers, where the reduction in any one pass may be as high as 45 per cent though a limit of 35 per cent represents better practice. In the intermediate sets of rolls, a maximum draft of about 30 per cent may be attained in the first passes, but as the rod approaches the finishing pass the amount of the reduction permissible gradually diminishes, until in the final pass it is well under 15 per cent. Within the limits mentioned, the drafts will be determined largely by the number of passes provided by the mill, which in turn will depend upon the type of mill, the size of billet used, and also the size of rod to be rolled. For rolling 4-inch by 4-inch billets to No. 5 rod, Garrett mills usually provide from 18 to 20 passes; some continuous mills use 21 passes, while others provided with an intermediate reheating furnace may use as many as 24; and combination and double Belgian mills may use 18 to 24 passes. Nearly all the mills are so arranged that the size of the finished rod can be varied, that is, made coarser than No. 5, either by "opening up the mill," thereby producing larger sections in each pass, or by omitting the use of some of the stands immediately preceding the last. In the latter case the finishing is done on one of the stands used as an intermediate when rolling No. 5 rod.

Housings and Guides—The construction of housings, with their mechanism for making necessary adjustments of rolls and guides, differs in the different mills, but is the same in principle as described elsewhere in this book for corresponding types of billet and mer-

chant mills. The setting and adjustment of the rolls and of the receiving and delivery guides require a deal of experience, and, upon the efficiency of these adjustments depends, in great measure, the quality of the rolled rod as well as the tonnage of the mill.

Rod Reels—The daily output of rod mills in use today varies from 450 to 1200 tons, which tonnages, translated into linear units of No. 5 rod, are equivalent to about 1500 and 4000 miles, respectively. From this statement it is evident that this enormous output would have been impossible without adequate reeling facilities. As previously pointed out, this need caused the early adoption of automatic reels, the first practical development of which was introduced about 1880. The rod reels now in use are mainly of two types, known as the **pouring reel** and the **laying reel**. The pouring reel consists essentially of a horizontal steel disc or pan mounted upon a vertical shaft so that it may be revolved at a speed corresponding to the delivery speed of the rod. Near the circumference of this pan are two circular and concentric rows of pins, or spokes as they are often called. Into the annular space bounded by these spokes the rod pours from the end of a pipe which conducts it from the finishing rolls, and since the rotative speed of the reel is adjusted to the linear speed of the rod, the latter is laid in a neat coil within this space, a single ring of the coil being formed at each revolution of the machine. In the laying reel, the receptacle is stationary, while the end of the delivery pipe, itself, is made to revolve in a circle, thus laying the rod within the annular space provided by a motion similar to that in coiling a long rope by hand, so that the rod is given a twist at each revolution of the pipe, or as each ring of the coil is formed. The laying reel, therefore, can be used only with small round sections; however, pouring reels can only be used up to about 4200 feet per minute. Laying reels provide a better coil on small rod. Hence, most modern mills rolling both small and coarse rod have both laying and pouring reels. In both types the diameters of the coils are determined by the relation between the delivery speed of the rod and the rotative speed of the reel, for the peripheral speed of the coil being formed must equal the rod delivery speed. As these two factors cannot always be kept in exactly the same relation, the diameters of the coils vary somewhat. In all cases the reels are designed to raise the coil or lower the spokes, and deliver the product upon a conveyor, which carries each coil to some designated point where it may be disposed of as desired. In case of wire manufacturers who roll their own rods, the rod mill is located very near the wire mill, and the coils are carried by the conveyor directly into the cleaning or heat treating departments of the latter. Most wire mills, whether they roll their own rods or not, will be provided with covered yards where rods of various kinds of stock may be stored for use as required.

Defects in Rods—To produce a satisfactory rod requires that proper care be exercised in every department concerned with its preparation. To begin with, the steel must be properly made and its composition should be uniform and as near that desired as possible; also, it is essential that the ingot be properly cast and rolled at a suitable temperature, and that the billet be free from pipe, seams, and slivers. Even with perfect billets, considerable care and skill are required to produce satisfactory rods. Assuming that the rod mill, itself, has been properly designed and speeded and equipped with all the necessary accessories, unsatisfactory rod may still be produced by improper heating, by the use of improperly designed ragging on the roughing rolls, by improperly adjusting the rolls and

guides, or by using the grooves too long. In order to finish the rod correctly, every pass in the mill should be made to produce, as near as possible, the section it was originally designed to produce, and to bring about this result all the passes must be kept full but none may be allowed to overfill. Great emphasis is to be laid upon all these features of operation, because, from the point of view of the wire mill it is most essential that

the rod be uniform in composition, and also in section, that it be close to the size and shape required, and that it be free from such surface defects as slivers, seams, and fins. These defects, as well as the composition, are matters of importance to both the consumer and the wire mill, for none of the subsequent wire-drawing operations or finishing processes will eliminate them or reduce their capacity to do harm in the final product.

SECTION 4

OUTLINE OF WIRE-DRAWING PROCESSES

The rolling of the rods is a fairly well standardized process. The primary object of rod rolling is, of course, to put the steel into such shape that it can be most efficiently cold drawn into wire. Using steel of the required composition, the rod is rolled to the desired size and shape by following a standard practice on a standard rod rolling mill. From this point, however, the method for manufacture into wire depends entirely upon the end use to which the wire is to be put. These uses are many and varied, and the wire drawing practices necessary to provide the wide ranges of required wire characteristics vary accordingly. Obviously, it would be impossible to cover all the practices in detail here. However, a brief summary of the various operations necessary to process the rod to finished wire will be outlined.

Preparing the Rod for Drawing—In order to prepare the rod for drawing, it is first given some necessary preliminary treatments. In general, these consist of acid cleaning and coating following a heat treatment when required. The heat-treating process is usually one of the following: patenting, annealing, or normalizing. These processes will be described in more detail later. Low-carbon rods which are to be drawn into wire usually do not require any heat treatment and are cleaned and coated after rolling. In the process of rolling, the rods acquire a mill scale or oxide on the surface. Oftentimes, it is necessary to store the rods for some time before drawing. In storage they may pick up either some additional oxide in the form of rust or just plain dirt. All mill scale, oxide, or dirt must be removed before the rods are drawn into wire. This is accomplished by placing the rods in a solution of hot dilute sulphuric acid for from fifteen to thirty minutes. The action of the acid loosens the scale and frees the rod of all rust or dirt, leaving the surface of the rods perfectly clean. The rods are then removed from the acid and given a thorough rinsing in a spray of high-pressure water. This removes all the acid from the surface and from between the various strands. The next step is to give the rods a suitable coating, usually lime. The purpose of the coating is threefold: first, to protect the surface of the cleaned rods from rusting in the atmosphere; second, to neutralize any traces of acid left from the cleaning; and, third, to serve as a carrier for the lubricant used in drawing the rod to wire.

For many years it was customary to coat rods which were to be drawn into ordinary finished wire with a sull coating. This is accomplished by placing the cleaned rods on a rack or a travelling conveyor and spraying them with a fine mist-like spray of water. The rods are then dipped in a tank containing hot milk of lime. In recent years the practice of sull coating has been discarded, and the rods are coated with the milk of lime directly after the water rinse. Sull coating is used now only on a few kinds of wire; in some instances for manufacture of cold-headed bolts.

Borax is another coating which has been used quite extensively in recent years with good results. After the coating is applied, the rods are baked in an oven or other type of baker to dry the coating. This function will be described later in more detail.

Drawing the Rod—After the rods are properly cleaned, coated, and baked, they are delivered to the wire-drawing equipment. Today, most wire which is drawn three or more drafts is produced on continuous machines. Wire which is to be drawn one or two drafts is produced on single or double-deck motor blocks. Wire in sizes $\frac{3}{8}$ -inch and coarser is drawn on horizontal bull blocks. However, there is still a considerable amount of wire produced on the so-called wire drawing frame. This frame, in the type most generally used for drawing rods, supports a single die and the power driven block for drawing the rod through the die, also a drawbar for drawing the first few feet of the rod. One end of the rod is now pointed, or tapered, so that it may be threaded through the die hole, which is somewhat smaller than the rod in section. Next the die holder, on the entering side of the die, is filled with a specially prepared grease, or some other suitable lubricant, so that in passing through the die the rod must first pass through the lubricant. The pointed end of the rod is now inserted through the proper die hole, where it is grasped by tongs attached to the drawbar, and drawn through far enough to be attached to the draw block. When this block is started revolving, it coils the wire about itself and thus continuously draws the rod through the die, thereby bringing about a fixed decrease in its sectional area and a proportional increase in its length.

Draft, Drawing and Process Wire—The amount of the reduction in the sectional area, as in the case of rolling, is expressed in per cent of the original area and is known as the **draft**, while the operation itself is called **drawing**. In general the draft on the rod varies from 10 to 45 per cent according to the kind of wire to be made. As soon as a rod has been given a draft, it is thereafter designated as a wire, though many more drafts and various other treatments may be necessary to work it into wire of the size, finish and temper desired. In wire-drawing plants, any wire which, following the initial drawing from the rod, is to receive further work or treatment before it is finished is designated as **process wire**. After the first draft from the rod, process wire may be finished in various ways.

Dry Drawing and Wet Drawing—There are two processes for drawing wire. These are designated as dry drawing and wet drawing. Mechanically, the processes are the same; that is, the wire is drawn through a die and wound up on a block. The difference in the processes is in the coating applied to the wire and the lubricant used. All wet-drawn wire is first given one or more dry drafts from the rod. This process wire is then thoroughly cleaned, rinsed, and immersed in a dilute solution of copper or tin sulphate or a mixture of both of these salts.

A chemical reaction takes place which results in the deposition upon the wire of a thin metallic coating from the solution used. After coating, the wire is usually kept under water to protect it from the influence of the atmosphere.

In subsequent wet drawing, the pay-off reel containing the wire is placed in a tub of water. To this water a special type of soap is usually added to act as a lubricant. In recent years special coating solutions have been developed which are applied to the wire after the copper or copper-tin solution treatment. These coatings protect the surface from the atmosphere and are also quick drying, so it is possible to store the wire without keeping it under water and also to permit drawing from a dry reel. In drawing, the copper or copper-tin solution imparts a characteristic metallic color to the wire which is known commercially as **coppered wire** or **liquor-finished wire**. The latter term designates the brass-colored finish obtained from the copper-tin coating mixture.

Types of Wire—For all practical purposes, wire may be described as being low-carbon or high-carbon. Al-

though there are additional classifications in AISI Product Manual Section 16, 0.30 carbon may be considered the dividing point and, broadly speaking, wire under 0.30 carbon is referred to as low-carbon, while that which is 0.30 carbon and higher may be considered as high-carbon. The end uses of the various types of wire require different characteristics in the finished wire as to physical properties, finish, gage tolerance, electrical conductivity and many others. These requirements are met by designing manufacturing practices which employ various amounts of cold work (drawing), various types of heat treatments, or combinations of both. All of these practices are carefully worked out and are designed to provide wire which will adequately meet consumer requirements and at the same time make the most efficient use of producing equipment.

A general discussion of wire manufacture follows which is divided into six sections, namely, "Preparation of Rods and Wire for Drawing," "Wire-Drawing Equipment," "Wire-Drawing Processes and Operations," "Heat Treatment of Wire," "Protective Metallic Coatings for Wire," and "Some Fabricated Wire Products."

SECTION 5

PROCESSES AND EQUIPMENT FOR PREPARING RODS AND WIRE FOR DRAWING

Importance of Cleaning—All of the hard, brittle oxide commonly called mill scale, which forms on hot-rolled rods or on heat-treated rods, and also the slow-forming red rust which forms on long exposure to the air, must be entirely removed before drawing. If these oxides remain on the surface of the rods, they result in very rapid wear on the wire-drawing dies and also cause scratched and off-gage wire. Proper cleaning and coating is largely responsible for success in the wire-drawing operation.

Method of Cleaning—As already stated, the method generally employed for cleaning the material preparatory to drawing consists in dipping the coils into a vat of hot, dilute sulphuric acid. The action by which this acid removes the scale is for the most part mechanical rather than chemical, for ferrous-ferrous oxide, Fe_3O_4 , which is the chief constituent of the scale, is but slightly soluble in sulphuric acid. The acid, however, is able to penetrate to the metal beneath the scale, where it reacts with the iron forming iron sulphate, a soluble neutral salt, and liberating a mixture of gases, mainly composed of hydrogen. This action results in loosening and detaching the scale from the surface of the metal, when it sinks to the bottom of the vat where it accumulates and must be removed at frequent intervals. Unlike the black scale, the red rust (which is a hydrated sesquioxide of iron of the general formula $\text{XFe}_2\text{O}_3 \cdot \text{YH}_2\text{O}$ or a double compound of ferric oxide and ferric hydroxide, $\text{Fe}_2\text{O}_3 \cdot 2\text{Fe}(\text{OH})_3$), is readily soluble in sulphuric acid, and

under the conditions of pickling this action results in the formation of iron sulphate, also. This salt remains in solution, and as concentrated acid is added from time to time to replace that neutralized, the solution eventually becomes saturated to such an extent that it is no longer fit for pickling. The exhausted solution must then be replaced.

Manner of Handling the Material—When the rods are removed from the hook conveyor in the rod mill, they are loaded into standard-gauge gondola cars. The coiled rods are loaded in an upright position to facilitate unloading with a so-called "C" hook (Figures 40—3 and 40—4).

From the rod mill, the rods are transported to the rod storage where they are unloaded by an overhead crane. The crane is equipped with the "C" hook referred to above, which has a capacity of two to three tons. Facilities for storing rods consist of large bins or racks. Each gage and steel composition is stored in a separate bin in order to avoid confusion and possible misapplication.

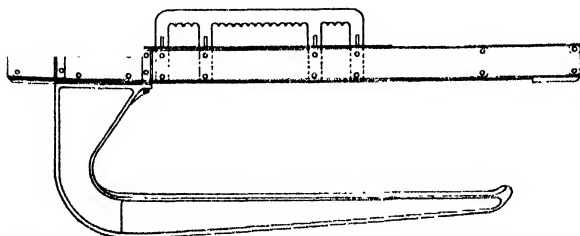


FIG. 40—3. Open-type "C" hook for handling coiled rods and wire.

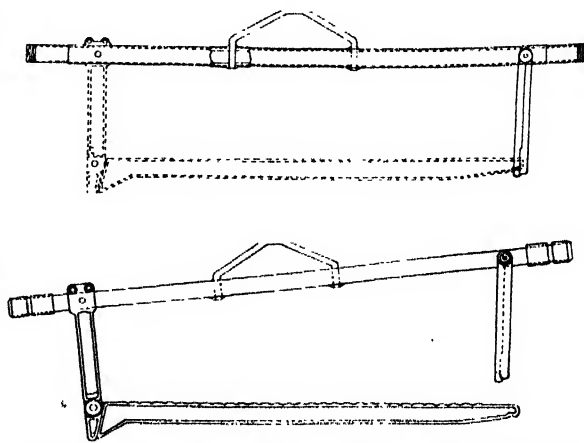


FIG. 40—4. Latch-type "C" hook for handling coiled rods and wire.

In modern integrated mills, the rods may be moved directly from the rod storage to the cleaning crane by the rod-storage crane. In other mills, the rods are set down in racks by the rod-storage crane and then moved to the cleaning crane by ram tractor or on rod trucks. In either case, the rod-storage crane sets up the exact amount of rods which the cleaning crane is designed to handle in one lift, so that there is a minimum of handling.

The "C" hook or hairpin hook, as it is sometimes called, is a specially designed device for handling coiled rods and wire. It is designed to lift as a unit the number of coiled rod bundles which the cleaning crane was designed to handle in one lift. There are two types, the open-end type and the latch type. Both are made by using a section of extra-heavy pipe, or I-beam, which is long enough to span the cleaning tank. This pipe has a metal eye attached at the approximate center, or balance point, into which the crane hook is inserted for lifting. From the pipe, a very heavy hanger is suspended. In the open type (Figure 40-3), this hanger is in the form of a large letter "C" with the bottom section straightened horizontally and made long enough to lift the proper number of coils.

In the latch type (Figure 40-4), the hanger is made more in the shape of the letter "L," and the horizontal section is pivoted to the vertical section which allows for a slight movement of the cross piece. At the opposite

end of the pipe, a second hanger is suspended. This hanger is pivoted to the pipe allowing it to swing freely. When the hook has been threaded through a lift of rods, this second hanger is swung down and latched to the free end of the cross piece. This gives support to both ends of the cross piece. All of the parts which come in direct contact with the acid are made of phosphor bronze, aluminum bronze, or other acid-resisting material.

A hook with its load is referred to as a "pin" or "stem" of rods or wire.

Types of Cranes—There are two types of cranes generally used in cleaning, namely, the circle crane and the overhead electric traveling crane. One type of circle crane consists of an upright standard which is set in bearings to allow the upright standard to turn in a complete circle. A horizontal boom is placed on the upper part of the standard. This boom or arm is built long enough to reach the center of the cleaning and coating tanks which are placed in an arc around the base of the standard. Power for lifting the material to be cleaned is supplied by a steam piston which is placed on the boom or standard. Power for turning the standard and boom is supplied by placing a ring gear around the upright standard. This ring gear is then driven by an electric motor which is connected by the proper gearing and shafting. Figure 40-5 shows a variation of this type of crane, in which the standard is fixed.

Use of the circle type of crane requires the cleaning

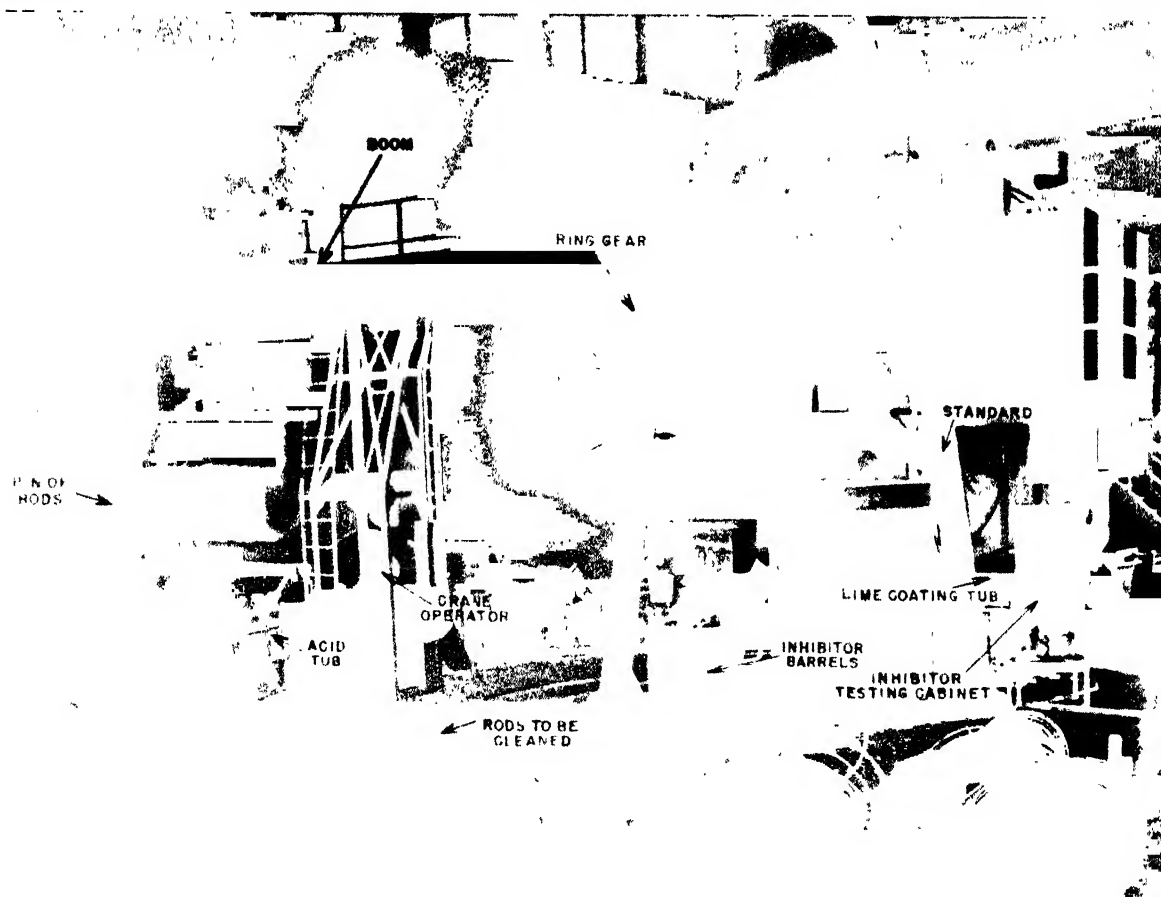


FIG. 40-5. General view inside a cleaning house showing cleaning and coating tubs arranged around the base of a crane.

and coating tanks to be placed in an arc around the upright standard (Figure 40—5). The number of tanks is then limited by the radius of the boom. Where a large volume of production is required, two circle cranes may be used. Here the first crane handles the work through all the acid cleaning and rinsing, while the second crane handles the work in the stages during which the proper coating is applied.

When the electric overhead traveling crane is used, the cleaning and coating tanks may be placed in a straight line or sometimes two lines are used.

The gantry crane is another type of electric crane in use. This crane requires all the tanks to be in one line as the crane has no cross travel, and the crane hook moves in relatively the same horizontal line at all times. This type of crane has value where there is a large volume of work and few varieties of product.

Construction of Tanks—The rectangular acid tanks are constructed of welded steel plate, and are usually large enough to hold two stems of rods. The bottom is reinforced with channels and angles, and the interior is lined with a $\frac{3}{8}$ -inch thick membrane of rubber or neoprene to protect the steel from the acid. Inside the rubber lining a layer of acid-proof brick is placed com-

pletely covering the sidewalls and bottom of the tank. The brick lining is bonded to the rubber or neoprene with acid-resisting sulphur-base cement. Acid-resisting synthetic-resin cement is used in the brick joints. All unprotected exterior surfaces of the tank are covered with a black vinyl-resin-base acid-resisting paint.

Water-rinse tanks are also usually rectangular and constructed of welded steel plate. Such tanks usually are of one-stem capacity. All surfaces are coated with a black vinyl-resin-base acid-resisting paint. The upper section of a water-rinse tank is provided with a series of high-pressure water nozzles, while the lower part of the tank contains fresh running water for rinsing.

Lime tanks are also rectangular and constructed of welded steel plate, and usually have a capacity of one stem.

Arrangement of Tanks—As already indicated, when a circle crane is employed, the number of tanks is limited by the radius of the boom of the crane. When an overhead traveling crane is used, the number of tanks is unlimited. Figure 40—6 shows a typical arrangement for so-called straight-line cleaning. In general, the acid tanks are used first in the cycle, then the water-rinse tank, and finally the coating tanks. Sull coating today is

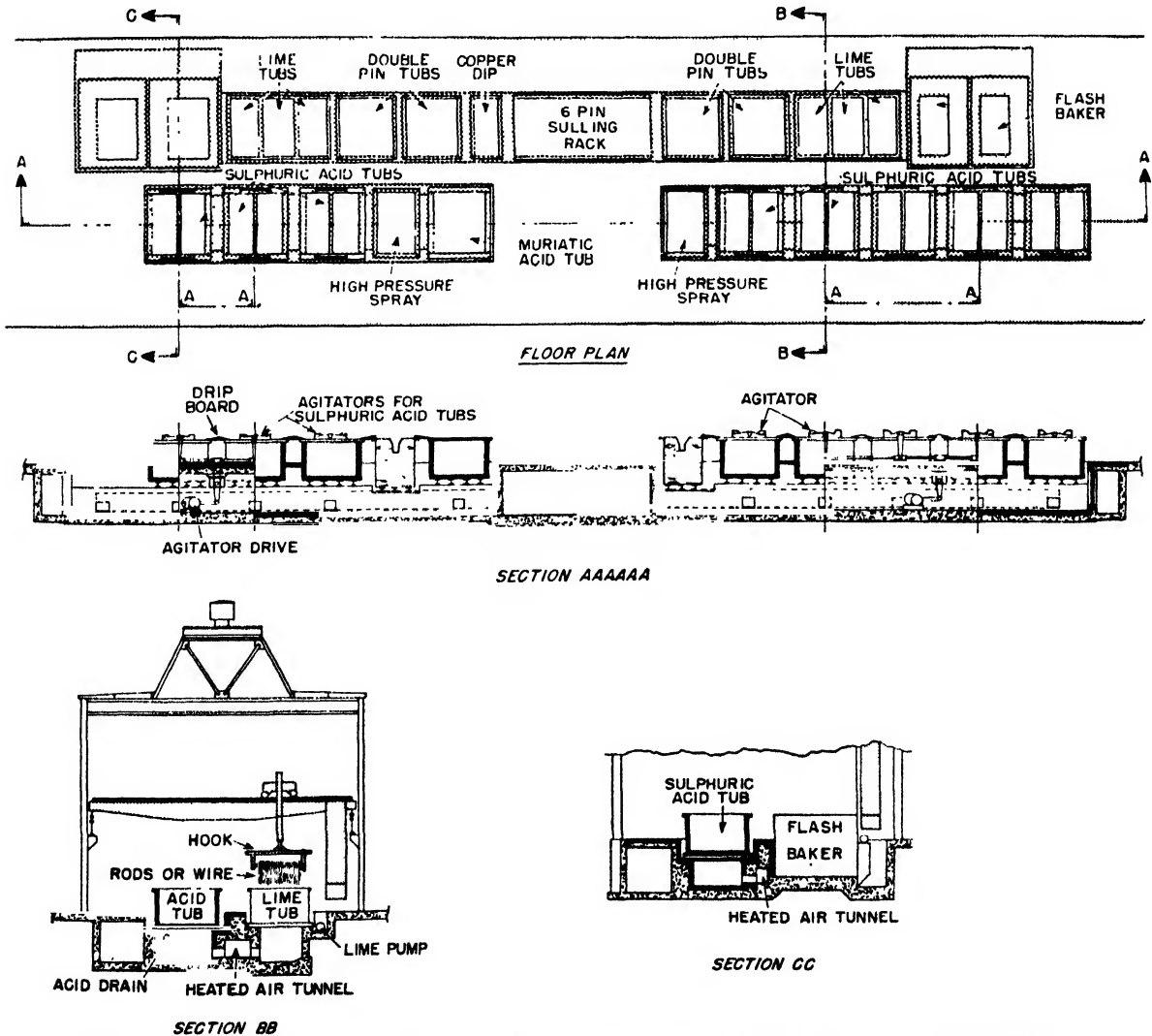


Fig. 40—6. Typical arrangement for so-called straight-line cleaning of rods in preparation for drawing.

in very slight demand, but usually a small section is provided where the rods can be treated for formation of this coating.

Concentration of Acid—There are three factors which affect the cleaning operation, namely, time, temperature, and concentration of acid. In operation, the tanks are first partially filled with water. Then the acid is added in the required amount and the steam for heating the bath is turned on. By the time the tanks are filled by the addition of more water, the steam has raised the temperature to the required point, usually about 180° F, and the acid concentration has been adjusted to the required amount.

Certain kinds of steel require longer time for cleaning than others. Low-carbon rods which may be finished at a high temperature on the rod mill will have a very heavy scale. Also, rod coils which are very compact, as is usually true with the coarser gages, are more difficult to clean. Bessemer and high-carbon steels react much faster to the acid, and the time of cleaning is shorter. This is particularly true of high-carbon patented rods which have the scale broken up by the sheaves on the take-up frame of the patenting furnace. The acid concentration, then, is varied according to the kind of steel being treated, the low-carbon heavily-scaled material requiring a solution having a higher concentration of acid, with the higher-carbon steels and patented material requiring a lower concentration.

Temperature for Cleaning—The control of temperature in the cleaning operation is very important since the rapidity of the reaction between acid and steel is greatly affected by temperature. It is known that the reaction is 100 times as fast at 190° F as at room temperature. Too high a temperature, therefore, can be very wasteful. It causes rapid and high usage of the acid, develops unnecessary fumes which, in turn, can cause excessive corrosive action on the steelwork of the building. What is more important, it causes pitting action on the surface of the steel being treated. The tanks in most modern cleaning houses are equipped with thermostatic controls so that a uniform pickling-solution temperature is maintained.

Inhibitors are used to aid in preventing pitting or over-cleaning. These usually have a nitrogenous hydrocarbon base. The theory of the use of inhibitors is that when the scale has been removed from the steel, the inhibitor forms a protective film on the cleaned surface and minimizes any additional attack by the acid.

Time of Cleaning—The time required for cleaning will vary according to the amount of scale to be removed and the type of steel being treated. This may vary from as little as 10 minutes for high-carbon patented rods to 30 or 35 minutes for low-carbon heavily-scaled rods. When the cleaning facilities serviced by the crane provide enough acid tanks, it is possible to use all tanks for the long-cycle work and reduce the number of tanks in use for short-cycle work. By this means, it is possible to keep the temperature and acidity at a fairly constant point and vary the time. Another factor which affects the time of cleaning is the change in specific gravity of the acid solution. As the work proceeds and more and more scale is removed, the iron content of the bath builds up. This is reflected in a change in specific gravity of the solution, which is usually measured by a hydrometer that provides a Baumé reading. As the Baumé figure increases, the action of the acid is retarded and the cleaning process gradually slows down. When the Baumé reading reaches about 20, the acid in the tanks is discarded, after which the tanks are washed out and refilled. In modern cleaning houses, where the tanks hold from 5000 to 7000 gallons, it is customary to have

one extra acid tank. This tank is filled and heated to temperature, ready to operate when one of the other tanks is being emptied of its spent acid. By rotating the use of the tanks in this way, continuous operation can be maintained.

Rinsing—It is very important to have the cleaned rods or wire thoroughly rinsed after the acid cleaning. The cleaned material is dipped into a tank of fresh running water. There is a series of high-pressure sprays placed around and at the top of the water tank. When the cleaned rods are dipped in the water, a switch automatically turns on a high-pressure pump. As the crane moves the rods up and down, the high-pressure sprays wash them thoroughly and remove all traces of acid or acid residue.

Coatings—After the rods are thoroughly cleaned and rinsed, it is necessary to apply a coating. The purpose of the coating is three-fold: First, it prevents oxidation or rusting of the surface; second, it neutralizes any traces of acid which may be left on the steel; third, and most important, it acts as a carrier for the lubricant used in drawing. For dry drawing, dry slaked lime or hydrated lime have been found to be the best materials for coating. Lime is a low-cost material and is easily applied. Borax is another coating which has been used for dry drawing in recent years. It is a little more costly than lime, but it offers some advantages. It dries very quickly, it does not pick up moisture, and it does not flake off or form a dust in the wire-drawing room, thereby making conditions much cleaner in the mill.

Process for Lime Coating—The lime coating is applied to the rods simply by dipping the cleaned rods into a tank of hot milk of lime. The temperature is kept between 190° F and 200° F. The rods should be kept in the solution long enough to bring them up to bath temperature. This aids in drying the coating and also makes it adhere better to the steel. Formerly, it was customary for the mills to purchase the burned lime and slake and age it before use. In recent years, very good grades of so-called quick-slaking lime and hydrated lime have been developed commercially. These are packaged in paper bags of about 50 pounds each. In using these limes, one end of the bag is cut open, and the lime is dumped directly into the lime tank where it quickly goes into solution.

Another recent practice is to recirculate the lime. This is done by installing a small tank, or section of a tank, alongside of the regular lime tank. A special pump is placed in this small section. One side of the lime tank is cut down so that the lime solution can flow over this low side and into the small section. The pump then forces the overflow back into the large tank, usually through a pipe connected into the bottom of the latter. This keeps the lime solution in constant motion so that there is no settling out. This action also produces a very fine grain lime which is quite adherent.

Coatings for Dry Drawing—In practice, several coating tubs are served by one cleaning crane. There may be three or four lime tubs. In these tubs, the lime is kept at various consistencies. This allows the several coatings which may be required in the wire-drawing operations to be kept in readiness. Rods which are to be drawn one or two drafts to produce a clean bright finish are given a very light lime coating. For three- and four-draft wires, a medium lime coating is applied, and for wires which may be drawn five and six drafts, a very heavy lime coating may be used. In fact, where many drafts are required, it is often customary to give the rod two or three dips in a very heavy lime.

Where borax is used, a separate tank must be provided. This solution is adjusted to a consistency depend-

ing upon the number of drafts the wire is to be given.

Baking—After the rods or wires are properly cleaned and coated, they must be thoroughly dried before drawing. The drying or baking is done in ovens or bakers. In the cleaning process, hydrogen is liberated by action of the acid upon the steel, and sometimes this hydrogen is absorbed by the steel. This can cause the drawn wire to be brittle, a condition known as "acid brittleness." The baking process has two purposes: one, to dry the coating so that it will function properly in dry drawing and, two, to remove any hydrogen that may have been absorbed by the steel.

There are several types of bakers. The **flash type** is the most popular. This is a baker of about the size and shape of a lime tub. One pin of rods is placed in the baker for from five to fifteen minutes, depending upon the weight of coating to be dried. The temperature is usually kept between 450 and 600° F. The **tunnel type** is another baker used. This is a long, rectangular oven with a roll or chain conveyor running through its length. In operation, the rods are placed on the conveyor and pass slowly through the baker. The time may be one or two hours at temperatures from 400 to 500° F. **Compartment-type** bakers are still used to a limited extent. In these bakers, the rods are loaded onto steel buggies which are pushed

into the oven on tracks running throughout its length. The temperature is usually between 250 and 325° F which requires a longer baking time. This may be from three to eight hours. The flash and tunnel bakers are fired with oil or gas and are equipped with high-velocity fans which rapidly recirculate the heated air. Figure 40—7 shows a sketch of a flash baker.

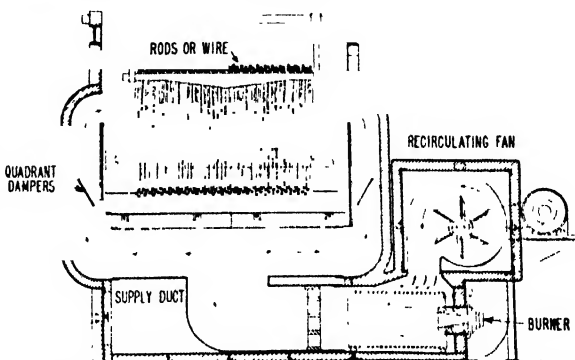


FIG. 40—7. Cross-section of a flash baker for drying rods and wire after cleaning.

SECTION 6

WIRE-DRAWING EQUIPMENT

Dies—The wire-drawing die is one of the most efficient tools used in industry. It has no moving parts; it does not remove any of the metal; yet it uniformly reduces the cross-sectional area of the steel and at the same time improves the finish and physical properties. Over the years many different materials have been used for wire-drawing dies. Chilled iron, steel plates, alloy steel, all have been used, but in the late 1920's tungsten carbide was developed and tried out. This material was an immediate success, and in a comparatively few years it replaced all other materials except the diamond. Diamond dies are still used for very fine sizes of high-carbon and alloy steels but even in this range the tungsten-carbide die can be used. Tungsten carbide is a very hard material and has great wearing characteristics but does not have very great resistance to impact. It must have some outside reinforcing when put to use. This is accomplished by pressing a small section of tungsten carbide, called a "nib," into a cylindrical steel casing, as shown in Figure 40—8.

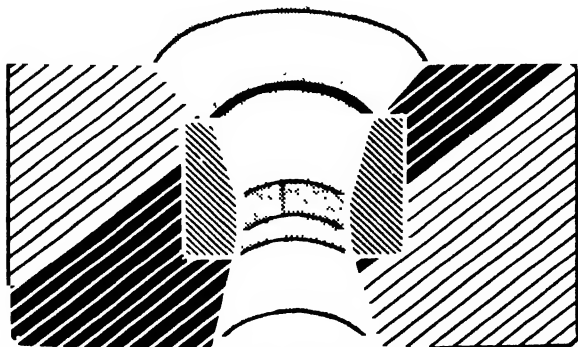


FIG. 40—8. Schematic cross-section (not to scale) of a single-hole wire-drawing die employing a nib of sintered carbide mounted in a circular steel holder.

Die Holes—The shape of the die has been found to be very important. There are four distinct zones in the die as shown in Figure 40—9. The first zone, on the entering side of the die, is somewhat larger in diameter than the rod or wire to be drawn; its purpose is to afford room for the die lubricant that adheres to the rod or wire to

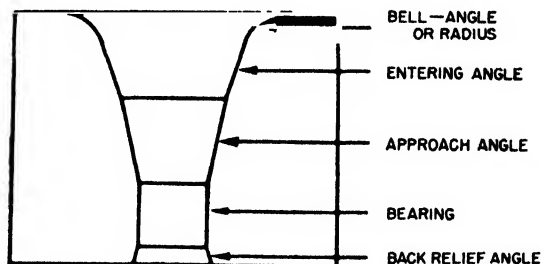


FIG. 40—9. Enlarged cross-section of die hole showing names of parts.

be drawn into the hole. This is called the **bell** and **entering angle** and gradually tapers into the second zone. The second zone is called the **approach angle** and is the section where most of the actual reduction takes place. The next zone is called the **bearing** zone and it may have a very slight angle of taper. The exit zone or **back relief** is in the form of a countersinking of the back part of the hole. This is done as a strengthening to prevent the circular edge of the hole from breaking away.

Diamond Dies—Diamond dies are frequently used when accuracy and uniformity of section are required in the finer sizes of wire, especially in the process of continuous wet drawing. In the construction of diamond dies, a diminutive flat crystal of diamond is securely fastened in a center opening of a small circular metal disc, then a hole of required diameter is worked through this diamond by special drills and diamond dust. Diamond is one of the hardest substances known and has great wearing qualities. Long lengths of fine wire can, in

consequence, be drawn through these dies with little or no change of sectional area due to wear of the die.

The Block—Wire was first drawn in very short lengths and was merely pulled through the die in straight pieces. As the lengths grew longer, some means of storage became necessary. The wire-drawing block serves this purpose. The block is a steel casting in the form of a cylinder the sides of which have a slight taper. The base of the cast cylinder is solid and there are enough cross members inside the casting to give adequate support to the sides. At the base, there is also a flanged section extending horizontally outward around the cylinder. The juncture of this horizontal flange and the vertical sides of the block is machined to a definite radius. This is called the **fillet** of the block. In the center of the block, a vertical cylindrical hole is left to allow for the drive shaft to be keyed to the block.

In operation, the wire is threaded through the die and attached to the block by a vise or clamp. The block revolves and pulls the wire through the die. As the wire is wound around the fillet of the block it is forced upward. At each revolution of the block the wire moves upward approximately one diameter of the wire being drawn. The sides of the block may be six or eight inches high. At the top of the block four vertical pins are placed.

As the wire feeds upward and reaches the top of the block, it is stored around these four pins. The result is a uniformly wound, compact coil of wire.

DRAWING MACHINES

There are several types of drawing machines. These may be grouped as follows: drawbench, bull block, motor block, multiple-draft machines, drawing frames. These will be briefly described.

Drawbench—A drawbench is a mechanism used to give a single draft to heavy material, which is afterwards usually straightened and cut to a definite length. It handles the largest sizes drawn and is especially adapted for drawing shapes, screw stock and small shafting. The machine itself consists of a horizontal framework 50 to 100 feet long, along the center line of which runs a heavy roller chain driven by heavy sprocket wheels. The die through which the material is drawn is located at the opposite end of the frame from the drive. A carriage mounted on wheels, and arranged to travel along the upper surface of the frame or bench, has suitable jaws for gripping the material and pulling it through the die, and also a heavy hook for connection with the roller chain. This equipment is similar to that used for drawing seamless tubes (see Chapter 41).



FIG. 40—10. Heavy machines employing horizontal blocks called "bull blocks" for drawing coarse ($\frac{1}{2}$ -inch to 1-inch) sizes of wire. (Courtesy of Morgan Construction Co.)

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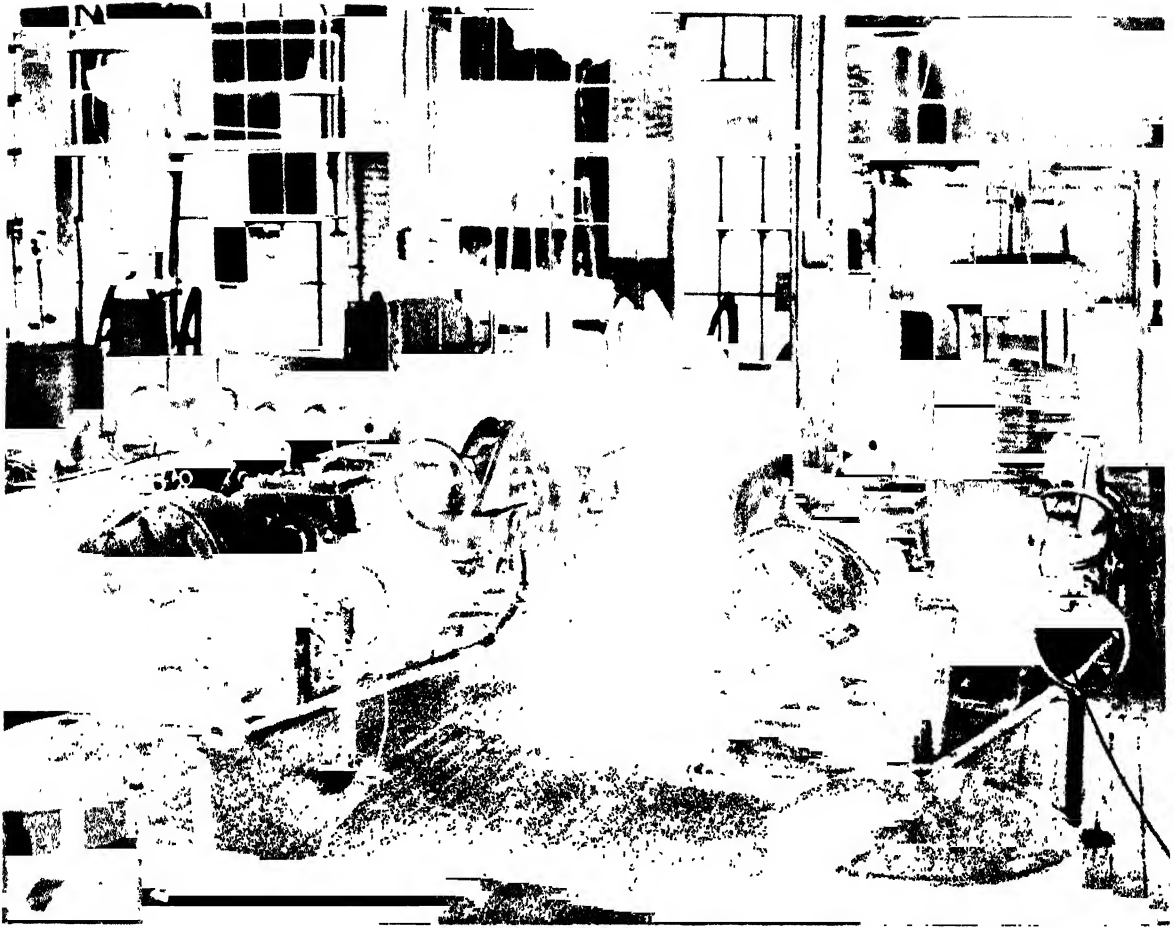


FIG. 40-11. Motor blocks, with vertical spindles, for drawing $\frac{3}{16}$ -inch to $\frac{1}{2}$ -inch wires. (Courtesy of Morgan Construction Co.)

Bull Blocks—Sizes $\frac{1}{2}$ -inch to 1-inch are usually drawn on horizontal blocks called bull blocks. These are very heavy machines built to pull these coarse sizes (Figure 40-10). They are driven by an individual variable-speed motor. The blocks are usually 36 inches in diameter. The horizontal block makes it easier and

safer to handle coils of drawn wire in these coarse sizes.

Motor Blocks—These are also driven by an individual motor, but the blocks have a vertical spindle (Figure 40-11). The size range drawn on these machines is usually $\frac{3}{16}$ -inch to $\frac{1}{2}$ -inch. The blocks may be 26, 30 or 36 inches in diameter. Motor blocks may also be

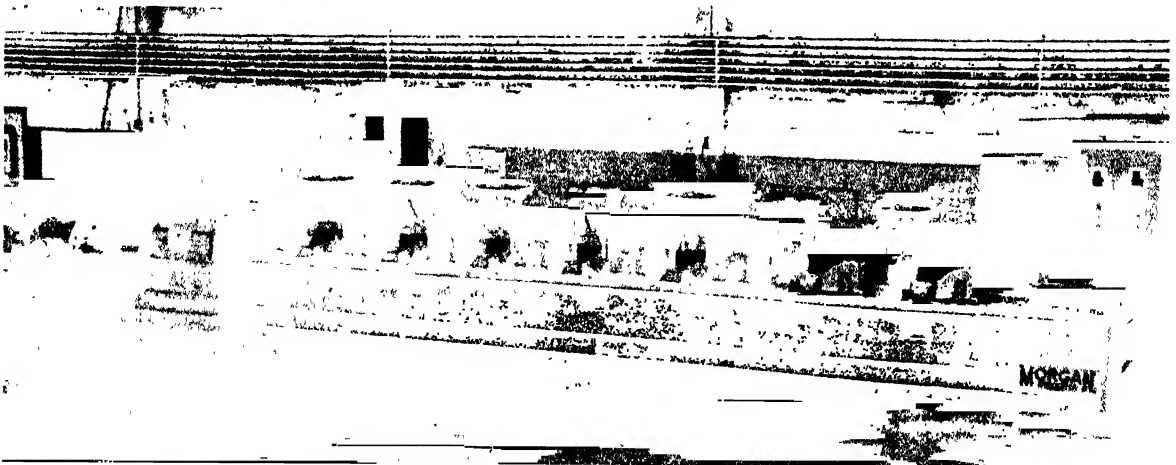


FIG. 40-12. Overall view of cumulative-type continuous wire-drawing machine. (Courtesy of Morgan Construction Co.)

equipped with double-deck blocks. This is an arrangement whereby one block is placed above the other, both being mounted on a common spindle. The lower block is made smaller in diameter than the upper block. In drawing, the wire is pulled through one die and wound around the lower block several times, then it passes around a sheave and into a second die and onto the second or finishing block. Thus two drafts are drawn on the one spindle. Sizes from $\frac{3}{16}$ -inch to $\frac{1}{8}$ -inch are usually drawn by the double-deck method. Due to the heat developed, only low-carbon wire is drawn in this manner.

Continuous Machines—Wire which requires three drafts or more from the rod is usually drawn on continuous machines. The tungsten-carbide die, due to its long-wearing characteristic, makes it practical and economical to use continuous machines for the so-called multiple-draft work. There are two general types of machines used in dry drawing. These may be described as the cumulative and non-cumulative types.

In the cumulative-type machine, the wire is drawn on a conventional type block and is allowed to build up around the block pins. At the top of the pins a ring is placed. The first end of the wire is threaded through a loop on this ring and then is fed over a dome and down through the center shaft of the block which is hollow. From here it is led around sheaves and into the next die and onto the next block. In another type of cumulative machine the wire passes from the ring up over a sheave, which is mounted above the center of the block, and then down around another sheave and into the next

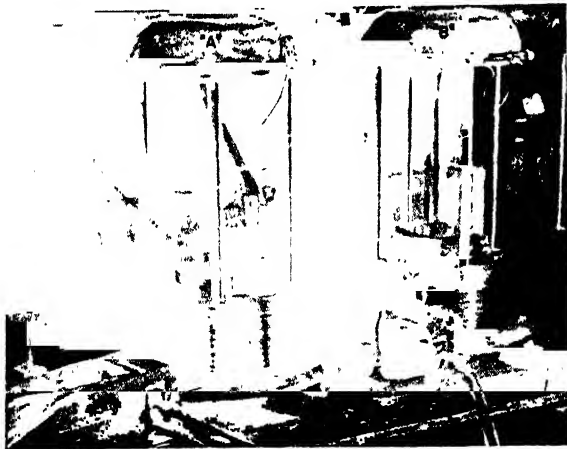


FIG. 40-13. Closeup view of the blocks of a cumulative-type continuous wire-drawing machine. (Courtesy of Morgan Construction Co.)

die. In this type of machine the blocks are usually geared so that the speed of each block is increased as the diameter of the wire is reduced. In drawing it is customary to lay out the drafting so that each block draws a little more wire than the succeeding block. This results in a gradual buildup on each block. When the blocks are filled up, the machine is shut down and all the blocks, except the finishing block, are de-clutched. The machine is then started again and the finishing block removes the accumulation from the block next to it. The slip ring at the top of this block revolves and keeps the wire paying off uniformly. When this block is nearly empty, the machine is stopped and the nearly empty block is re-clutched. This process is repeated until the build-up is worked down from all blocks. Figure 40-12 shows a

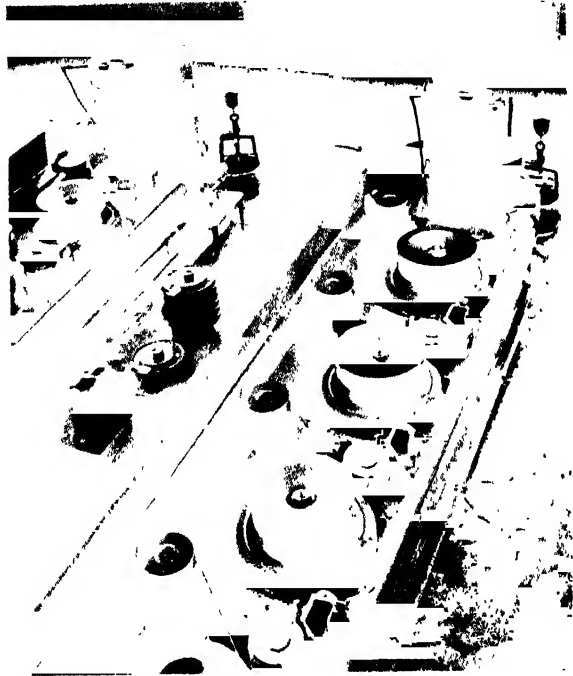


FIG. 40-14. General view of a non-cumulative type of continuous wire-drawing machine. (Courtesy of Vaughn Machinery Co.)

cumulative-type machine and Figure 40-13 shows a closeup of the blocks.

On the non-cumulative type of machine, the wire is drawn through the die and wrapped around a block in the conventional manner. This block will be about six to eight inches in height. When the wire reaches the top of the block, it passes around the sheave. This sheave is mounted at the end of an arm which is attached to a rheostat, which in turn controls the speed of the block. From this sheave, the wire passes to another sheave and into the next die. Each block of this type of machine has its own individual motor. The drafting is usually laid out to meet the nominal speeds for which the machine is designed. If, however, this drafting should get out of line and one block draws more wire than the preceding block supplies, the arm moves forward and actuates the



FIG. 40-15. Closeup view of a block of a non-cumulative type of continuous wire-drawing machine. (Courtesy of Vaughn Machinery Co.)

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reostat so that the speed of the preceding block is increased. By this means the production of each block is kept in balance with all the other blocks and there is no great amount of build-up on the blocks.

Both types of machines are equipped with means for cooling the wire between each draft. In drawing higher-carbon wire (0.30 carbon and over) it has been found that the wire will become brittle if the heat developed in drawing is not removed between drafts. Both types of machines use a blast of air directed at the wire to cool it, but the non-cumulative machine also employs a water spray inside the block. This is necessary because the time the wire is on the block is much shorter than on the cumulative machine. Figure 40-14 shows the non-cumulative machine, and Figure 40-15 a closeup of a block.

Intermediate Machines—It is usually economical to use the continuous machines which draw from the rod for not more than six or seven drafts. This produces sizes down to 0.072-inch or 0.062-inch. For sizes finer than 0.062-inch down to 0.030-inch an intermediate-size machine is used. This machine is usually the same in design principles as the rod machines but its construction is lighter and speeds are usually faster to take care of the smaller-gage wire.

Fine-Wire Machines—Sizes finer than 0.030-inch are usually drawn by the wet process on specially designed machines. There are several types of these machines. The **tandem type** uses horizontal spindles and the drawing blocks are vertical. The wire is drawn through the first die and given a few wraps around the block and passes directly into the next die. The **step-cone type** uses horizontal spindles and has several blocks of different diameters mounted on each spindle, giving the appearance of a cone. The coarser wire is drawn on the smaller block and passes around an idler sheave and into the next die and onto the next larger block. The diameters of the blocks are designed to take care of the normal elongation of the wire. Another type is the **circular tandem type** in which the blocks are arranged in a circle, with a die between each block. Each block is geared to draw enough faster than the preceding block to take up the increase in length. The finishing block is usually eight inches in diameter, but many of

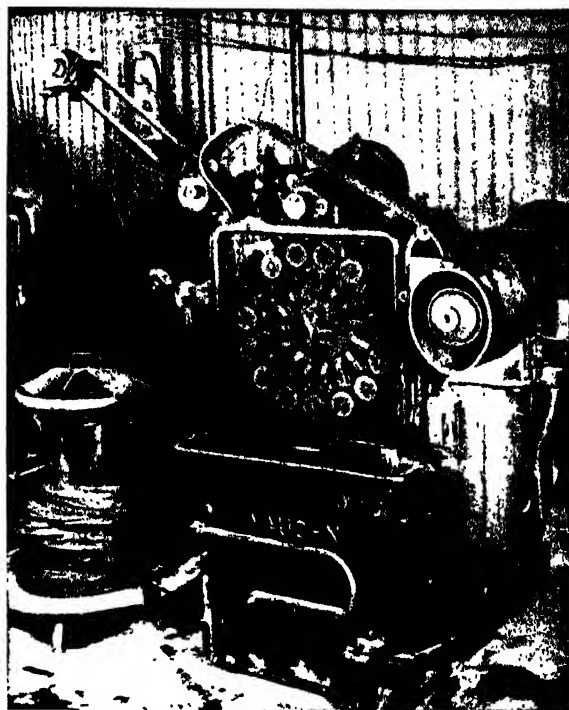


FIG. 40-17. General view of a circular-tandem type of fine-wire drawing machine. (Courtesy of Vaughn Machinery Co.)

these machines are equipped to take up the wire on spools.

In wet drawing, the drawing lubricant is a soap solution diluted with water. This solution is pumped from a tank on the machine or from a central tank supplying several machines. The solution is piped directly to a nozzle which sprays the lubricant into the bell of the drawing die. There is enough surplus to spray the drawing block also. In this manner the die, the wire, and the drawing block are kept well lubricated. The surplus lubricant falls to the bottom of the machine where it drains out to the supply tank or into a filter system for recirculation. Figure 40-16 shows a cone-type machine and Figure 40-17 shows the circular tandem type.

Drawing Frames—Before the development of continuous machines, all wire was drawn on what are known as frames, and a large percentage of wire still is drawn by this method. A frame is equipped with several wire-drawing blocks, all capable of being driven from a common shaft, either singly or in unison. The horizontal shaft extends the length of the machine, usually below floor level. From this shaft, vertical spindles are geared, and at the top of each spindle a drawing block is mounted. A separate clutch for each block permits the individual blocks to be started, operated or stopped without affecting the others on the frame. The blocks are usually 22 or 26 inches in diameter and the spindles set at about 5-foot centers. The blocks are about 25 to 30 inches above the floor level and all gearing and other moving parts are enclosed. Each block has a die holder or die box which also holds the drawing lubricant. Each block is also equipped with a pull-out mechanism, which is geared to the main shaft. In operation, the rod is placed on a pay-off reel and one end is pointed. This pointed end is then threaded through the die. The pointed end is then grasped by a pair of pincer jaws which are attached to the pull-out mech-

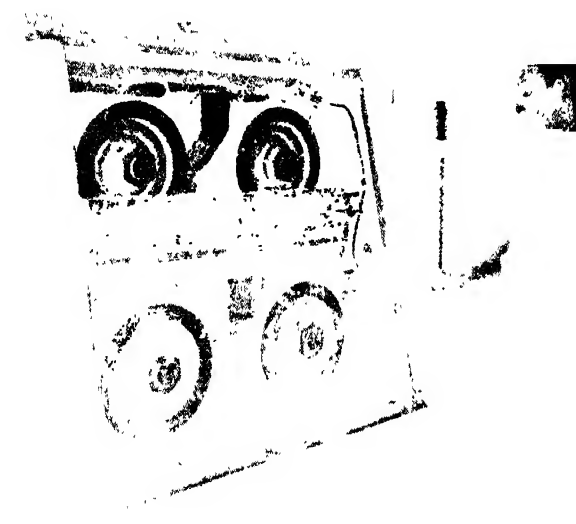


FIG. 40-16. General view of a cone-type fine-wire drawing machine.

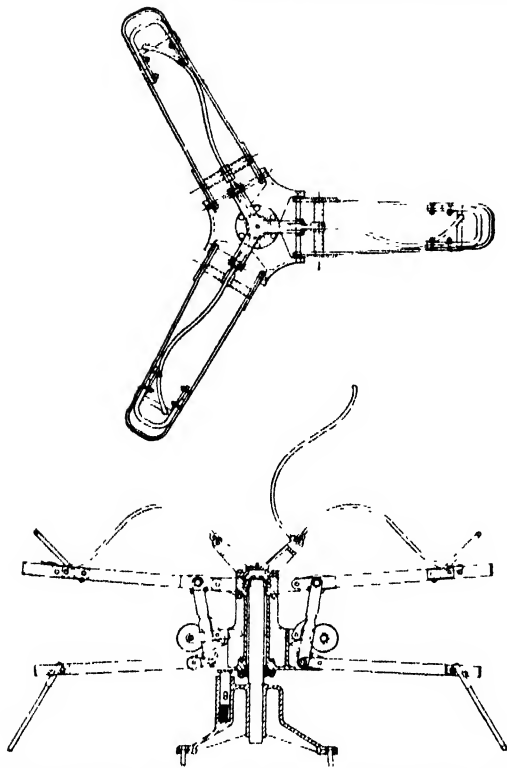


FIG. 40-18. Plan and elevation sketch of a three-arm pay-off reel. (Courtesy of Morgan Construction Co.)

anism. About three feet of wire is then pulled through the die. The pincer jaws are released and the wire is attached to the block by a clamp on the side of the block. The block is then started by attaching it to the spindle by actuating the jaw clutch which is operated by a foot lever. When the coil has been drawn, the block is declutched and the coil is removed by an overhead hoist. The coil is then placed on another pay-off reel and the process is repeated on another block until the required size is produced.

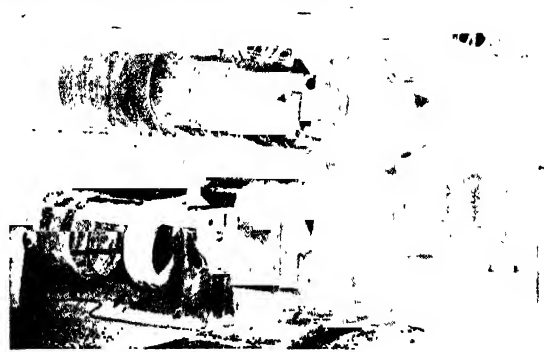


FIG. 40-19. Stem of rods from the cleaning house being loaded by tractor into the stem-type pay-off.

Obviously, there are disadvantages to this method. Each coil must be handled separately for each draft. All blocks on a given frame turn at the same speed and as the diameter of the wire is reduced, the length of the wire and, consequently, the running time increase, and more blocks must be provided for drawing wire to the smaller sizes in order to keep pace with the blocks drawing coarser wire through the early drafts.

AUXILIARY EQUIPMENT

Pay-Off Reels.—For rods $1\frac{1}{32}$ -inch and smaller, a flipper-type reel is used. This consists of two horizontal arms placed one above the other, both extending from a vertical standard. The arms are hinged so that they may be moved up or down. In use the arms are collapsed and the rod coil is suspended on the upper arm. Both arms are then moved to the horizontal position and the rod is held vertically. In this manner each convolution of the rod coil can be paid off or flipped and pulled up to the die. Some reels are made with three arms set at 120° . While one coil is paying off, the other arms are loaded and the ends are welded so there is no stoppage necessary between rod coils. Figure 40-18 shows a 3-arm reel. Another method is the stem pay-off, Figures 40-19 and 40-20. By this method the full stem of rods from the cleaning operation is placed on a special holder. A long bar or U-shaped rod is placed through the

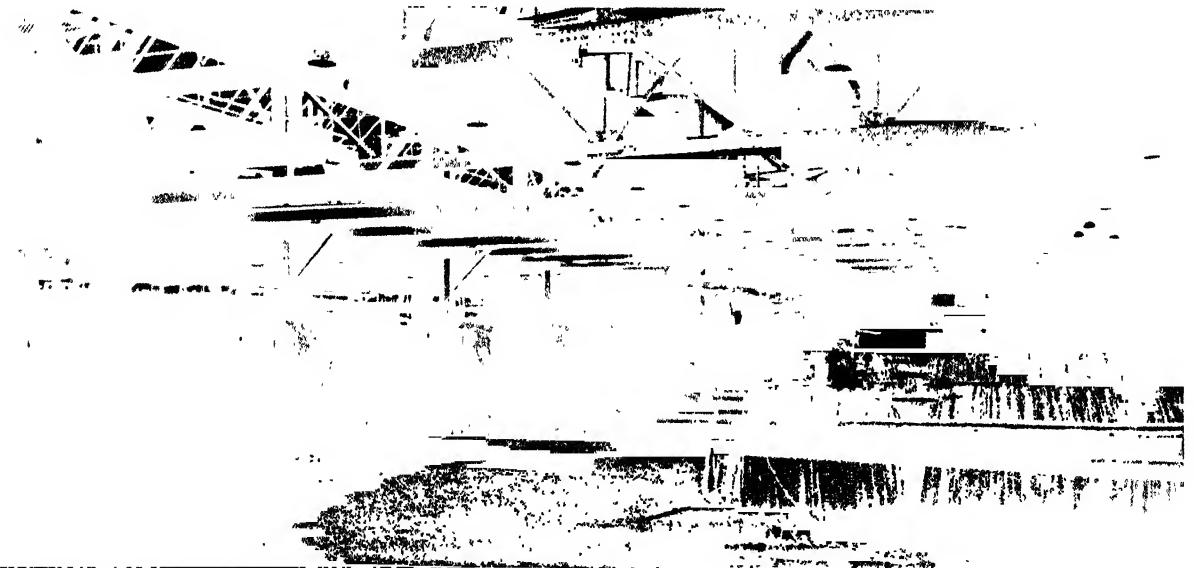


FIG. 40-20. Battery of wire-drawing machines (left) being fed by stem pay-offs in the right foreground.

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center of the coils and acts as a hold-back to keep the convolutions paying off uniformly. The coils are welded as in the flipper pay-off.

Welders—For continuous drawing, electric butt welders or flash welders are used. These have been developed to perform very well on high- and low-carbon steel. For welding high-carbon steel, the welders are equipped with annealing jaws to give the steel the proper structure for drawing after the weld is made. Grinders are also provided to quickly remove the "burr" or upset formed by the weld.

Safety Stop—All machines and frames are equipped with stops to cut off the power quickly in case of trouble. Between the flipper reel and the first die a snarl stop is placed which will stop the machine or block in case of a tangle or snarl before the rod is broken. All machines are equipped with bars around all working parts. By merely pushing against these bars the machine can be stopped. The motors are also equipped with dynamic brakes so that there is immediate response to the stop signal and there is no tendency for the machine to "coast."

Pointers—Several methods are used for pointing the rods or wire for the initial threading of the dies. The fine sizes are literally pulled apart and the "necking down" forms a point. The intermediate sizes are pointed by means of a "roll pointer" as shown in Figure 40—21. This consists of a pair of oscillating rolls with grooves of varying diameters. The rod end is worked down to smaller sizes by successive rolling in the grooves of decreasing size. The larger sizes are swaged or hammered on rotary swaging machines or are machined by specially designed cutting machines.

"Turks-Head" Shaped-Wire Drawing Machine—Common four-sided shaped wire, such as squares, oblongs or keystone shape, are frequently produced on cold-rolling equipment known as a "Turks-head" machine. This consists of four hardened-steel rolls set in planes at right angles to each other. The narrow face of the rolls, as set in the framework, is adjustable on the same plane so that the assembly of the overlapping roll edges facing each other will project the contour of the opening so formed, into the desired shape of the cross-section of the wire to be made. The process wire, of a size somewhat larger than the finished size desired, is pointed and pulled through the Turks-head, being thus rolled to shape and size, after which it is coiled on a regular wire-drawing take-up block.

Heating Effect in Wire Drawing—The plastic deformation involved in drawing wire will induce internal heat, and this rise in temperature of the wire will be dependent on the composition or hardness of the steel and the amount of cold work or reduction of cross-

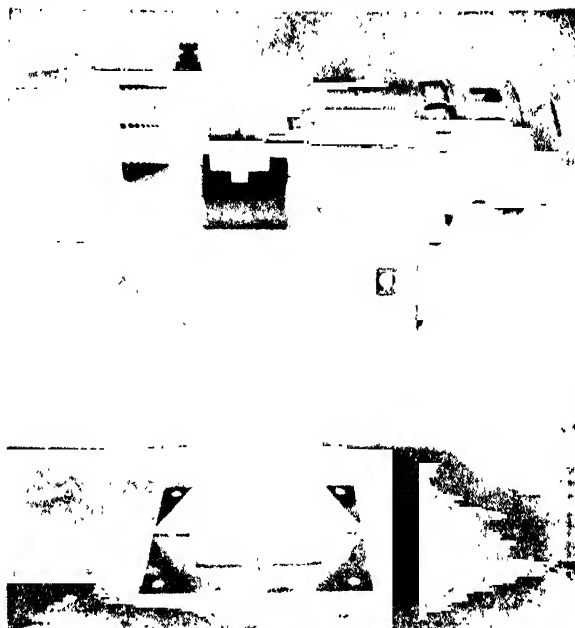


FIG. 40—21. "Roll pointer" for pointing rods or wire for initial threading through the dies. (Courtesy of Morgan Construction Co.)

sectional area of the wire. In the ordinary dry drawing of frame or individual drafting, the wire will cool somewhat between successive drafts, but on continuous drafting, the heating of the wire may be cumulative resulting in temperatures frequently detrimental to the quality of the wire. This rise in temperature may be restricted by several methods, such as air or water cooling of the wire-drawing capstans or blocks and the use of water-cooled dies, the latter being more of a secondary rather than a direct effect. The effect of the wire-drawing temperatures is not as detrimental to low-carbon steel wires as to high-carbon wires. The toughness and uniformity are adversely affected, and in drawing high-carbon wire of high tensile-strength requirement, by either single or continuous drafting, the temperature must be controlled and kept as low as possible. The efficiency of lubricants used in the wire-drawing die, and carried on the wire in continuous drafting, is adversely affected by excessive wire-drawing temperatures. Without the cooling devices, the present high wire-drawing speeds could not be attained and die life would be considerably reduced.

SECTION 7

WIRE-DRAWING PROCESSES AND OPERATIONS

Results of Cold Drawing—The results attainable by the wire-drawing process may be summarized as follows:

1. Metal may be elongated and reduced in section to an extent not attainable by other methods.

2. A greater degree of accuracy as to size and section can be attained than is possible by other methods excepting cold rolling, which is not applicable to common sizes of wire.

3. A uniformly smooth and highly polished surface can be produced.

4. The process serves as a test for the detection of hidden flaws in the metal. The fact that a wire has satisfactorily withstood the drawing operation may be taken as an indication that the metal was originally sound and free from defects liable to cause it to fail in service. This statement does not mean that the wire itself is free from all flaws or defects, for it is possible to produce certain flaws by improper drawing.

5. Finally, the process affects the mechanical properties of the metal, which fact, as noted under the heading of tempers, makes it possible, by employing this process in conjunction with heat treatment, to produce

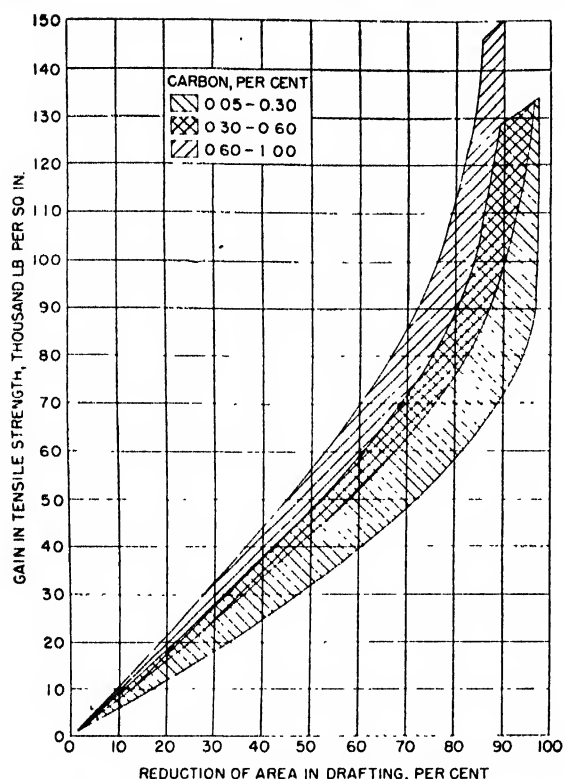


FIG. 40-22. Gain in tensile strength on drafting steels of carbon contents from 0.05 to 1.00 per cent.

many wires from the same steel having different mechanical properties.

Effect of Drawing Upon Mechanical Properties—Wire drawing, like any cold working of metals such as iron or copper, will increase the hardness, stiffness, tensile strength and elastic limit. The ductility, as indicated by the elongation and reduction of area, will be correspondingly decreased. The extent of these changes in



FIG. 40-24. Structure of 0.06 carbon-steel wire after one draft. Longitudinal section. Magnification: 500X.

mechanical properties is not always directly proportional to the amount of drafting or cold work done upon the metal, as it is affected by various factors such as the total amount of drafting, number of drafts, per cent draft per draft and the type of material itself. However, for each set of conditions, the change in mechanical properties has been determined, and the processing necessary to produce the required grade of wire may be regulated accordingly. The gain in tensile strength in drawing of various carbon steels under standard conditions is shown in Figure 40-22 and indicates the hardness increase as the cold reduction of the wire increases.

The Cause of These Changes—These changes in the characteristics of the metal brought about by cold working are to be attributed to the changes in grain



FIG. 40-23. Full-annealed 0.06 carbon-steel wire. Longitudinal section. Magnification: 500X.



FIG. 40-25. Longitudinal section of hard-drawn 0.15 carbon-steel wire. Magnification: 500X.



FIG. 40-26. Structure of 0.15 carbon-steel wire after short-time-cycle sub-critical anneal. Longitudinal section. Magnification: 500X.

structure such working produces. In a hot-rolled rod or in an annealed wire the grains have a polygonal form and are arranged about as shown in Figure 40-23. The microstructure of an annealed low-carbon steel wire is shown in the photomicrographs of Figures 40-26 and 40-27. All photomicrographs shown represent longitudinal sections, parallel to the direction of drafting. Figure 40-23 exhibits grains in the steel before drafting. Figure 40-24 shows the condition of the grains after the material has had one draft. Here it can be seen that, as the wire is being elongated while passing through the die, the grains in the steel actually become elongated, also. The grains elongate in the direction of the drafting and become correspondingly narrower at right angles to that direction. Figure 40-25 shows the

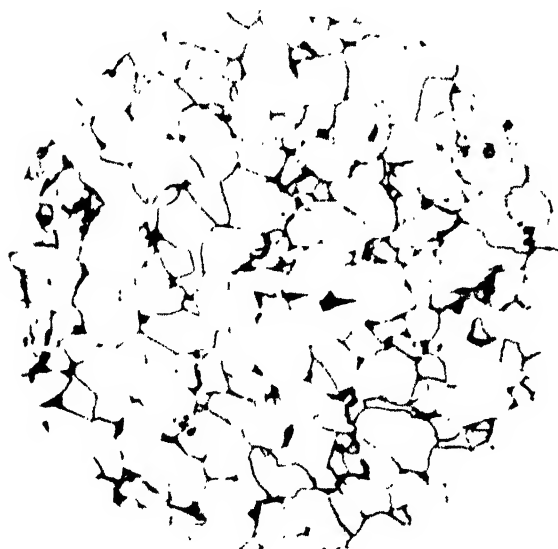


FIG. 40-27. Structure of 0.15 carbon-steel wire, full-annealed after hard drawing. Longitudinal section. Magnification: 500X.

same wire after having had a number of drafts. Here the grains have been elongated to a considerable extent. Due to the stretching of the wire in drawing and the crushing effect produced by the pressure exerted by the die in all directions towards the center of the wire, the structure has become so altered that it is difficult, if not impossible, to locate the exact boundaries of the individual grains in their strained and distorted condition.

Limitations of Drawing—Steel may be drawn several drafts further than the state illustrated by Figure 40-25, but when wire has been put into this highly strained condition by drawing, it loses its ductility to such a degree that it is not practicable to submit it to further drawing, and if required to be reduced in section still more, it becomes necessary first to restore the grains to the form and arrangement characteristic of the unworked (not strained) condition. This changing back of the grains to a non-distorted form simultaneously restores the properties that existed in the wire before it was subjected to drawing. The processes employed for restoring the grain to strain-free formations are annealing and patenting. The effects of these processes will be discussed in the section on "Heat Treatment of Wire." Enough having been said to explain the reasons for them, some of the chief features of wire-drawing practice will now be considered.

DRY DRAWING

Dry Drawing Low-Carbon Coarse Wires—Dry drawn wires may be sub-divided into three groups according to the number of drafts given them. The first group includes all one-draft wire, normally in 8 gage and coarser sizes. These wires may be used for the fabrication of cold-forged parts, such as bolts and rivets, or formed materials as wire screens, chains, etc. The second group consists of two-, three- or four-draft wire in sizes 9 gage to about 14 gage. These sizes are common wire sizes such as usually required for hardware purposes, galvanized wire for fencing and telephone lines as well as bright wire for the manufacturing of coat hangers, nails, steel wool and similar products. Some of this group of wire will be used for process wire to be heat treated and drawn to finer sizes. The third group consists of five-, six- and seven-draft wire in sizes 15 gage to about 18 gage. In addition to being used as manufacturing wire, it is largely used as process wire, for which latter purpose it is annealed for drawing to fine-wire sizes, since 18 gage is usually considered to be the smallest size wire normally drawn from a $\frac{1}{2}$ -inch rod without annealing.

Drawing Single-Draft Coarse Wires—On single-draft wires, as light a coating and lubricant is used as is possible to meet the needs of each case, the exact nature of both coating and lubricant depending upon several conditions, such as size and kind of stock, uses to which the wire is to be put, etc. These wires are generally drawn on 22-inch blocks. Most mills now provide "block strippers" which are small electric or hand-operated hoists erected over the blocks, for lifting the heavy bundles off the blocks and onto buggies. The usual speed of drafting on this class of product is from 25 to 60 r.p.m., or 144 to 346 feet per minute. The reduction in size of this one-draft wire is usually about $\frac{1}{32}$ inch in diameter, and varies according to the purpose to which it is to be put. For bright finish wire, the principal object of the drawing is to remove the inequalities in dimension and finish of the hot-rolled rod and at the same time to retain, as far as possible, the mechanical properties characteristic of the hot-rolled steel. If the light draft necessary to accomplish the first result

renders the wire too hard for the purpose intended, the requisite softness in the finished wire is obtained by using a hot-rolled steel with very low carbon content or an annealed wire as the starting material.

Drawing Two-, Three-, and Four-Draft Wires—In this second group of dry drawn wires are found practically all the process wire for finer sizes and the largest tonnages of finished wire. For the two-draft wire, the rod may be given a medium lime coating with or without sll, according to the finish desired; while for three- and four-draft wires, a medium or heavy lime coating is needed on the rod. The lubricant for the first draft should be of such character and consistency that all succeeding drafts may be effected without damage to the wire or die. The reduction per draft, the speed of drawing, and other such features will vary for this class of wire, as well as for other kinds, with different companies and in different mills, but the following example represents conservative practices: Starting with a $\frac{1}{32}$ -inch low-carbon rod, this would be drawn to a No. 8 $\frac{1}{4}$ wire on the first draft, to a No. 10 $\frac{1}{2}$ wire on the second draft, to a No. 12 on the third draft, and to a No. 13 $\frac{1}{2}$ on the fourth draft—all on 22-inch blocks. The corresponding percentage reductions would be approximately 41, 35, 32 and 34, for each draft.

Drawing Five- and Six-Draft Wires—The third group of wires are given their finish drafts on 16-inch blocks, which run at a peripheral speed of 160 to 272 feet per minute on frame drawing, or about 1200 to 1500 feet per minute on continuous machines, depending on the composition of the steel. In this group are found the hard-drawn wires which have already received three or four drafts on 22-inch blocks, and are to receive one, two or three drafts to No. 15, 16 $\frac{1}{4}$, or 17 $\frac{1}{2}$ gage, as well as wire drawn five or six drafts direct to finished size on the 16-inch continuous machines.

Process wire, which was annealed at 13 or 13 $\frac{1}{2}$ gage, may also be drawn five or more drafts to about 20 gage, finishing on 16-inch blocks.

Drawing No. 14, 15 and 16 Wires—In addition to the wires discussed in the two preceding paragraphs, there is a relatively large tonnage of wires drawn four, five and six drafts from the rod on 22-inch blocks without process annealing. The most important of such wires are No. 14, drawn four drafts from the rod on 22-inch blocks, and five- and six-draft No. 15 and No. 16, drawn the same way.

Drawing High-Carbon, or Special Wires—The class here referred to as high-carbon wires is made up of a great variety of wires used for many different purposes. These wires include the four different grades of rope wire, the various kinds of spring wire, as well as music wire, spoke wire, brush wire, card wire, pile wire, pivot wire, safety pin wire, sewing-machine needle wire, phonograph needle wire, and various other highly specialized products. Because these wires are made to specifications somewhat out of the ordinary, either as to finish or mechanical properties, requiring special treatment at some stage in their manufacture, they are collectively known as special wires. As to the method of drawing, the actual processes for drawing these wires do not differ from those for drawing low-carbon wires except as to details, and such differences as do exist are due to the greater care that must be exercised with the high-carbon wires. These differences may be summarized as follows:

1. In the case of certain products, such as music wire and some grades of rope wire, the rods are patented before they are drawn. This treatment refines the grain structure of the rods, homogenizes the structure, and provides better drafting qualities.

2. The cleaning of high-carbon rods or wire requires greater care to avoid "smutty" surfaces and to prevent acid brittleness, which has a more harmful effect upon high-carbon than upon low-carbon wires.

3. The coating must be heavier and more uniform on the high-carbon stocks in order to prevent scratching of the wire and cutting out of the dies.

4. A longer time of baking is required to counteract the effects of the cleaning acid.

5. As a lubricant for drawing high-carbon wires, a specially prepared soap powder is used, as the very best lubricant possible is necessary for satisfactory results in drawing.

6. The speed of drawing and the amount of reduction per draft must be carefully regulated to prevent the structure of the wire from being harmfully affected by the drawing.

7. High-carbon process wires are usually patented instead of annealed, whenever it is necessary to remove the hardening effect of previous drafting. This special heat treatment (which will be described later) provides, in the case of high-carbon wires, better drawing qualities and produces a much stronger and tougher finished wire than annealing. This treatment, when followed by proper drafting, gives the highest possible strength. High-carbon wire is spheroidized annealed if maximum softness is required.

8. As is well known, the higher carbon steels are more sensitive to heat treatment than the low-carbon steels, hence, they require correspondingly greater care in heating and cooling. Not only must too high temperatures in heating be avoided, but care must be taken to maintain uniformity in temperatures and rate of cooling in order to obtain uniformity in the finished product, which is particularly important and desirable in the case of the uses to which most high-carbon wires are put.

WET DRAWING

As previously stated, wet or liquor-finish drawing is applied to a large percentage of wires smaller than 20 gage and to all coarser wires requiring a liquor finish, and is distinguished from dry drawing by the kind of coating given the wires and the fact that they are drawn wet from a vat of fermented rye-meal liquor or a substitute alkaline soap solution. As to the actual operation of drawing, the wet process does not differ from the dry, except that little or no lubrication is needed other than that furnished by the metallic coatings and the liquor. The percentage reduction per draft, however, is materially less in wet than in dry drawing, and of the two processes, wet drawing is the more expensive. For this reason it is the practice, in the case of coarse wires, to draw the wire dry to within one size or a half size of the finish before coating and drawing wet to finish. The size to which fine wires are drawn dry before coating and drawing wet depends upon the size of the finished wire and the purpose for which it is to be used, but generally the dry drawing will stop at about No. 17 or No. 20 gage, the latter representing the usual limit for dry drawing.

Manner of Handling Material for Wet Drawing—After being cleaned, all wire for wet drawing is delivered to the wet-wire department immersed in water contained in barrels or tubs, where it remains until the mill is ready to draw it. Just before drawing, each coil of wire is removed from the water and dipped into a coating solution of the correct composition to give the kind of coating desired. If the wire is to be finished by single drafts, the coil is then placed on a reel set in a tub containing the fermented rye-meal liquor, from

which it is drawn wet through a die of correct size and kind by a block of proper size and strength. If the wire is to receive several wet drafts, the drawing may be done on a continuous machine. These machines give very satisfactory service in wet drawing, because the drums may be partly submerged in the liquor which keeps the wire cool. While the coating and liquor are generally relied upon to supply the necessary lubrication, tallow occasionally will be rubbed on each coil at one or two places after coating, and large sizes of wire are sometimes dipped in soap water, to supply additional lubrication. As to the composition of the coatings, the brass (copper and tin) coatings give a better wearing or drawing surface than the pure copper coating, and these coatings are, therefore, used in all liquor drawing, especially of high-carbon wires, unless a copper-colored finish is required. The drawing operations have the effect of burnishing the coatings so that the wire has a fine finish, but, while these metallic coatings cover the wire completely, they are extremely thin and cannot be relied upon to give much protection against corrosion.

Number and Sequence of Operations for Fine-Wire Drawing—Previous discussions have indicated that a great number of operations are required to produce a wire of very small size, and also that these operations vary in number, type and sequence according to the size of the wire, the grade of steel, and fabrication requirements. No one kind of wire, therefore, can be taken as typical of all fine wires and space is not here available for describing in detail the manufacture of even a few of the more common wires. However, it seems desirable at least to give some idea of the great amount of work required to produce these wires, and this may best be done by an example. For this purpose, a No. 34 gage (0.0104-inch) low-carbon wire has been selected which, as fine wire goes, is produced in relatively large tonnages. The steel used is usually a low-carbon basic open-hearth steel of a type which lends itself to continued heavy drafting and the severe plastic deformation involved in such section reduction. After this steel has been properly made in the open-hearth furnace, cast into ingots, and soaked, it is rolled into billets, which are reheated and rolled into the standard $\frac{7}{32}$ -inch (0.218-inch) rod, each rod coil weighing around 600 pounds as rolled from the more modern continuous hot mills. This rod is cleaned, lime coated, and baked, after which operation it is dry drawn about five drafts to 14 gage (0.080-inch) on 22-inch blocks, usually on continuous-drawing machines, at 1500 feet per minute. It may be annealed or not, depending on further processing, drawn six drafts to 20 gage (0.0348-inch) on 16-inch blocks, usually at 1500 feet per minute on continuous machines. Coils may be split at the machines and then are bright annealed in pots under controlled atmosphere, cleaned and metal coated, placed on pay-off reels and drawn wet about twelve drafts on cone-type continuous drawing machines to 34-gage.

Tungsten carbide dies are used on all of the twenty-three drafts. The wire, in 25-pound coils, is next inspected and tested, when it may be shipped as bright wire without further work, or be tinned, galvanized, or annealed as required by the purpose for which it is intended. Low-carbon wires much coarser than No. 34 gage, may be given but a single anneal in process, usually between the dry and wet drawing, or the two annealings may be located at different points from those cited above, while wires much finer than No. 34 size may be given three annealings in process.

Drawing Copper Wire—While this chapter is concerned only with the drawing of steel wire, a few words

concerning the drawing of copper wire should be included, perhaps, especially since many of the wire manufacturers regularly draw wire from both of these metals. As to the actual drawing operation, the drawing of copper wire differs from that of steel wire only in minor details. Copper rod or wire is never coated. It is merely dip-cleaned in sulphuric acid to remove the oxide, then washed with a hose and dipped into hot water, the last operation being employed so that it will dry quickly. Fine sizes in coils, however, may require an additional operation, as they are usually dried in bake ovens. As the lubricant for the dry drawing of copper wire, tallow or tallow mixed with a drawing compound is used. To anneal copper wire after hard drawing it is heated to redness and cooled either rapidly or slowly. Most of the copper-oxide scale can be removed by quenching red-hot copper wire in cold water.

Precautions in Drawing—Proper Alignment of the Die—The correct setting of a die is an important part of a wire-drawer's work, to which too much attention cannot be paid. As indicated in Figure 40—28, the die holder is fastened to the frame by a bolt allowing a lateral swinging movement. The pull of the wire is on a line from the center of the bolt to the fillet at the base of the block, and the face of the die should be at right angles to this center line. The hole through which the wire is drawn should be so in line that the center line of the hole will coincide with the pull of the wire, or with a line drawn from the center of the bolt holding the die box at a tangent to the block. Suppose the wire-drawer carelessly uses a hole at one side of the center. The die box swings around, but if the center line of the hole does not coincide with the pull of the wire, off-sized or scratched wire will be produced, and the die will speedily wear out. The sketches of Figure 40—28 show these two conditions. The upper sketch shows two positions of the die holder. The position shown by the solid line is assumed when wire is being drawn through

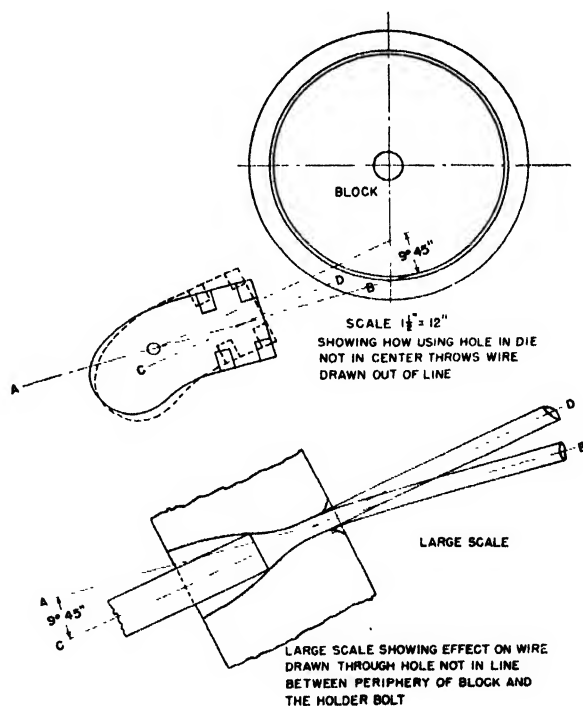


FIG. 40—28. Sketches illustrating the importance of proper alignment of the die in wire drawing.

a hole of a die which is placed correctly in the holder. The line CD represents the center line of another hole in the same die. If the wire drawer starts to draw a piece of wire through hole CD without moving the die to bring it into a central position in the holder, the holder will assume the position shown in dotted lines and its center line will not be tangent to the block and center line CD will then not be tangent to the wire-drawing block. The effect upon the wire is illustrated in the larger scale drawing. The wire is there shown drawing off the lower edge of the die, and thus off-sized wire will be produced, and the life of the die will be shortened. The die should also be set in an upright position. Sometimes the clamping screw is not in just the proper position, and a bar of steel is used to shim it up, which is allowable only if properly used. The die should be lined up a little below the fillet of the block. This will permit the leading of the wire on the block in such a way as to cause it to slide up properly. Some die holders are equipped with a hinged joint to permit a motion in a

nearly vertical plane. This is to overcome the tendency to pull hard on the upper side of the hole when the block is first started. This tendency is due to the fact that the vise is usually on the top of the block and the pull is upward.

Drawing Limits and Tolerances—Although every effort is made to draw the wire as true to the required shape and size as possible, exactness in these respects is almost impossible under commercial conditions, and all wire produced under such conditions will vary somewhat in diameter and section. These variations may be due to a varying degree of hardness in the metal being drawn, to somewhat rapid wearing of the die, or, in the case of wet-drawn wires, to unsuitable or imperfect coatings. Some tolerance is necessary therefore, but it is important that these tolerances be kept within certain limits defined by the use to which the wire is to be put. The tolerances used by the wire industry are divided into four groups of allowable variations in the diameter of the common grades of round wire, other than special grades, and designated as **standard**, **semi-special**, **special**, and **extra special drawing**. Each of these sets of tolerances contains the variations allowable, which are adjusted to the size of the wire as indicated in Table 40—III. The standard tolerances are those adopted by the American Iron and Steel Institute (AISI) and are given in their Steel Products Manual, Carbon-Steel Wire, Section 16.

All wire is gaged carefully in two or three places around the circumference, just after the first end of the coil is pulled through the die and at the finished end to insure that it is of the correct size and shape. Wire ordered by gage number, by decimal, or by fraction of an inch, is gaged with a micrometer gage.

Table 40—III. Standard Size Tolerances for Wire

Size Tolerances for Uncoated Coarse Round Wire in Coils

Size, in.	Tolerance, Plus and Minus
0.500 and larger	0.003
Under 0.500 to 0.076, incl.	0.002
Under 0.076 to 0.035, incl.	0.001

Out-of-round is customarily one-half the total gauge tolerance.

Size Tolerances for Coarse Round Galvanized Wire in Coils

Size, in.	Tolerances, Plus and Minus, in.			
	Regular Coating	Type 1 Coating	Type 2 Coating	Type 3 Coating
0.500 to 0.251, incl.	0.004	0.004	0.005	0.006
0.250 to 0.148, incl.	0.003	0.003	0.004	0.005
0.147 to 0.076, incl.	0.003	0.003	0.003	0.004
0.075 to 0.035, incl.	0.002	0.002	0.002	0.003

Size Tolerances for Uncoated Fine Round Wire in Coils *

Size, in.	Tolerances, Plus and Minus, in.
0.0625/0.0348	0.001
0.0347/0.0271	0.0008
0.0270/0.0200	0.0006
0.0199/0.0151	0.0005
0.0150/0.0101	0.0004
0.0100/0.0060	0.0003
0.0059/0.0044	0.0002

Size Tolerances for Galvanized Fine Round Wire in Coils *

Size, in.	Standard Tolerances, in.			
	Type 1 Coating		Type 3 Coating	
	Plus	Minus	Plus	Minus
0.0625/0.0348	0.0015	0.0015	0.002	0.002
0.0347/0.0271	0.0015	0.0010	0.002	0.001
0.0270/0.0200	0.0013	0.0005	0.0015	0.001
0.0199/0.0151	0.0010	0.0005	—	—
0.0150/0.0101	0.0008	0.0004	—	—
0.0100/0.0060	0.0005	0.0003	—	—

* These tolerances do not apply to special wires which have been annealed as a separate operation following cold drawing or immediately prior to coating.

SPECIAL FINISHING OPERATIONS

Straightening and Cutting Wire—With the exception of the largest sizes, all wire, both round and shape, is drawn on a block and, therefore, at the finished size, is still in the form of a coil. But for certain purposes it is desired to have the wire furnished in short straight lengths, rather than in coils. This straightening and cutting work is usually done on some type of machine.

Whirls—For common round wire, automatic machines are employed for straightening and cutting the wire. These machines are required to perform three operations simultaneously or in very rapid succession, namely, pull the wire forward, straighten it, and cut it into the lengths desired. The mechanism for straightening the wire is known as a **head or whirl**. It contains a number of staggered dies which bend the wire slightly, as it passes through, in reverse directions so as to remove the bends and kinks and leave it straight. The wire to be straightened is automatically fed through the whirl and onto an apron, where it actuates a mechanically or electrically driven cutting tool which cuts the wire into uniform short pieces of the length desired.

Roll Straighteners—For shapes and flats the roll straightener is used. This machine consists of a set of vertical and also of horizontal rolls which can be so adjusted that pressure can be brought to bear on the sides, top and bottom of the wire. On these machines the first sets of rolls put a considerable bend in the material to remove the kinks, and succeeding sets of rolls reverse this bend just enough to leave the wire perfectly straight. Squares, rectangles and hexagons, narrow flats that are not too thin, most shape wires, and also very fine wires, can be straightened on this type of machine provided the machine is properly designed for them.

Stretching Machines—A process of straightening by

stretching is used on some fine soft annealed wire. The usual products straightened in this manner are copper wire, which may be as coarse as No. 15; tinned tag and florist wires, from No. 20 to No. 35 gage; and soft annealed wire. To straighten such wire, it is wound onto a wheel about 8 feet in diameter, and the coil is then slipped off and placed over two lugs on a hydraulic machine. When the ram of the machine is set in motion one of the lugs moves away from the other, first drawing the coils out straight, then stretching the strands some three or four inches. The stretched and straightened strands, which may number as high as 1,000 on the smaller sizes, are then cut to length. This process does not produce wires that are so straight or that are so exact and uniform in length as the other methods.

INSPECTION AND TESTING

Importance of Inspection—Of equal importance with the various processes in making wire is the thorough system of inspection and testing to which the material is subjected during its progress through the mill and after finishing. Beginning with the hot-rolled rod, the material undergoes frequent and careful inspection during manufacture and is subjected to whatever tests are needful to determine its fitness for the service required.

Final Tests on Wires—Wire for shipment to customers is regularly inspected and tested at both ends of each bundle for size, finish and temper. For temper, small sizes are tested by a kink test, medium sizes by a bending test, large sizes by a nick-and-break test. The better grades of wire are subjected to a machine test for tensile strength and elongation, and some of it may be given severe torsion tests. All telephone and telegraph stock is tested also for conductivity. All exact-weight bundles are net weight of wire, excluding wrapping material. Wet-drawn wires are invariably prepared; that is, bundles are covered with paper, but they may also be wrapped in burlap when so ordered.

Defects in Wire—After the wire has been drawn, it may be found defective or unsuitable for the purpose for which it was intended, and every effort is made to detect and discard all such wire. The chief sources of trouble can usually be placed under one of the following headings: off size or shape, internal defects, surface defects, and improper mechanical properties.

Size and Shape—The reasons for wires being off size or shape have been already given. While every effort is made to draw the wire to the tolerances given, the tolerances themselves may be too great to permit use of the wire in an exacting application. Where extreme dimensional accuracy is needed, proper specifications should be established before the wire is drawn so that product can be made to suit the special requirements.

Internal Defects—If the steel from which the wire is made contains pipe or seams, these will remain in the wire, but are usually so minute that they cannot be found by any of the ordinary means of inspection. Any slight segregation in the steel will also induce defects in the wire. This segregation causes uneven temper and is also thought to induce, at times, a defect known as "cupping." A "cupped" wire is one the fracture of which exhibits the cup and cone form. This defect, it is believed, may also be caused by improper drafting. Such hidden defects are very difficult to detect and frequently cannot be found except by actual use of the wire.

Surface Defects—Sometimes the finish of the wire does not meet requirements. If the finish is not satisfactory it may be due to poor cleaning, to the use of improper coatings or other causes. In the case of liquor-finished wires, especially, the color may not be of the particular shade desired, or it may vary too much in different bundles. When these features are of special importance, they should be stated on the order, so that special attention may be given to meeting requirements in this respect. Other surface defects in wire are scratches, slivers, and seams. Scratches are caused by using a die in poor condition, or they may result from improper lubrication or from pieces of metal or other gritty substances being drawn into the die with the wire. Slivers, as the name indicates, are sharp pointed projections of metal that rise from the surface of the wire. They may be caused from any one of a great number of sources, including the ingot and rolling practice. Seams are longitudinal cracks in the surface of the wire and are generally traceable to lack of proper care in rolling. Therefore, in the wire they are very small, and are sometimes very difficult to detect.

Mechanical Properties—The mechanical properties of any wire will depend upon the chemical composition and quality of the steel as well as the exact nature of the drafting practice and of the heat treatment it has undergone. The particular application or method of fabrication determines the grade of wire necessary. A finished wire taken at random will not be suitable for all purposes, as the strength and ductility may vary from a 50,000 lb. per sq. in. tensile strength for an annealed low-carbon tie wire to 350,000 lb. per sq. in. tensile strength for a high-carbon music wire. If the wire manufacturer is fully informed both as to the desired properties of the wire and its application, the requirements can be met by proper selection of composition, heat treatment and predetermined amount of hardening by cold working in wire drawing. Many products such as upholstery springs, rope, telegraph wire, etc., are recognized in the industry as regular grades of wire for standard applications, being covered by standard industry specifications, and may be ordered as such.

SECTION 8

HEAT TREATMENT OF WIRE

Heat-Treating Processes used in the wire industry include subcritical or process annealing, patenting, and hardening and tempering. Of these, patenting may be described as a special toughening process used only in the wire industry, while annealing, hardening, and tempering play such an important role in the industry and the methods of applying these processes have become so highly specialized that, although their general principles and their application to other branches of the steel industry have been thoroughly discussed in other chapters of this book, a brief account of their

specific application to the wire industry is included in this chapter. These processes are briefly described and discussed in the order of their importance, which is the order in which they were first mentioned above.

Importance and Purposes of Annealing—Of the three wire heat treatments mentioned above, subcritical or process annealing is the most common, and it is practically the only heat treatment given to the soft, or low-carbon, steel wires, which constitute by far the bulk of wire production. In general the process is employed in the wire industry to accomplish any one of the following

objectives: (1) to refine, soften, and make uniform the grain structure of rods or wire; (2) to obtain a specific structure in, and thus impart special properties to, process wire or the finished material, and (3) to soften the wire after cold-working, i.e., drawing or cold-rolling.

Normalizing (Cycle Annealing)—Normalizing consists of heating the rods or wire to above the transformation temperature and cooling in still air at room temperature. This practice is usually followed when treating medium-carbon steels of about 0.25 to 0.45 per cent carbon to obtain uniformity of structure and hardness. Normalizing imparts the proper ductility or flowability as required for certain cold upsetting and forging applications, and much of the medium-carbon bolt wire is therefore normalized in the regular course of manufacture. This normalizing process consists of passing full coils of rods or wire on a moving conveyor through a furnace made up of three zones, "preheating," "soaking" and "cooling" zones, the temperatures of which are automatically controlled. In the soaking zone, a temperature of approximately 1600° F is maintained. Combinations of several variables permit variations in the heating and cooling cycles and the cooling rate is usually retarded. By the use of controlled atmospheres in the furnace, decarburization and scaling may be held to a minimum.

Annealing for Definite Structures—This type of annealing is applied to the higher carbon steel wires, i.e., those with a carbon content of 0.40 per cent or more, in order to obtain certain definite structures, and is similar to the treatment given carbon tool steels and other high-carbon steels in order to make them easy to machine. If steel having a pearlitic (lamellar) structure is heated to 1300° F (704° C), and is held at that temperature for a sufficiently long period of time, the iron carbides slowly coalesce, changing from a platelike form to a granular, globular, or spheroidal form. By properly regulating the temperature of the treatment, structures intermediate between these two extremes and best suited to the purpose for which the wire is to be used may be obtained with temperatures in excess of 1330° F yielding a correspondingly greater percentage of pearlite as the temperature increases. These structures are sought in high-carbon wires mainly to enhance their machinability. Close regulation of the temperature is imperative in this type of annealing, for in heating the steel to a point so near the critical range, a difference of 50° F in the temperature often means the difference between obtaining the spheroidized carbide structure sought and the lamellar (pearlitic) structure which it is desired to efface.

Process Annealing to Soften Hard-Drawn Wires—In the previous section, it was pointed out that each successive draft in drawing a wire has a certain hardening effect upon the metal; that the grains become elongated in the direction of drafting; that, as the drafts are repeated and the wire becomes progressively harder, a point is reached at which the wire will break if subjected to further drafting; and that well before such a condition of brittleness is reached, it is necessary to heat-treat the wire in order to put it in a condition that will permit further processing without injury. For reasons which have already been stated or will be made apparent in the discussions to follow, high-carbon wires are generally patented, while low-carbon wires are annealed.

Process Annealing—For softening wires in process of manufacture, process annealing at sub-critical temperatures is usually employed. This type of annealing is used mainly to restore the ductility of hard-drawn low-carbon wires at process size so that they may be

drawn to finer sizes, although it is also employed to adjust the mechanical properties of some wires at finished size as required by their application. To accomplish these results, it is only necessary to heat the wire to below the lower critical range, as a temperature between 1000° F and 1250° F is sufficient to soften the steel to such an extent that the material is almost as ductile as it was before any drafting was done. The reason for this is illustrated by a comparison of the photomicrographs of hard-drawn and annealed wire in Figures 40—23 to 40—27. Figure 40—23 shows the form, arrangement, and size of the grains in the full annealed steel. In this condition, this steel exhibits a tensile strength of approximately 48,000 lb. per sq. in. and an elongation of approximately 33 per cent in 10 inches. Figure 40—24 shows the structure after a single draft; the grains are somewhat elongated in the direction of drafting. Figure 40—25 shows the structure of the wire after it has had about five drafts, and it will be observed that the grains have been generally elongated into a stringer-like or fibrous structure, greatly changed from the original equiaxed grains of Figure 40—23. This wire would have a tensile strength of about 125,000 lb. per

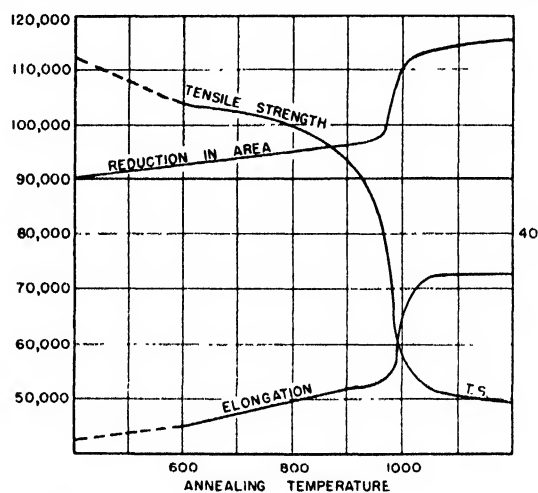


FIG. 40—29. Curves showing changes in mechanical properties of hard-drawn low-carbon steel wire after annealing at various temperatures up to 1200° F.

sq. in. and an elongation of about one per cent. At an elevated temperature, such as 1000° F, the grains recrystallize, thereby forming new grains. The greater the amount of plastic deformation, the lower the recrystallization temperature. Such incipient recrystallization or forming of new grains is indicated in Figure 40—30. Grain size is shown in the photomicrographs to increase as the temperature of annealing and the time the wire is held at temperature increase. The grain size obtained after the long time annealing cycle and recrystallization at 1200° F is similar to that of the original structure, although certain remnants of the structure produced by drawing still remain, with some residual directional properties.

The mechanical properties obtained by annealing hard-drawn low-carbon wires at various temperatures up to 1200° F are shown in the graphs of Figure 40—29.

Size of Grains—While the preceding paragraph shows the effect of grain distortion upon the mechanical properties of steel, nothing is said about the effect of grain size on these properties, and in all annealing, as well as other forms of heat treatment, the matter of grain size is of considerable importance. Large grains lower

A. Hard drawn wire.

E. Annealed 900° F, 1 hour.

I. Annealed 1050° F, 1 hour.

B. Annealed 600° F, 1 hour.

F. Annealed 950° F, 1 hour.

J. Annealed 1100° F, 1 hour.

C. Annealed 800° F, 1 hour.

G. Annealed 975° F, 1 hour.

K. Annealed 1150° F, 1 hour.

D. Annealed 850° F, 1 hour.

H. Annealed 1000° F, 1 hour.

L. Annealed 1200° F, 1 hour.

FIG. 40-30. Series of 12 photomicrographs illustrating the changes in microstructure of a hard-drawn wire after annealing at various temperatures. All longitudinal sections. Magnification: 1000X.

the tensile properties of the metal somewhat, while small grains make it not only stronger but tougher. The reasons for this, as well as the effects of cold working, are to be found in the structure of the grains themselves. Structurally, each grain is in reality a small crystal in which the atoms of which it is composed are arranged in regular rows and "layers," as described on pages 11 through 15. Crystals deform through movement of adjacent parts along definite planes, called slip planes, whose location and direction within the crystal are determined by the positions of the atoms within the space lattice. Anything that interferes with this movement will make the metal more resistant to deformation, while anything that promotes slip will lower its resistance to deformation. Small grains offer greater interference to slip than large grains, because the slip planes in different grains lie at different angles, and in any one of such small grains the slip cannot progress far before it is stopped by an adjacent grain. Each grain offers considerable resistance to the force causing the slip, because the direction of possible slip movement may differ greatly from the direction of the applied force. In a small section, such as a wire, where the ratio of the diameter of the wire itself to that of the crystals is relatively small, the effect of large grains on lowering the tensile properties is likely to be very noticeable. The increase of strength due to drawing may also be explained from the standpoint of slip interference, for, as the grains are distorted, the slip interference becomes greater and greater until most of the grains have been drawn into parallel stringers and there are no more planes of easy slip on which further motion can take place. In this condition, the force required to pull the wire apart should approach the force of attraction of

the molecules or atoms themselves, since the wire no longer can adjust itself to the load by deforming, but it never does because some of the grains begin to break before the others. The resulting ferrite grain size is referred to as "structural grain size," and is not to be confused with austenitic grain size.

The Conditions for Process Annealing involve three factors, namely, strain, time, and temperature. Without strain no change in the crystalline structure takes place on heating below the critical range. As to the effect of the amount of the strain in causing grain growth, or rearrangement, below the critical range, Sauveur has shown that, in the case of steel containing from 0.04 to 0.12 per cent carbon, very slight straining followed by annealing at 1200° F for 7 hours produces no grain growth, but that moderate straining (approximately 15 per cent), such as might be effected by a single light draft, and annealing cause the grains to grow to very large size resulting in "orange peel" surface if the wire is deformed; over-straining, or severe distortion, and annealing at the same temperature and for the same length of time produce grains of small size, comparable with the structures shown in Figures 40—23 and 40—24. The condition of strain producing large grains is seldom met with in process annealing of wire, because the grains of such wires are usually distorted far beyond that point. However, when a consumer expects to anneal wire, this fact should be made known to the manufacturer; it will then be possible, except in a few cases, to supply wire that will not form large grains under any annealing conditions. In the case of pure or almost pure iron, and steel containing more than 0.12 or 0.15 per cent carbon, the amount of strain, according to Sauveur, has no effect in producing large grains on an-

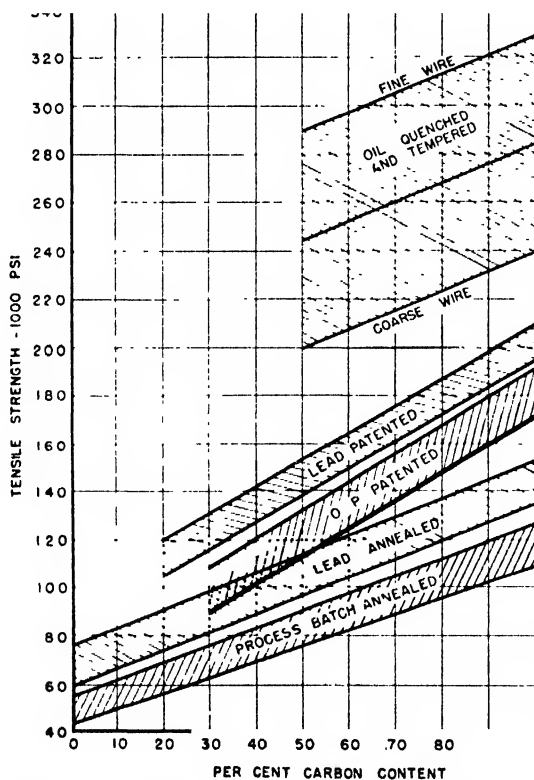


Fig. 40—31. Curves showing approximate effect of carbon content and heat treatment on tensile strength of steel wire.

nealing, but others have shown that the same tendency persists in 0.20 to 0.30 per cent carbon steel.

Time and Temperature for Annealing—From what has already been said, and the evidence supplied by Figure 40—30, it is evident that, for the best results in annealing, careful regulation and adjustment of the time and temperature are necessary. In all types of annealing, sufficient time must be allowed for the whole charge to reach the required temperature, and in low-temperature annealing additional time is required for the development of the structure desired. Process annealing requires careful adjustment of time and temperature in order to avoid the formation of large grains, and, at the same time, eliminate the effects of the cold working, because at a low temperature the atoms or molecules move at a slower rate, and more time is required for their adjustment than at higher temperatures. As to temperature, the fact that full control and careful regulation of this factor in all annealing is necessary to satisfactory results cannot be too highly emphasized. If the temperature is too low, it may fail entirely to accomplish the results desired in process annealing. In some cases adjustment of the temperature to the size of the wire is also necessary. Fine wire in coils, for example, is generally annealed at as low a temperature as practicable, in order to avoid the sticking of the wire in the coils. After these remarks it is scarcely necessary to say that accurate and carefully standardized pyrometers form an essential part of any equipment for the proper annealing of wire.

The tensile strength (at various carbon levels) of wire full-annealed or recrystallized by annealing above the upper critical point, followed by slow cooling, is shown in Figure 40—31, as well as that of wire subjected to annealing in a molten lead bath (one form of a "sub-critical anneal" involving a short time cycle of heating and cooling). It should be noted that it is possible for sub-critical annealing to produce approximately the same tensile strength values obtained by full annealing, providing the long time cycle of heating and cooling used in the former treatment is observed. A comparison of the effects of full annealing on recrystallization and grain size with short time sub-critical annealing of hard-drawn wire is indicated by the photomicrographs of Figures 40—26 and 40—27. These differences in structure are reflected in the lower tensile properties of full-annealed wire shown in Figure 40—31.

METHODS OF ANNEALING WIRE

While it is possible, theoretically at least, to anneal wire in any apparatus in which the proper conditions of time and temperature can be maintained, the efficient handling of the material calls for, and some of the finishes and conditions to be met demand, equipment specially designed for the purpose in mind. These conditions, in general, call for differences in the time of annealing, and differences in the methods of applying the heat, and usually require that the material be protected against oxidation. To meet these conditions, four different methods requiring entirely different apparatus and equipment are used. These are known as **muffle annealing, pot annealing, continuous lead annealing and tube annealing.**

Muffle Annealing represents the crudest form of annealing used in the wire industry and must not be confused with heat treatments carried out in modern muffle furnaces of the type described in Chapter 21. The common muffle furnace consists of a brick-lined chamber about 12 feet long, 8 to 12 feet wide and 2 to 2½ feet high. At the back of the chamber and separated from the muffle proper by a baffle wall is a grate for

burning the fuel which heats the furnace. At the opposite end of the muffle chamber is a hanging door spanning the width of the furnace, through which, when open, the wire is charged and discharged. The bottom of the chamber is provided with a grating of iron rails or castings known as "muffle ladders," the object of which is to form a support for the wire so that the furnace gases will be free to circulate beneath it. The wire is seriously oxidized on its surface by being in contact with the open flame, which passes over the fire wall into the muffle proper and then out of the flues to the stack. For this reason, muffle annealing is generally employed only to meet unusual conditions.

Salt-Bath Annealing—Molten salt-bath annealing is sometimes practiced for process annealing of common sizes of wire. The wire, in coils, is immersed for one-half hour to one hour in gas-fired pots containing molten salt which is held at some predetermined temperature between 1000° and 1300° F, depending on steel composition and structure or physical properties desired. Advantages over other methods are that small amounts of wire may be quickly annealed at closely controlled temperatures without scaling the surface on the wire. Such annealing is somewhat limited as to its application in wire processing.

Pot Annealing—This method of annealing is identical in principle with that known as box annealing, the material being placed in a sealed container, or pot, and thus protected from the air and furnace gases during the entire process of annealing. The method is largely used for annealing steel process wire, but a considerable amount of copper wire is also pot annealed.

Equipment for Pot Annealing—The older pot-annealing equipment consists of pit furnaces, annealing pots and power cranes. The pit furnace is constructed of fire-brick and is direct fired with coal burned on a grate at the bottom of the pit below the pots, and the flame is circulated up and around the pots. The pots are cylindrical and are made of cast iron or cast steel. The wire coils are loaded within the pots, covers are sealed thereon, and the pots are lowered into place in the pit furnace, which is then closed with a fire-brick cover. As the coal-fired furnaces were not always adapted to provide uniformity of temperature throughout the pot, they have been largely superseded by the bell type of annealing furnace, either electrically heated or gas fired. In the bell type of furnace, the wire coils are placed one on top of the other on stems, forming a cylindrical stack or pile on a refractory-brick or cast alloy-iron base. A cylindrical cover or hood of thin sheet steel is placed over the stem of wire and a second cover or hood, somewhat larger in diameter, is then set in place. This outer cover may carry the heating equipment, either the electric resistance heating elements or radiant-tube gas-fired heaters; in some of the latter types, the gas-fired flame may circulate in the space between the inner and outer covers or hoods. Thermocouples are placed within the inner hood to indicate, record and control temperatures of the top and bottom of the stem of wire. To aid in attaining the desired temperature uniformity throughout the mass of wire, fans are placed in the base of the furnace to circulate the heated non-oxidizing or reducing gas atmosphere which is maintained within the inner cover throughout the heating and cooling cycle. The covers containing the heat sources are interchangeable and are moved from one base to another as required. This allows one charge which has been annealed to cool slowly under cover of the inner hood only, while the outer hood is placed over a new charge to begin its heating cycle.

Advantages of Pot Annealing—The chief advantages

of pot annealing are the following: The furnace gases do not contact the wire being treated and, if desired, air may be excluded from the inner covers to such an extent that a bright clean surface may be obtained on the annealed wire. Also, as compared with muffle annealing, pot annealing makes possible a greater uniformity of temperature throughout a load of wire coils and a more exact knowledge of the actual temperature attained by different parts of the load, providing thermocouples are used to best advantage. Another advantage is that when the wire is cooled in the pot, the rate of cooling may be comparatively slow, resulting in a softer product; this is especially advantageous in the annealing of the high-carbon steels. Most of the heavy work of handling coils in pot-annealing departments may be done with cranes. Also in pot annealing, there is little snarling of wire. With proper appliances, bright annealing is possible, and in general the loss due to scaling is comparatively slight in any pot-annealing apparatus, because there is little opportunity for scale formation. The lighter coating of scale also means that less acid is needed for cleaning. Furthermore, various finishes may be produced on steel in pot annealing, the results obtained depending largely on how the wire is prepared and how the pot is sealed and cooled. If the air is allowed to come in contact with the wire when it is at high temperature, oxidation will occur, of course. But by controlling the air admitted to the pots, and by retarding the cooling, it is feasible to obtain wire that is black, blue, or bright, while liquor-finished wire may be annealed "white" by entirely excluding air.

Continuous Lead Annealing consists merely in drawing the wire through a bath of molten lead heated to the proper temperature. The molten lead is contained in a shallow rectangular steel pan, some 10 to 15 inches deep, 3 to 4 feet wide, and 15 to 25 feet long, the exact dimensions depending upon conditions. (Sometimes two pans are used, the first known as the cold pan and the second as the hot pan, and the wire is drawn through each in succession.) The pan is heated from below usually by gases from an oil or gas fire and is supported by a brick setting adapted to the method of heating. In practice several strands of wire are drawn through the bath in parallel. For this purpose the coils of wire to be annealed are placed on free-running reels before the annealing furnace, and the wire from each coil is drawn through the bath by a take-up block placed at a convenient distance from the other end. For keeping the wires immersed in the molten lead, devices of various forms, known as sinkers, are used. To obtain bright annealing by this process, it is necessary to cool the wires without their being in contact with the air. Consequently, when such a finish is desired, the wires from the hot lead bath are conducted directly into long tubes which are kept as nearly sealed as possible.

The Advantages of Lead Annealing are obvious. In addition to protecting the material from the air, the bath of molten metal is readily maintained at a uniform temperature throughout its mass, and its temperature can be accurately ascertained at any time by a pyrometer. Also, since the process is continuous, it eliminates much handling of the material.

The Principal Use of Lead Annealing is in connection with galvanizing plants, where it is employed to anneal process wire. In these plants, layouts are provided that permit the wire to be annealed, cooled, cleaned, washed and dried, or dried and galvanized or tinned, all in one continuous operation. In continuous annealing and cleaning, when the wire is not to be galvanized or tinned, the wire, after passing through the lead bath, is conducted through crushed coal or sand banked at the end

of the lead pan; this bank of coal or sand is known as a header. The wire then passes successively through a water bath, a weak acid bath, a cold water bath, and a hot water bath, thence to the take-up blocks, which are located several yards beyond the hot water bath to permit the wire to dry. This elimination of the customary process of dip-cleaning with cranes is very desirable in the case of process wire.

PATENTING

Patenting is a heat treatment applied to rods and wire having a carbon content of 0.25 per cent and higher, the term being peculiar to the wire industry. The object of patenting is to obtain a structure which combines high tensile strength with high ductility and thus impart to the wire the ability to withstand hard drafting to produce the desired finished sizes possessing a combination of high tensile strength and good toughness.

Methods of Patenting—Patenting is always conducted as a continuous process and consists in first heating the material to a point well above the upper critical temperature, then cooling through the critical temperature at a comparatively rapid rate to a predetermined temperature level at which the transformation will yield the desired microstructure and mechanical properties. In practice, there are three ways of carrying out the treatment. Thus, the wire may be heated by passing it through alloy-steel tubes arranged in an open muffle or in an open flame without tubes and be cooled by pulling it from the furnace into the open air, which method is now referred to as "O.P." (old process or air) patenting. In a second method, the wire may be heated as in the first method, then cooled by passing it into a lead bath held at a relatively low temperature; this process is known as the **metallic hardening process**. The wire, in the third process, can be heated in a bath of very hot lead and cooled in another bath of lead at a lower temperature, as in the **double lead process**. In this last process the temperatures of both baths can be readily controlled and accurately measured. These features make it possible to obtain any desired structure even in wires of highest carbon content, a condition not easily attainable in "O.P." patenting. The third method is also more advantageous in that less scale is formed than in the other two methods. In the wire industry, both the metallic-hardening process and the double-lead process are generally referred to as "lead patenting."

Properties of Patented Wires—The structure obtained by patenting is extremely tough and possesses the best characteristics for drawing to high tensile strengths. As indicated in Figure 40-31, the tensile strengths of air- or lead-patented rods or wire are considerably higher than the same steels in the annealed condition. The lead patenting process is definitely required in the production of any exceedingly high strength and tough wire such as music wire. As an illustration, by properly patenting and drawing 0.75-carbon steel, a wire is produced having a tensile strength of 375,000 lb. per sq. in. or over. In spite of the great amount of drafting required to raise the tensile strength to this point, such wire will be tough enough to wrap around itself (i.e., can be wound around a pin of the same diameter as the wire) or can be hammered flat to one-half its original thickness without cracking.

HARDENING AND TEMPERING

In other branches of the steel industry, hardening and tempering are usually considered as separate operations, but in wire-making they are more often conducted in a single continuous operation. As in the case of tools and other products of steel, the object in hardening and

tempering wire is to control the hardness, or "temper," in order to adapt the material to the use for which it is intended. Hardening and tempering are applied to a great variety of wires, such as typewriter wire, umbrella wire, dress-stay wire, curtain spring wire, and wires for measuring tapes, clock and watch springs, automobile-engine valve springs, door-check springs, etc. Wires for such purposes are generally treated in wire form by a continuous process, but many kinds of springs, tools, and miscellaneous special products, are heat-treated after the wire has been formed into the article desired, when it is not always possible to use a continuous hardening and tempering process.

Methods of Hardening and Tempering Wire—For greatest efficiency in hardening and tempering, it is necessary to adapt the method of treatment to the nature of the material and the size and form of the article. Therefore, because of the great variety of steels and of articles produced by the wire industry, various methods of treatment involving the use of various kinds of apparatus and of various heating and cooling media, are employed. Thus, for hardening, the material may be heated in muffle, tube or electric furnaces, or in molten lead, and be quenched in water, oil, molten lead, or some other liquid. For tempering, the material may

be reheated in open furnaces, in baths of molten lead or hot oil, or by bringing it in contact with hot sand or with hot plates or discs of iron or steel. In continuous hardening and tempering, the heating for hardening may be accomplished in a manner similar to that for patenting, and the quenching may be accomplished by passing it quickly into a tank of water or oil; in the tempering operation, the material may then be drawn through a suitable bath, such as molten lead, the wire being in continuous motion throughout the process.

AUSTEMPERING

Austempering, described in Chapter 42 on "Heat Treatment," is a method adapted to the heat treatment of wire, and particularly formed parts. It is employed to develop the highest combination of strength and toughness in certain kinds of wire, particularly for fabrication into springs. In this process the transformation of austenite is effected at some preselected and constant subcritical temperature, which usually requires a longer time than does the regular continuous oil tempering operation; consequently this process is preferably used for batch or semi-continuous operation on high-carbon springs or small articles made of steel of $\frac{1}{4}$ -inch maximum thickness or diameter.

SECTION 9

PROTECTIVE METALLIC COATINGS

Kinds of Coatings—Probably more than one-third of all the steel wire drawn is given some kind of metallic coating, either for decorative or protective purposes. This phase of the industry has reached huge proportions and is of great importance. As already indicated, appreciably large quantities of the medium and larger sizes of wire are given a light coating or "liquor finish," varying in color from that of copper through the various shades of brass to the color of tin, for decorative purposes only, and practically all soft fine wires have to be given this coating for lubricating purposes. This coating, which may consist of copper or tin or a combination of the two, is very thin and has but little value as a protection against corrosion. As is well known, all of the common grades of steel will rust sooner or later when exposed to atmospheric conditions unless their surface is covered with some substance that will protect it from moisture and air. To afford such protection, it is the practice to galvanize wire, i.e., coat it with zinc, or to tin it, or to coat it with aluminum, or, for certain purposes, to give it a coating of paint, lacquer or japan. While it is also possible to coat steel wire with copper, lead, nickel, and the precious metals, these metals are seldom used for coating wire, except in a few particular cases, so they need not receive any further consideration here. The remainder of this section will, therefore, be devoted to galvanizing, tinning, and aluminum coating.

WIRE GALVANIZING

Advantages of Galvanizing—As a protective coating, zinc offers several advantages over other metals: (a) Zinc is electropositive to iron, which means that if the steel of a galvanized wire should be exposed to corrosive influences, owing to defective or damaged coating, the zinc will corrode first, and its presence will protect the iron from corrosion until after a considerable area has been exposed. Just the opposite reactions might occur with other metals, such as tin, which are electronegative to iron. The presence of tin alone under the

same conditions would hasten the corrosion of the iron, but, for specific purposes, tin is added in small amounts to zinc for galvanizing to improve adherence of the metallic coating, and the size and appearance of the spangle, without affecting corrosion resistance. (b) Zinc may be obtained in a comparatively pure state at a relatively low cost. (c) With zinc, it is easy to obtain a hard, smooth coating comparatively resistant to abrasion. (d) The color of the zinc coating is satisfactory for general purposes.

Methods of Galvanizing—There are two common methods in use for galvanizing wire, known as (1) the hot galvanizing process and (2) the electro-galvanizing process, but the second is the less widely used. In both processes, some 30 (more or less) parallel strands of wire pass first through certain preparatory processes, thence through the galvanizing bath to take-up frames on which each strand of wire is separately coiled, all these operations being made by attaching the tail end of each coil to the head end of the succeeding one. As the wires pass through the various baths required to clean and coat them, they are submerged by suitable forms of sinkers (either of the stationary or rotary type) or, in the case of electro-galvanizing, under contact fingers in the electrolytic or plating solution.

Processes Preliminary to Hot Galvanizing—In nearly all cases, wire to be galvanized must be annealed to remove the effects of cold working, and, in order to minimize handling of the wire, it is common practice to do this work in conjunction with the galvanizing operations by one of the continuous annealing processes. For this purpose a hot lead furnace will be built in front of the cleaning and galvanizing apparatus. Since the rate of cooling in process annealing has little effect upon the physical properties of the wire, the wires are cooled in air, or, if the space is limited, low-carbon wires will be cooled by conducting them from the annealing furnace into a vat of water. Following annealing, the next step is cleaning, for which purpose the wire is conducted into a bath of hot muriatic (hydro-



FIG. 40—32. Overall view of the apparatus employed in the continuous hot-dip galvanizing of steel wire.

chloric) acid at predetermined concentration. This acid is used instead of sulphuric, which is the acid commonly employed for cleaning in the galvanizing of sheets and tubes, because it acts much more quickly than sulphuric, and is much more effective in removing traces of lime remaining from the drawing operations. The iron chloride formed by the action of the acid, as well as any particles of loosely adhering scale, must next be removed and these objectives are accomplished by passing the wire through a bath of hot water.

While the wire is now perfectly clean, it must also be perfectly dry before it is brought into contact with the molten zinc, but in drying, any exposure to the air, which in practice it is impossible to avoid, results in the formation of a light coat of oxide, or rust. This difficulty is overcome by the use of a flux. In galvanizing sheets, tubes, and various wire products, this flux consists of fused ammonium chloride, commonly known as sal ammoniac, which lies upon the surface of the zinc bath, but for the continuous galvanizing of wire, better results are obtained by passing the cleaned and thoroughly washed wires directly through a hot solution of zinc chloride or zinc ammonium chloride flux at predetermined concentration or Baumé; and then through or over a dryer into the molten zinc. The wire thus becomes coated with a thin film of zinc chloride, which tends to protect it from oxidation during drying and also removes any traces of rust that may be formed.

Apparatus for Hot Galvanizing—From the dryer, the wires are drawn at once into the molten zinc, or spelter, as it is commonly called. This molten metal is contained in a spelter pan, which is usually made of boiler plate and is supported by a brick setting of suitable construction for firing with the most satisfactory fuel available. Figure 40—32 shows the general form of the pan and the furnace commonly used. The dimensions of these pans are subject to much variation, and depend upon several conditions. Thus, the width will vary according to the number of strands it is desired to galvanize at once, which is usually about 30. The length

is dependent upon the size of the wires, the speed of travel, and the thickness of the coating desired. Pans designed for galvanizing coarse sizes of wire may reach a length of thirty feet. The depth of the pan must be sufficient to prevent the wires from coming into contact with the dross which settles and collects upon the bottom. This dross is an alloy of iron and zinc containing from 3 to 7 per cent iron, which is solid at the temperature of molten spelter, forms a pastelike mixture, and is very harmful to the coating. As molten zinc oxidizes rapidly, the pan is provided with some form of covering medium which rests upon the molten spelter and protects it from the air, except at the ends where the wires enter and leave the bath. Here, the surface of the metal must be kept free from oxide by frequent skimming. The mixture of zinc and zinc oxide thus obtained is known as zinc skimmings; these skimmings, together with the dross, represent a considerable loss, for the proportion of these waste products to the total zinc used is relatively large, and, although both are refined, the cost of recovering the zinc they contain is relatively high.

Wiping the Wire—As the wires emerge from the galvanizing pan, some of the zinc they carry remains in the molten state for a brief period, and, unless prevented from doing so, tends to flow downward on the surface of each wire, making the coating rough and uneven. Such a coating makes the wire hard to handle, and renders it unsuitable for many purposes, especially when it is to be fabricated by machinery. Further, such a coating represents an actual waste of zinc, for the durability of an unevenly applied coating is determined by the thinnest part of the coating, not by the thickest parts. It is plain, therefore, that the removal of this excess zinc is a matter not only of economy, but of expediency as well. It is also evident that in order to form a smooth, evenly distributed coating, the surplus zinc must be removed while it is still in a fluid state. This is accomplished by passing the wire, just after it emerges from the zinc bath, through either one of two

devices known as **wipes and headers**. In the case of wipes, the parts in contact with the wire are made of asbestos, and the type most commonly used is known as the **split wipe**. It consists essentially of two balls, or molds, of asbestos fibers held together by some suitable binding material, or stranded asbestos rope of suitable diameter formed into wiping pads. In service, these two balls are pressed lightly against and about the wire, and are held securely in place by cup-shaped holders. The aim in the split wipe is to make it possible for the operators to separate the two halves in order to permit the splices joining two coils of wire to pass, as otherwise the wipe is torn out entirely or badly damaged. Headers are employed for wiping only when extra heavy coatings are desired. A header consists of crushed charcoal of definite mesh, blended with a certain wax, beef tallow or certain oils and arranged so that the wires shall pass upward through it. In any of these cases the wipe thus artificially provided acts as such only for a short time, for a hollow cone of solidified spelter soon forms about the wire, either within or in front of the wipe, and this cone of zinc then becomes the real wipe.

Cooling the Coated Wire—After passing the wipes, the coating of the wire is completed, but the manner in which it is cooled should also be given some consideration. If the wire is allowed to cool naturally in the air, the coating, because of the formation of a thin film of zinc oxide on the surface, will be dull and lusterless in appearance, known as air-cooled galvanized wire, but if the coating is cooled suddenly, as by immersing the wire in cold water as soon as possible after leaving the wipes, it will have a bright lustrous surface. However, since the real value of the coating as a protection against corrosion depends upon the thickness of the coat and the completeness with which it excludes moisture from the iron, it is clear that the luster, which is a characteristic of suddenly cooled coatings, is no indication of the quality of the coating, and since this luster can be preserved only for a short time in any natural atmosphere, its attractive appearance is of little value.

Coiling the Wire—The wire, which is delivered to the galvanizing department in coils, is drawn through the annealing, cleaning and galvanizing apparatus by blocks, which form it into coiled bundles again (Figure 40—32). The blocks commonly used for this purpose are of the so-called continuous type, and are similar to a wire-drawing block except that their spindles are mounted in a horizontal position. With these blocks, bundles of wire may be removed at any time without stopping the blocks, so that the speed of travel of the wire through the annealing and galvanizing baths may be uniformly maintained.

Some Features of the Operations for Hot Galvanizing—To the novice the galvanizing of wire may appear to be a simple process. The wires being drawn through the various pieces of equipment and recoiled on the blocks apparently require little attention from the operator. As a matter of fact, however, the process demands considerable experience and skill in order to obtain uniformly good results. For example, the firing of such long shallow pans requires not only a properly designed furnace, but constant care and observation on the part of the operator to keep the temperature of the spelter uniform and constant and to avoid injury to the pan. Similar statements apply also to the annealing and cleaning equipment. Some of the factors that determine the quality of the zinc coat are as follows:

The Quality of the Zinc Coat, by which is meant its effectiveness in protecting the wire from corrosion and its adaptability to the intended use of the wire, de-

pends upon the completeness with which the zinc covers the iron, the thickness, uniformity, physical properties, structure, and chemical composition of the coat. The completeness with which the iron is covered by the zinc depends upon the thoroughness of cleaning and fluxing and also upon the aftertreatment of the wire. Attention has already been called to the effects of the manner of cooling the wire as it is leaving the spelter bath. It should also be noted that the coating may be injured in fabricating the wire. If the cooled wire is bent at sharp angles or otherwise severely deformed, the coating is likely to be cracked or peeled off, especially if the coating is a heavy one. As to the thickness of the coat and the uniformity with which the spelter is distributed over any given wire and adherence of the zinc coating, these features depend upon the type of steel used, thoroughness of cleaning, the methods of wiping, the temperature of the molten spelter, and the time of immersion of the wire in the spelter bath. Up to certain limits, the thickness of the spelter coat increases as the temperature of the bath increases and as the time of immersion is prolonged. The time of immersion is controlled by the length of the spelter pan and the speed of travel of the wire through the bath. In this connection it should be pointed out that the thickest coat is not always the best, for, as the thickness of the coat increases, its brittleness and its tendency to peel or crack also increase. Some of the brittleness is to be attributed to the fact that zinc itself is a comparatively brittle metal. This brittleness might be lessened somewhat by alloying with other metals, but as these might decrease the resistance of the zinc to corrosion or otherwise affect it adversely, it is the practice to keep the spelter as pure as possible. The composition of the spelter is checked repeatedly by chemical analysis, and close limits are fixed for all the impurities it may contain. Another reason why the brittleness increases with the thickness of the coat is the fact that the conditions necessary to produce a heavy coat are also likely to affect the structure of the coat itself, for the coat obtained by hot galvanizing is not wholly composed of pure zinc.

The Structure of the Zinc Coat—The structure of the zinc coating as obtained on hot-dipped galvanized wire is shown in the photomicrograph of Figure 40—33. It will be seen that the spelter coating is made up of three different layers. The lower portion of the photomicrograph represents a section of the steel base, and the zinc layer represents the outer layer of spelter, which may be considered pure zinc. The alloy layer is made up of two different alloys of iron and zinc, the one next the steel base containing slightly more iron in solution than the other. Formerly it was thought that when iron is brought in contact with molten spelter, two compounds, having the formulas FeZn_2 and FeZn , are always formed. More recent studies indicate that the constitution of these coating layers is not precisely represented by these formulas. Assuming for the present that the formulas are substantially correct, however, the alloy next to the steel base consists of the compound FeZn_2 , while the remainder of the alloy layer is composed of the compound FeZn . The dark lines separating different areas in the photomicrographs are not cracks, but the result of grain boundary effect in etching, similar to that observed in developing the structure of pure metals.

Electro galvanizing—Electro galvanizing is the name applied to the process of covering any metal with a coating of zinc by means of an electric current. It is sometimes termed "cold" or "wet" galvanizing to distinguish it from the more common method of hot

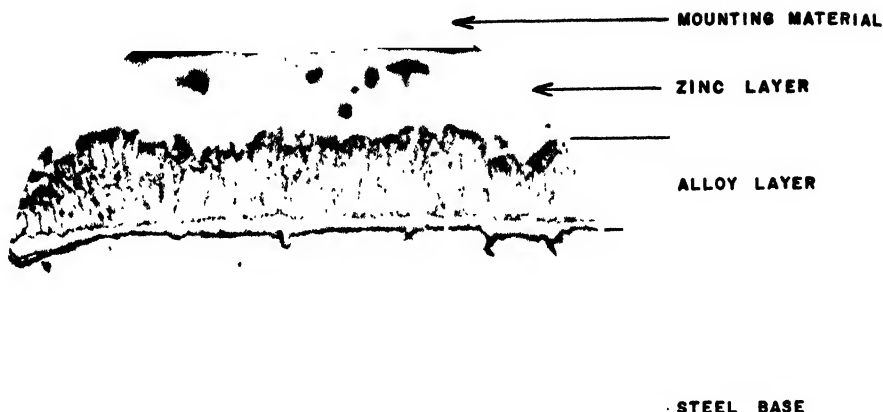


FIG. 40-33. Structure of the zinc coating on a steel wire galvanized by immersion in molten zinc. Magnification: 250X.

galvanizing. For galvanizing flat wire and also strip steel, this cold process has certain advantages over hot galvanizing. For example, in hot galvanizing such materials, the edges, especially of strip steel, tend to destroy wipers, and special care is required to secure a smooth uniform coating, whereas the thickness of the zinc coating may be readily controlled in electrogalvanizing. The thickness of an electrodeposited coating depends primarily on two factors, namely, current density and time. By current density is meant the number of amperes flowing per unit of surface of the metal exposed in the electrolyte. The coating, when well applied, adheres firmly to the metal and will stand severe bending without flaking off. But many features of the process require special attention, because the quality of the coat depends upon the condition of the electrolyte, the cleanliness of the surface, and on general working conditions.

The Equipment for Electrogalvanizing consists of a long shallow horizontal vat, known as the plating vat, which is usually made of steel lined with rubber or a synthetic mixture. Attached to one outer side of this vat are the positive bus bars, or feeders, connected to the plating generator, while on the other side are similar negative feeders. As considerable time is required for suitable electroplating, the vats are usually from 100 to 200 feet long in order that the speed of the wires or strips through the vats can be maintained at a reasonable rate for production and still remain submerged in the electrolyte long enough to receive the desired coating. Since the working voltage is low and the amperage high, the conductors consist of heavy copper bars. In addition, there are needed a low-voltage direct-current supply source, and means for making and storing fresh electrolyte.

Operation of the Process—The vat is nearly filled with the plating liquid, called the electrolyte, consisting of a solution of some zinc salt, such as zinc sulphate. This solution must be continually agitated to maintain a uniform density, and it must also be kept clean and at a fairly constant and suitable temperature. For anodes, slabs of zinc are submerged at selected locations in the electrolyte and are electrically attached to the positive bus bars. Alternating with the anodes are strong copper contact bars which are placed across the vat, immersed a few inches below the surface of the electrolyte, and con-

nected with the negative bus bars outside the vat. These contact bars support the wires being plated, which collectively form the cathode. The wires to be electrogalvanized are generally annealed, thoroughly cleaned in acid, then rinsed before they pass through the plating vat. From the vat the wires pass through or over some simple form of wipe and onto take-up blocks placed far enough away from the wipe to permit the wires to dry in air.

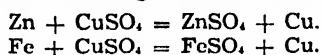
Factors in Controlling the Thickness of the Coat—Theoretically, about 2.5 pounds of zinc can be deposited per hour per 1000 amperes of current. In practice, the amount of zinc deposited will be somewhat less, as no electrochemical operation can be carried on at 100 per cent efficiency. The practical maximum current density varies and must be determined experimentally for each set of working conditions. Having determined this and also the efficiency to be expected, it will be possible to calculate the required total current needed to deposit a required amount of zinc in a given time. The current flowing from the zinc anodes through the electrolyte into the cathode wires causes the zinc to dissolve into the solution at the same rate that the zinc is deposited on the wires, so that the solution, if it is continually and correctly agitated, will remain at a uniform density, and only pure zinc will be deposited, although the slabs used usually contain some impurities. As the zinc ions traveling through the electrolyte actually carry the current, the amount deposited will be directly proportional to the current strength and to the time of current flow. Since the former factor is fixed, the latter must be increased as the required weight of coating becomes greater.

Tests for Galvanized Coatings—The whole galvanizing process is placed in the hands of experienced operators, who keep constant watch over each phase of the process and at frequent intervals make certain tests to determine the fitness of the material for the purpose for which it is to be used. With respect to the galvanized coatings, these tests are both physical and chemical. The more common of the physical tests are known as the button test and the mandrel test, each being applied to different classes of wire to meet different requirements. In the button test, the wire is wound tightly about its own diameter, while in the mandrel test the wire is similarly wound about a smooth mandrel having a diameter that

is some multiple of its own diameter. The object in both these tests is to determine if the coating will crack or flake under these conditions and, if so, to what extent. For determining the thickness of the coat and whether or not the zinc has been uniformly distributed, two chemical tests are employed. One of these, called the **antimony hydrochloride test**, is a laboratory test used chiefly for determining the exact quantity of zinc per unit of surface galvanized. The other, known as the **Preece or copper sulphate test**, though only qualitative, is the best known and most generally used chemical test for galvanized coatings. This test, when made under properly standardized conditions, will determine whether or not the coating is uniform in thickness and whether the thinnest portion will dissolve in a certain solution of copper sulphate in one, two, three, four or more minutes. The terms "one minute wire," "two minute wire," etc., are based on this test, and mean that the coating will successfully withstand immersion in a standard neutral copper sulphate solution for one, two, or more minutes.

Methods of Carrying Out the Copper Sulphate Test—The standard solution used in the test is prepared by dissolving approximately 36 parts of crystallized commercial copper sulphate in 100 parts of distilled water, adding an excess of pure cupric oxide (CuO), and filtering. The concentration of the solution is then adjusted, by adding water or a stronger solution, so that its specific gravity is exactly 1.186 at 65°F . The samples to be tested are thoroughly cleaned before making the test by dipping in benzene or gasoline, or rinsing in water, and wiping with clean white cotton waste or cloth. A clean test jar, 5 inches high and 2 inches in diameter, is then filled to within one inch of the top with the standard solution at a temperature between 62°F and 68°F , which temperature must be maintained throughout the test. The sample is now immersed in the solution by lowering it endwise into the jar, where it is allowed to remain for exactly one minute; it is then removed, rinsed immediately in water of the same temperature, and wiped until dry with the clean cotton waste or cloth, so as to remove the dark deposit of copper which forms on the surface. These operations are then repeated, the wire being immersed for exactly one minute at each trial, until traces of bright metallic copper begin to appear on the surface of the steel, indicating removal of the zinc, or until the number of one-minute immersions exceeds the number required by the specifications. In case more than one wire is tested at the same time, the samples, numbering not more than seven, are immersed simultaneously in the same jar, and these must be kept well separated. The lower inch of each sample, being adjacent to the cut exposing the iron, is disregarded in the test. A fresh portion of the standard solution must be used after each complete test of seven wires or less.

Principles of the Test—The principles upon which the copper sulphate test depend may be explained as follows: The copper sulphate dissolves both zinc and iron, these two elements displacing the copper in the solution of copper sulphate according to the following reactions:



From these reactions it will be seen that copper is precipitated, or thrown out of solution, in each case. Since the reaction can take place only where the wire is in contact with the solution, this liberated copper is deposited upon the surface of the wire, and the test depends wholly upon the difference in the manner in which this deposit is made upon zinc and upon iron. In the case of zinc, the deposited copper is in a finely

divided form, and appears as a black flocculent coating completely covering the wire. As this coating increases in thickness, it tends to shield the wire from the solution, thus retarding the reaction, hence the necessity for removing the wire from the test jar and cleaning its surface at regular intervals of one minute. In the case of iron, however, the copper, if all the conditions of the test are properly observed, is deposited upon the surface in the form of a bright metallic coating which cannot be wiped off and is similar in appearance to that obtained in electroplating. Thus, as soon as the zinc has been removed from any part of the surface of the wire, this bright coating appears on that spot.

Value of the Copper Sulphate Test—The chief merit of the copper sulphate test is that it reveals any unevenness in the thickness of the coat. Although the test is not directly quantitative, it provides a relative measure of the amount of zinc at the thinnest point of the coating, provided the surface condition, composition and structure of the coatings being compared are the same. This information is important, for the corrosion resistance of the coat depends not only upon the weight of the zinc on a given surface but also upon the evenness with which it is distributed. In service, corrosion of the steel begins where the zinc coating first disappears, so that for a given total weight of zinc applied, the best protection to the steel is offered by the most uniform distribution of zinc. The test can be quickly made with simple apparatus, and the effect is very apparent, but each phase of the test must be carefully performed under standard conditions to obtain fair comparative results. The latter fact is not generally recognized, the impression being that almost anyone can make the test without instruction as to the points to be observed. In this connection it should be noted that the copper will plate out upon an alloy of zinc and iron as well as upon pure iron or steel, but the coat of copper on the former does not adhere firmly to the surface and may be brushed off with a fine steel brush.

WIRE TINNING

Because of the high cost of tin, and of the difficulty of entirely avoiding pin holes in tinning steel wire, only a small amount of steel wire is tinned, and that largely for decorative purposes. As already explained, tin is cathodic to iron. Consequently, in the case of steel wire coated with tin and exposed to the atmosphere, the destruction of the steel base will be accelerated by the presence of tin when corrosion begins. The tinning process is very similar in principle to the hot galvanizing process and need not be described at length. The wire has to be thoroughly cleaned and fluxed before entering the bath of tin, and as it emerges from this bath it passes through a wipe of wicking wound about the wire, after which it goes to the take-up blocks, either with or without an intermediate water cooling. The tin must be maintained at a fairly constant temperature (500° to 550°F), for if the bath temperature is too low, the coating will be rough and uneven, and if too high, it will be discolored by the yellowish tin oxide formed on the surface.

ALUMINUM COATINGS

In recent years aluminum coatings have been applied to steel products for resistance to corrosion in marine, industrial and the usual environments. The aluminum is applied to steel wires by the hot-dip process, where the cold-drawn wires are cleaned prior to immersion into a molten aluminum bath. Aluminum forms alloys with steel in a manner similar to zinc, and the process must be carefully controlled; otherwise, a brittle alloy layer

will form which will cause cracking and spalling during subsequent forming operations.

The aluminum bath is operated at a temperature of approximately 1225 to 1250° F. This temperature is sufficient to soften carbon steels with a corresponding loss in tensile strength. Aluminum-coated wire can be cold drawn after coating to improve or raise the tensile

strength. Also, by drawing after coating, very bright finishes can be produced which make the wire suitable for many decorative applications.

Wire which can be coated with aluminum for corrosion protection includes: field fence, barbed wire, broom wire, netting wire, paper-clip wire, spiral-binding wire, tie wire and welding wire.

SECTION 10

TYPICAL FINISHED WIRES FOR MANUFACTURING PURPOSES

The object of this section is to list according to their common trade names the different kinds and grades of wire used for manufacturing purposes and indicate the uses of each.

COMMON WIRES

Bright Bessemer Wire—Wire of this grade is hard or stiff wire, ordinarily drawn from a rod to finished size without an intermediate annealing. It is not well suited for purposes involving special finishes or specific mechanical properties.

Bright Basic Wire or Bright Hard Basic Wire—Wire of this classification receives the same treatment as Bright Bessemer, is slightly softer wire than Bright Bessemer Wire, has a finish obtained by regular dry drawing and is used in applications not requiring specific physical properties.

Bright Soft Basic Wire is a soft grade of bright wire for manufacturing purposes, and is adapted to difficult forming, severe twisting, swaging, crimping, etc. Wire of this classification is also suitable for severe upsetting.

Medium Classifications—Medium hard and medium soft Bessemer and basic wires are finer gradations between the "bright" and "bright soft" grades. They are common only to the finer sizes, 8¼ gage (0.152-inch) and finer, and their use depends upon the severity of the bending and forming operations which are to be performed.

Annealed Wires—Annealed wire is the softest grade that can be produced. Because of differences in chemical composition (assuming the carbon content is the same), annealed basic wire is softer than annealed Bessemer wire of the same size. In producing annealed wire, the coarser sizes must be given a special draft before annealing, otherwise brittle wire known as "crystallized wire" will be obtained after the annealing. The finer sizes are drawn the same as the bright (hard) grade and then annealed. "Annealed," or "annealed, cleaned and limed basic" Cold-Heading Quality wire is used largely for extremely severe upsetting, such as special bolts, certain kinds of rivets and for any purpose in which the wire is severely deformed by pressing or heading operations. If maximum softness is essential, the use of annealed basic wire is indicated.

Cold-Heading Wire—Wire for cold heading or cold forging has become one of the large production items in the wire industry. Wire of this grade is used for bolts, rivets, screws, and similar applications requiring, in many instances, severe plastic deformation. For common fabrication, low-carbon steel may be used, but with the advent of highly stressed parts as in automobile and aviation applications, higher carbon, low-alloy and stainless steels are now required. As upsetting the head on the bolt or screw will readily accentuate surface defects of the wire such as seams or slivers, bolt wire must necessarily be a high quality product with great uniformity of structure and composition. Production of higher carbon and alloy wire for heat-treated bolts requires close control at all stages: melting

the steel, rolling the rod, heat treating and drawing the wire. In addition to the above prerequisites, the surface coating of the wire, which is controlled primarily by the wire-drawing lubricant, is important as a factor in the cold-heading operation. The type of upsetting die and whether or not the bolt shank is extruded in the operation determine the type of wire coating required; the coating, in turn, has a significant bearing on the life of the heading die and the quality of the finished bolt. The finished hardness and chemical composition of bolt wire both are definitely controlled according to the severity of the deformation required in forming the bolt. As the hardness increases with increasing carbon and/or alloy content of the steel, the final hardness will be regulated by normalizing, process annealing, or spheroidizing the steel. Machinability requirements and the ability of the wire to respond to heat treatment may also be factors influencing selection of the manufacturing processes to produce wire having the proper microstructure and hardness, particularly in steels over 0.35 carbon. The finished product is rigidly inspected, and includes adequate mechanical, chemical and metallographic testing.

Liquor-Finished Fine and Weaving Wire—Some of this grade of wire, which may be used for weaving household window-screen cloth, is made in very fine sizes, 0.0118 inches and 0.0104 inches being the common sizes. Wire for this purpose is usually made from low-carbon basic steel and to reduce the wire from a ¾-inch rod (0.218-inch) to such fine sizes may require two annealings in process and 22 to 24 drafts, depending on composition and requirements.

Welding Wire—Various steels, both carbon and alloy, are used to produce welding wire for the manufacture of electrodes and rods for electric and gas welding, and these range from 0.05 carbon to about 1.0 per cent carbon. While considerable gas welding is still employed, by far the greater proportion of welding operations are performed by the electric-arc process. Wire for gas welding is copper coated and used in straight lengths, composition being governed by properties required in the weld metal.

Electric-arc welding is divided into two classifications: manual welding and automatic welding.

Wire for manual welding is furnished in straightened and cut lengths with surface as free from wire-drawing lubricant as possible to enable electrode manufacturers to apply a flux coating by an extrusion method. Welding steels are made especially for this application, and selection of heats or portions of heats is mandatory to conform to required composition limits and ranges. Low sulphur, not exceeding 0.035 per cent, is generally preferred. Most of the grades of steel made for welding electrodes are of the rimming type, with carbon in the neighborhood of 0.10 to 0.15 per cent. However, there is also some demand for rimming-type steel with 0.31 per cent carbon as the maximum of the range. Killed steels of 0.55 to 0.65 per cent carbon and 0.90 to 1.10 per cent carbon are also used to some extent, along with some alloy grades for special purposes such as resistance to abrasion. Composition of

Table 40—IV. Common Uses of Various Commercial Grades of Wire

Grade Name	Some of the Common Uses
Bright	
Bessemer wire.	Nails, flat-head wood screws, tie wires, stove rods, brace rods, low-grade furniture springs, etc.
Bright soft	
Bessemer wire.	Pail bails, double-pointed tacks, jack chains, trace chains, weldless chain, heavy window screens, shelf brackets, staples, market wires, etc.
Bright basic wire	Round-head wood screws, bolts and rivets, large head nails, caster pins, stove bolts, link fabric, cork-screws, electric-welded chain, chaplets, etc.
Bright soft basic steel	Weldless chain, buckle tongues, buckles, electric welding, fine rivets, special screws, link fabric, etc.
Coppered or liquor-finish Bessemer wire.	Common grade furniture springs, tire wires, button hooks, chairs, chair rods, curtain rods, carpet beaters, etc.
Coppered or liquor-finish basic wire.	Pail bails, link fabric, reeds, transom rods, cork-screws, damper rods for pianos, piano pedal rods, etc.
Coppered or liquor-finish soft Bessemer wire	Pail bails, bird cages, buckles, button eyes, spring seat supports, jack chains, piano actions, chairs, coat and hat hooks, garment hangers, hog rings, rapid tag fasteners, rat traps, and similar purposes.
Tinned	
Bessemer wire...	Link fabric, lantern handles, etc.
Tinned soft	
Bessemer wire...	Pail bails, lanterns, bird cages, buckles, card racks, egg beaters, etc.
Tinned basic wire.	Link fabric, bird cages, buckles, lanterns, skewers, cork-screws, etc.
Tinned soft basic wire	Link fabric, buckles, and similar articles.
Annealed wires ...	Bundling, bail ties, rims for wash boilers, check rowers, etc.
Annealed and cleaned basic wire	Balls for bearings, bolts and rivets, toe calks, electric welding, drawer pulls (pressed), and other purposes requiring soft ductile wire.
Annealed and cleaned Bessemer wire	Pump chain, etc.
Galvanized wire ...	Fences, pail bails, rose stakes, clothes lines, bottle stoppers, eave trough hangers, staples, clothes pins, and other uses requiring corrosion resistance at minimum cost.

the steel is governed by composition of the flux coating, the combination being designed to provide certain welding characteristics and specific properties in the deposit metal.

The wire thus serves as the core of the coated electrode and is expected to provide good welding characteristics and a dense, sound deposit. Wire sizes in popular demand are $\frac{1}{8}$ -inch, $\frac{3}{32}$ -inch, $\frac{1}{16}$ -inch, and $\frac{1}{4}$ -inch. Tolerances for diameter, length, camber, and burrs in the straightened and cut wire are listed in the AISI Products Manual for Wire.

The flux coating on the electrode provides a blanket which surrounds the arc during welding and serves as a protection against oxidation from the atmosphere. In automatic electric-arc welding, there are two main processes: submerged-arc and inert-gas. In the former, the welding is shielded by a blanket of fusible material, usually referred to as flux which is fed on to the work. In the latter, shielding is obtained from inert gas such as argon or helium. Both automatic and semi-automatic equipment is used for each process.

Wire compositions vary for submerged-arc welding according to requirements in the deposit metal. The wire is produced with a bright copper finish to improve the contact surfaces and so facilitate the introduction of the relatively high currents used. It is important that the surface of the wire be as free as possible from wire-drawing lubricant which would tend to clog the nozzle through which the wire is fed. Wire for submerged-arc welding is furnished in layer-wound coils with cardboard cores. Dimensions of these coils are designed for insertion into a reel from which the wire is fed through the welding machine. Wire sizes in demand range from about $\frac{3}{32}$ -inch up to about $\frac{1}{16}$ -inch.

The inert-gas welding process calls for smaller wires ranging from 0.045-inch to around 0.125-inch. Wire for this application is furnished with a bright, clean copper coating for the same reason as mentioned above and is wound on spools of the dimensions necessary for insertion in the welding equipment. The required wire compositions vary according to requirements of consumers.

Scratch Brush Wire—This grade of brush wire is produced in sizes from 0.0045-inch to 0.0025-inch. Bessemer stock is used for the production of the wire, and annealings at three different process sizes are required in reducing it to the finish size. The wire is drawn dry to the second annealing, at which point it is bright annealed and drawn wet to the final annealing size; then it is bright annealed and drawn wet to the finish size. As in drawing liquor-finished weaving wire, all drawing from the rod to finish size is performed on continuous machines. This wire is used for jewelers' brushes and other fine wire brushes.

Market Wire—Market wire is a term applied to exact-weight bundles supplied to the jobbing trade. The quality, if not otherwise specified, is a soft basic or Bessemer grade. The standard weight of bundle is 100 pounds, but it is also put up in 63-pound bundles, when so specified.

Uses of Common Wire—The wires described in the preceding section are applied for many purposes, some of the more common are listed in Table 40—IV.

HIGH-CARBON OR SPECIAL WIRES

To select the proper grade of wire for any given purpose, it is necessary to know the mechanical properties required to meet the demands of that particular service. As a result of work correlating a large number of experiments in the wire mills with data collected from

actual experience, it is now possible to supply wire meeting various combinations of mechanical properties, including tensile strength, torsional strength, toughness, etc. These studies have led to the discovery of the principles governing the behavior of high-carbon stocks, and these principles determine the processes to which the wire must be subjected to obtain the desired results. Other elements of uncertainty have been removed by careful standardization, so far as possible, of the manufacturing practices for producing the various classifications and grades of high-carbon steel wires.

Rope Wire—The stock used for manufacturing high-carbon rope wire varies in carbon content from 0.20 to 0.90 per cent. The coarser the finish size and the higher the tensile strength required, the higher must be the carbon content of the stock from which the wire is produced. At least one and as many as three patentings are necessary to produce the various grades and sizes of rope wires. The finer the size and the higher the tensile strength of the finished wire, the greater the number of patentings in the process of manufacture in order to reduce the wire and obtain the proper mechanical properties at the finished size. The finer the size and the higher the carbon content of the wire, the lighter must be the reduction in cross-sectional area per draft. Practically every size and grade of rope wire must be produced from a different final patenting size in order to obtain the proper mechanical properties in the finished size.

Music Wire—Music wire is the term applied to wire having the physical requisites necessary for use in musical string instruments. Although commonly known as music wires, these wires are generally referred to as piano wire, harp wire and mandolin wire, depending upon the purpose for which it is used. In conjunction with the manufacture of music wire, there is a grade of wire manufactured known as music spring wire. This wire is not quite as high in quality as piano wire and is used for high-grade, cold-wound, high-tensile-strength springs.

Piano Wire—Piano wire has rightly been called the "specialty of specialties." It represents the highest attainment in the art of wire manufacture. The wire possesses the highest tensile strength of any form in which carbon steel is used for purely stress-resisting purposes. It is made from the finest quality of steel, having a carbon content of from 0.80 to 0.95 per cent. Different grades and sizes require varying drafting and heat-treating practices to develop the proper tensile strength and toughness.

In addition to the mechanical properties required, piano wire must possess acoustic properties. Attainment of this requirement is dependent upon the accuracy of the size, the soundness of the steel and the finish of the wire. Furthermore, the tension required for different pitches and lengths means that high elastic limit with uniform tensile strength is imperative for a given wire. The toughness must be developed to a maximum for the tensile strength to permit forming the wire into loops or eyes by the bass-string manufacturers. The bass strings must also be flattened locally on the ends to form an anchor for the covering wire. Tensile strengths for the highest grades of piano wire are given below, from which data it should be noted that the higher strengths are obtained in the finer sizes.

Kind of Wire	Diameter in Inches	Tensile Strength (Lb. Per Sq. In.)
Bass Strings	0.035 to 0.067	290,000 to 351,000
Treble Strings	0.029 to 0.049	319,000 to 390,000

Tinned and Liquor-Finish P.S.R. Tire Wire—Tinned P.S.R. (Plow-Steel Rope) Tire Wire is used in the manufacture of automobile tires, the initials P.S.R. indicating the grade of steel used. This product receives special care at every stage of its manufacture to be sure it will meet requirements, which are very exacting. It therefore receives a rigid inspection. It must be perfectly round and true to gage, and possess the proper tensile strength, torsion and elongation qualities. Practically all consumers require this wire to stand a torsion test of ten twists per inch under tension of a five pound weight, usually in a length of six to eight inches. Some demand only a minimum breaking weight while others specify both a maximum and a minimum. A wire having the high tensile strength demanded, that will withstand a minimum of 80 twists in 8 inches, must be free from all steel defects, such as pipe, seams, and segregation.

Spoke Wire—Satisfactory spoke wire to be used in the manufacture of automobile wire wheels must possess special mechanical properties. It is made from sound stock, free from all imperfections, so that a good finish can be obtained on the final product. The method of manufacturing spokes requires that the wire shall be capable of being swaged, headed and threaded. It is stiff enough to withstand the loads to which it is subjected and yet it possesses considerable toughness and ability to withstand the shocks and vibration that are imposed on it in service. The increase in strength obtained by heat treatment, plus that developed in the wire by drawing to the finished size, results in a tensile strength ranging from 140,000 to 160,000 pounds per square inch. It is furnished in sizes ranging from 0.0645 to 0.2045 inch, either in bright or liquor-finished grades. Bicycle-spoke wire (usually 0.0795 inch in diameter), also follows this general requirement, but is normally supplied with a drawn galvanized finish.

Baby carriage and similar types of wire wheels do not require the high tensile strength developed in the wire for automotive wheels. A bright or tinned low-carbon basic wire, drawn direct to finished size from the rod without any intermediate heat treatment, is furnished for this purpose. It has a tensile strength of 90,000 to 110,000 pounds per square inch and is usually supplied in sizes ranging from 0.0625 to 0.162 inch.

Valve Spring Wire—Formerly, automobile engine speeds averaged about 2200 r.p.m., but this speed has since been increased up to 3800 to 4200 r.p.m. Ordinarily, any good commercial oil-tempered wire with a carbon content of from 0.60 to 0.80 per cent would make a satisfactory valve spring capable of giving dependable service at 2400 r.p.m. At 3300 to 4200 r.p.m., however, there is a terrific increase in the fatigue stresses so that failures of ordinary wire frequently take place in less than 1000 miles of operation. But by careful selection of material at each stage in the manufacture of the wire, all the requirements of the high-speed valve spring have been met. Some of the precautions taken with this material are: (1) Use of basic steel of exceptional cleanliness. (2) Generous cropping from both top and bottom of the ingot. (3) Inspection and conditioning of billets. (4) Pyrometric control of rod temperatures. (5) Extra heat treatments from rod to finished wire. (6) Special heat treatment on finished wire. (7) Special tests, including sharp right angle bend tests in four directions. (8) Tensile tests to destruction from each end of every bundle. (9) Spring winding and extension tests for uniformity of temper. (10) Vibration tests of sample springs on motors.

Tempered Wire—Tempered wire is made in rounds, flats and rectangles. The most important feature of this class of material is its temper, which depends upon both

the composition and heat treatment. The tensile strength of this product may ordinarily be required to range from as low as 160,000 pounds per square inch to as high as 300,000 pounds per square inch. Occasionally, strengths in excess of 300,000 pounds per square inch are required. The tempering is accomplished by passing the wire through a continuous tempering furnace in which the wire passes successively through a heated muffle or heated tubes or a hot lead pan, an oil quenching bath and a cold lead pan.

The temperatures of the muffle, tubes and lead pans are controlled automatically by pyrometers which hold each of these at a very uniform temperature and consequently give uniform temper to the wire within very close limits.

Tinned armature-binding wire is another specialty wire. It is made in several grades, which differ in tensile strength. The specifications require a product having high elastic limit and high elongation combined with toughness. To obtain this combination, important essentials are: first, steel of the proper composition; second, proper heat treatment; third, a uniform drafting practice; fourth, a tinning temperature which will give a good finish and will aid in obtaining the desired physical properties in the finished wire.

Metal-Stitching Wire was developed for fastening non-stressed parts in automotive assembly operations and is now also used in the assembly of non-stressed aircraft structures as well as in a multitude of other types of assemblies. In this application, the wire is fed from a spool through a stitching machine which cuts the wire to length, forms it into a staple and drives the legs of the staple through the material to be fastened. In most cases, the legs are folded back or clinched after penetrating the two or more layers being assembled. This wire is frequently used to unite metal to metal or metal and non-metal parts, such as rubber, fiber, felt or plywood. The tensile strength of wire for these applications may range from 200,000 to 330,000 pounds per square inch, depending on the thickness and type of metal required to be penetrated by the fasteners. For this purpose, it is evident that the wire must be very uniform in temper and possess great toughness. Metal-stitching wire is made with several different finishes, according to whether a protective or decorative finish is desired.

Other Special Wires—Besides those already mentioned, there are numerous other grades of drawn high-carbon wires produced. In practically every case, the various grades differ from each other in one or more respects, such as stock, temper, quality of toughness, finish and coating. Practically all of the high-carbon wires are classified as "special wires." By special wires is meant wires made to uncommon specifications, either as to finish or mechanical properties. The range of stocks used for special wires may include basic open-hearth steel from the lowest carbon steel up to a steel having a content of 1.20 per cent carbon. There are at least twenty-five different carbon grades, and these are further increased by variations in manganese and other elements. Following selection and rolling of the stock, preliminary testing of the rods is usually necessary to prevent unnecessary work on stock which will not give the desired qualities in the finished wire. Extreme care must be used in the cleaning, coating, and every step of the manufacture. Special drafting, various heat treatments, both before and after the complex drawing processes, special testing and extraordinary precautions must be followed to secure the unusual properties and degree of uniformity that may be required.

STAINLESS-STEEL WIRE

Stainless-steel wire has become an established commodity for many purposes because of its utilitarian nature, its stainless and heat resisting characteristics, and its ability to withstand a variety of forming operations. Among the established products are cold-heading wires for the manufacture of bolts, rivet pins, and screws; tinned non-magnetic armature-binding wire for use in highly-powered motors for Diesel engines, and other motors which require a high-tensile non-magnetic wire; welding wires furnished for all methods of welding; spring wires to be used where resistance to chemical solutions and atmospheres is a necessity; weaving wires for a wide range of screens from very fine mesh flour sifters to coal and coke screens and continuous high-temperature conveyor belts; rope wires for specialized purposes such as mine sweeping, aircraft-control cables and strands for yachts and ships. The unusual properties of stainless steel which permit a lasting bright finish to be developed, combined with its utilitarian nature, give it great intrinsic value when used as display racks, dishwasher and refrigerator racks.

While Types 430, 410, 302, 304, and 305 may be considered the more popular grades, almost the entire list is drawn to wire for a great variety of uses.

The processing and drawing of these steels are similar, in general, to that of the carbon-steel wires. They will differ somewhat in individual practices because of the inherent nature of the alloys, such as resistance to some acids and activation by others, all of which combine to make the coating and lubricating problems in drawing stainless-steel wires one of meticulous control.

STAINLESS COLD-ROLLED STRIP STEEL

Stainless cold-rolled strip steel is manufactured from hot-rolled annealed and pickled strip by cold rolling in mills equipped with ground or ground and polished rolls, depending upon the surface finish requirements. Strip steel is normally supplied up to and including 23 $\frac{1}{16}$ -inches wide in a great many of the more common types of stainless steels.

A range of mechanical properties can be provided by taking advantage of the possibilities offered by combining the influences of chemical composition, cold work (cold rolling), and heat treatment. These are varied according to the mechanical properties desired. Physical properties developed in this manner are referred to as tempers and are associated with the capacity each temper level possesses in regard to resistance to cold deformation. In the straight chromium grade (400 series), only three tempers are generally recognized by the industry, viz., full hard (Rockwell C 20 minimum), No. 4 Temper (approximately Rockwell B 89-90), and No. 5 Temper (approximately Rockwell B 83 maximum). In the chromium-nickel steels (300 series), No. 4 Temper will show approximately 80-90 Rockwell B and No. 5 approximately Rockwell B 83 maximum. In addition to these soft tempers, a variety of hard tempers is available classed as $\frac{1}{4}$, $\frac{1}{2}$, and $\frac{3}{4}$ hard and full hard. These are based on minimum values for tensile strength or yield strength or both.

Finishes—Various surface finishes are possible in cold-rolled stainless strip steel although they are gradations. For example, a dull finish without luster can be produced by rolling on rolls roughened by chemical or mechanical means. A luster finish can be produced by rolling on rolls having a moderately high finish. For the very best or high luster, all of the treatments must be carefully done and the rolling performed on highly-polished rolls.

Basically only two finishes are recognized in the stainless steel strip industry:

- No. 1 Finish, which is cold rolled, annealed, and pickled, and
- No. 2 Finish, which is cold rolled, annealed, pickled, and rerolled.

Because of the difference in the alloy contents between the straight chromium and the chromium-nickel steels, it is possible to anneal the former so that a bright finish results, while the annealed and pickled chromium-nickel steel will be dull. In the chromium-nickel steels, No. 1 Finish is classed as dull, while the hard tempers are relatively bright as will be the No. 2 Finish, No. 4 Temper.

Since the annealed and pickled straight chromium steels result in a bright No. 1 Finish, it is possible to change the processes to enhance this finish in making a No. 2 so that it is even brighter. Dull finishes are also available in these grades where the requirements are necessary.

There are three edges available in stainless strip depending upon the width and thickness:

- No. 1 Edge is a rolled edge, either round or square as specified, and is recommended when a very uniform width is required. It is limited to strip of approximately 5 inches in width and under.
- No. 3 Edge is an approximately square edge produced by slitting. This edge is not burr free. Width is not a limiting factor in furnishing this edge.
- No. 5 Edge is an approximately square edge produced by rolling or filing for the primary purpose of removing burr originating in the slitting operation. Width is not a limiting factor in furnishing this edge.

COLD-ROLLED CARBON-STEEL STRIP

Cold-rolled carbon-steel strip is manufactured in a variety of finishes, tempers, and edges, all depending upon the end use. By common custom, cold-rolled carbon-steel strip is made in a width range of over $\frac{1}{2}$ inch to $23\frac{1}{16}$ inches and up to 0.2499 inch thick. It may have a carbon content of 0.25 per cent maximum; material of this form containing more than this amount of carbon is considered as flat carbon spring steel.

Cold-rolled strip is produced in coils on any of the conventional reversing mills, tandem mills, or by single-stand rolling. While the strip may be supplied in coils, it can be furnished in cut lengths by the straighten-and-cut process.

Before cold rolling, the mill scale is removed from the hot-rolled strip by pickling. From this point, the strip may be cold reduced to final thickness or to some intermediate gage where it is annealed and further cold reduced to obtain the desired temper and gage.

Temper—Many degrees of temper are possible in the manufacture of cold-rolled strip by controlling the combinations of cold rolling and annealing. However, many years of use have brought certain ranges of temper into common usage, and these have come to be recognized by number, as follows:

Temper	Maximum Carbon Content (Per Cent)*	Rockwell Hardness	
		Minimum	Maximum**
No. 1 (Hard Temper)	0.25	B-84 ^a	
		B-90 ^b	
No. 2 (Half Hard)	0.25	B-70	B-85

No. 3 (Quarter Hard)	0.25	B-60	B-75
No. 4 (Skin Rolled)	0.15	—	B-65
No. 5 (Dead Soft)	0.15	—	B-55

- Ladle analysis
- Approximate
 - * For thicknesses 0.070 inch and greater
 - * For thicknesses less than 0.070 inch.

No. 1 Hard Temper is a very stiff springy strip intended only for flat work where no bending is required.

No. 2 Half Hard Temper is less stiff than No. 1, but is intended for limited cold forming and will only withstand 90 degree bends made across the direction of rolling.

No. 3 Quarter Hard Temper is intended for limited bending and cold forming and can be bent 90 degrees in the direction of rolling and 180 degrees across the direction of rolling over its own thickness.

No. 4 Skin-Rolled Temper is intended for cold forming such as bending flat upon itself in any direction and for deep drawing. The purpose of the skin pass is to prevent the formation of stretcher strains.

No. 5 Dead Soft Temper or annealed temper is intended for severe cold forming and deep drawing where the formation of stretcher strains are not objectionable.

Three finishes have come to be recognized as standard within the industry, again by common usage.

No. 1 Dull Finish does not have any luster and is actually made rough intentionally by rolling on rolls roughened either mechanically or chemically. This finish is suitable where paint adherence is desired, or in deep drawing since the lubricant will stick to it.

No. 2 Regular Bright Finish is cold rolled on rolls having a moderately smooth finish. It is suitable for many requirements, although it is not generally applicable to plating.

No. 3 Best Bright Finish is the highest luster finish produced by cold rolling and is particularly suited for electroplating.

Cold-rolled carbon-steel strip in No. 1 Dull Finish, No. 2 Regular Bright Finish, and No. 3 Best Bright Finish is also available with "rolled-in" designs produced through the use of embossed rolls.

Six types of edges have become recognized as standard.

No. 1 Edge is a prepared edge of a specified contour (round, square, or beveled) which is produced when a very accurate width or edge finish is required.

No. 2 Edge is a natural mill edge carried throughout the cold rolling from the hot mill without additional processing of the edge.

No. 3 Edge is an approximately square edge produced by slitting.

No. 4 Edge is a rounded edge produced by edge rolling either the natural edge of the hot-rolled strip or the slit-edge strip. This edge is produced when an approximate round edge is desired and when the finish of the edge is not important.

No. 5 Edge is an approximately square edge produced by rolling or filing of a slit edge to remove the burr.

No. 6 Edge is a square edge produced by edge rolling the natural edge of the hot-rolled strip or slit-edge strip when the width tolerance and finish required are not so exacting as for No. 1 Edge.

FLAT WIRE

Flat wire is ordinarily cold rolled from a drawn round wire, properly annealed and treated to permit additional reduction in rolling and to produce a reasonably bright

surface. Natural round, smooth edges are produced by this method of manufacture. Flat wire can be made up to $\frac{1}{2}$ inch wide, with ratio of width to thickness being in accord with good manufacturing practices. Normally, flat wire is best produced in sizes up to $\frac{3}{8}$ inch in width with roughly a 5-to-1 ratio in thickness. However, this ratio can in some instances be as high as 10-to-1 or 15-to-1. Flat wire is available or can be made to include soft or medium or hard rolled tempers. This material has a variety of uses: some typical applications are in window-shade roller springs (Curtain Spring Wire), leaf-

type feeler (thickness) gauges (Feeler Gauge Steel), electricians' tools for "fishing" wires between walls and through conduits (Fish Tape Wire) and staples for many fastening operations involving stapling machines (Flat-Preformed Staple Wire). Flat wire can be formed into flat wound springs. It also is used in miscellaneous products where spot welding is required in assemblies. Types 302 and 430 are grades of stainless steel which have been most commonly furnished in the form of flat wire, although flat wire can be produced from many of the other grades.

SECTION 11

SOME FABRICATED STEEL-WIRE PRODUCTS

Importance of Fabricated Wire Products—Steel wire is fabricated into thousands of different kinds of articles, which are used for a great variety of purposes. In a list of such articles will be found many items of common use, such as automobile and bicycle wheel spokes, hoops, rivets, bolts, chains, buckles, cotterpins, sifting screens, wire netting, wire cloth, and a host of others, each of which consumes large amounts of wire. Since even the briefest description of the fabrication of all these articles would require a great deal of space, only a few of the more important commodities, in the fabrication of which great tonnages of steel wire are consumed annually, are briefly discussed here.

WIRE NAILS

The wire nail has almost entirely displaced the old-style cut nail, being better in all respects and much cheaper. A review of any modern nail catalogue will show that there are many varieties as to length, size and style, ranging all the way from the tiniest tack to the long heavy spikes. The common finish on nails is obtained by tumbling, though large quantities are galvanized, tinned, blued by oxidation, or cement coated. In the last named process the nails are given a light coating of some compound of a resinous nature which greatly increases their holding capacity.

Nail Machines—All steel-wire nails are made in automatic machines. These machines differ greatly in size and in design, but the principle of operation is much the same in all of them. Nails are made on a machine by five distinct operations; namely, (1) forming the head, (2) feeding the wire, (3) pinching the wire, (4) cutting off the wire and forming the point, (5) expelling the nail.

The head of the nail is formed by compressing and flattening against a die the portion of the wire which projects beyond this die and remains after the previously-formed nail has been cut from the wire. This compressing and flattening is done by a hammer which is attached to a reciprocating member, called a hammer stock, which in turn is actuated usually by a crank and pitman. The amount of wire which projects beyond the die governs the size and thickness of the head and is regulated by adjusting the cutting knives to the proper distance from this die. The various shapes of heads are obtained by cutting the desired depression in the die. This die is split, that is, made in two parts, one fixed or stationary and the other movable.

Feeding—After the head is formed, the hammer moves away from the die, and the die opens up and allows the feed mechanism to push the wire, with a nail head on the end, the correct distance through the die to give a nail of the length required. The feeding mechanism is driven by an adjustable crank on the flywheel of the machine, and, by adjusting this crank, various lengths

of nails can be obtained. This feed mechanism also pulls the wire through a series of staggered rolls, as it leaves the reel, to straighten it.

Pinching—When the hammer has reached the end of the stroke, the wire has been fed the correct amount for the nail required and the die closes to pinch the wire. This pinching action is motivated by a cam on the crankshaft.

Cutting—Immediately after the wire is pinched, two knives, each attached to a lever, move together and cut the wire. These cutting knives are ground to form the point on the nail at the same time that the cut is made. This point is formed by pressing the wire into the shape required, and, in doing so, some of the metal is squeezed out or protrudes between the knives and is cut off by them. These cutoff particles are called whiskers. The cutting levers to which the knives are attached are actuated by various forms of mechanisms deriving their motion from the crankshaft.

Expelling—Sometimes, because of dull knives or insufficiently close adjustment, the nail will still adhere to the wire when the cutting knives open up. The cutting knives open up on the return stroke of the hammer, and, in order to remove this adhering nail, an expeller comes into action, knocking the nail downward out of the path of the hammer and breaking it off. The hammer on the return stroke forms another head on the wire for the next nail, the wire being pinched during this stroke. The finished nails drop into a pan placed on the floor beneath this mechanism.

Finishing Common Nails—The nails in these pans are collected and placed in a tumbler, care being taken to have nails of only one kind in the tumbler at a time in order to avoid mixing. Into this tumbler some sawdust is also placed. The tumbler has projections on the inside, causing the nails to be churned when it is rotated. This churning polishes the nails and removes any whiskers which may have adhered to the nail by a thin fin of metal. The sawdust absorbs the grease and oil which the nails collected during their manufacture. The cover of the tumbler has perforations or a screen which allows the whiskers and sawdust to pass through but holds back the nails. After the nails are tumbled sufficiently they are packed in kegs ready for shipment.

WIRE FENCE

An increasing amount of steel wire is used for fence purposes. Today an enormous tonnage of galvanized steel wire is fabricated into woven-wire fence and barbed-wire fence.

Woven-Wire Fence—There are a great many types of woven-wire fence, varying in style or design, and each may be made up in many different sizes. In a general way, the various styles resemble one another in that

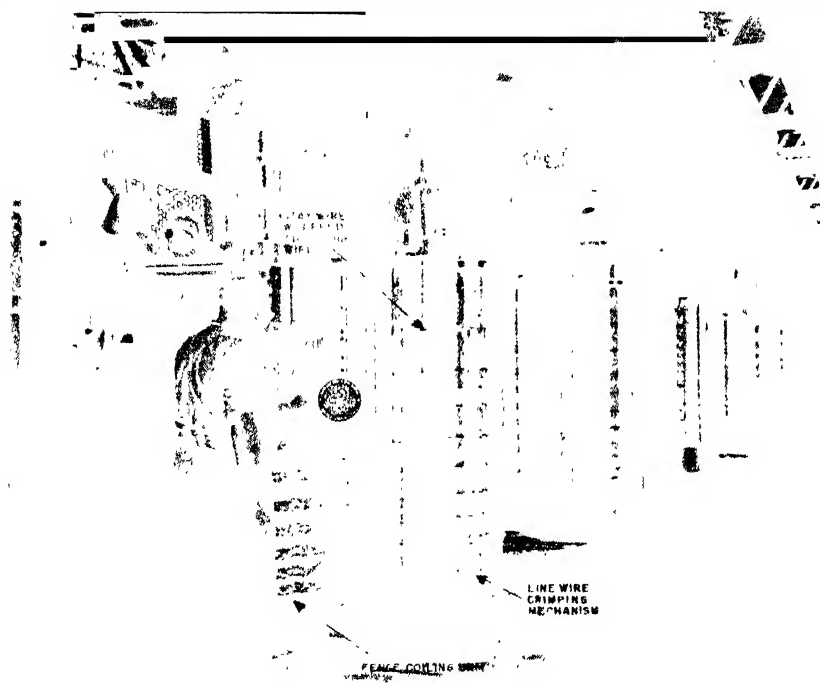


FIG. 40-34. Battery of machines producing various types of woven wire fence.

they have several horizontal or lateral wires which are secured in position with vertical or diagonal stay wires, the former being stronger and stiffer than the latter and provided at frequent intervals with tension curves, to take care of expansion and contraction due to temperature changes. The crossing of stay wires, with the horizontals, form the meshes, which may be quite large, as in cattle fences, or very small, as in poultry netting, and may be of any one of four forms, namely, triangular, rectangular, hexagonal, or diamond-shaped. They are also fastened together in various ways. In the cut-stay fence there is a short piece of stay wire for each space, having its ends twisted about the laterals. The stays may be electrically welded to them, or the two wires may be woven together. In any case, the work is done by specially constructed and rather complex machines from which the coils of fence will emerge all ready for the market (Figure 40-34). In the making of these fences, a most rigid system of inspection and tests is maintained, not only in the drawing and galvanizing of the wire, but also in the weaving room, in order to turn out as perfect a product as possible. These same fences made of extra-strong wire constitute exceptionally good reinforcement for concrete work. The leading fence manufacturers also produce a line of gates, special fittings, steel posts, and other articles used in fencing.

Barbed-Wire Fence—In this fence, two wires, usually galvanized, and known as line wires, are twisted together, and, at regular and frequent intervals (3 to 6 inches apart) either two or four barbs, which may be round, flat or oval in section, are wound about one or both of these line wires. The barbs are diagonally cut so as to produce a long sharp point extending at right angles to the line wires. Here again, a great variety of styles and sizes of fence are made by fast-running and rather complex machines. The bulk of barbed wire, however, has the two line wires of No. 12½ wire, while the barbs are usually made of No. 14 wire. The fence is furnished to the market in 80-rod, or in "catch-weight," spools.

CONCRETE REINFORCEMENT

Concrete reinforcement is a steel fabric which, as the name implies, is used to reinforce concrete work in its numerous applications. Although concrete offers great resistance to compressive stress, it is lacking in tensile and bending strength unless reinforced by some material which possesses these characteristics to a marked degree. In this respect steel excels, particularly cold drawn steel, because of its high tensile strength and high yield point. Concrete-reinforcing wire fabric is electric welded: the longitudinal and transverse wires are fixed in position by an electric-welding machine designed to space the wires accurately and weld them together at the contact points.

The accurate spacing of the longitudinal and transverse members is important, since it not only enhances the reinforcing value of the fabric by placing the steel exactly where it is designed to be, but also facilitates handling of the material. Furthermore, each welded intersection develops a positive anchor in the concrete.

Electric-welded wire-fabric reinforcement has a varied application. It is used to reinforce concrete roads, buildings, dams, etc. As a matter of fact, every type of concrete construction should be reinforced. This fabric is used in such precast concrete products as pipe, posts, and slabs. It is supplied in flat sheets or rolls of desired length and width. Standard rolls are 60 inches wide by 150 feet long. Welded wire fabric which is too heavy to be rolled must of necessity be supplied as flat sheets.

PRESTRESSED CONCRETE

Prestressing of concrete is the introduction of desirable compressive forces into a concrete member. These compressive forces are designed to offset or neutralize any subsequent tensile forces which occur when the concrete member is loaded. Since concrete has very little tensile strength, prestressing permits a concrete member to withstand tensile forces without cracking.

Prestressing is done by two general methods:

(a) *Pretensioning.* Steel wire or strand is tensioned on a precasting bed and then concrete mix is poured around the steel. When the concrete has attained full strength, the strand or wire is released from its tensioning apparatus, and the tensile forces of the steel induce equal and opposite compressive forces in the concrete. The bond between the concrete and steel continues to hold the steel in tension and the concrete in compression.

(b) *Posttensioning.* Steel wire or strand is threaded through ducts in a precast concrete member and tensioned by means of end fittings. The tensile forces of the steel induce equal and opposite compressive forces in the concrete members.

Wire and strand for prestressed concrete must have high tensile strength—usually on the order of 250,000 lb. per sq. in. Both wire and strand are furnished from cold-drawn wire, and after the final draft or stranding they are stress-relieved by immersion in molten lead at temperatures of about 800° F. Stress relieving improves the elastic properties and ductility of the product and also burns the surface clean of any residual matter on the wire or strand.

The greatest use of prestressed concrete to date has been in bridge and tank construction and in the manufacture of pressure pipe.

BALE TIES

Large quantities of wire are used annually for tying or bundling bales, bundles, boxes, or packages of every description. While there are several styles of bale ties, each, however, is made of wire that is high in tensile strength and fairly pliable, a goodly percentage of this commodity being made of annealed medium-carbon steel having a light oxide finish. The wires are cut to length, and one end is fabricated to form a means of fastening the two ends together. Lengths and sizes vary between wide limits, but tie wires are made of sizes No. 13 to 16½, inclusive, plain or galvanized. They are put up and sold in bundles of 250 and 500 per bundle, the ends of the bundles being wrapped in burlap.

WIRE ROPE

A single, solid bar of steel, for example a bar one inch in diameter, is very stiff and will stand relatively little "back-and-forth" bending before breaking, but if this same bar of steel is drawn and then formed into a twisted bundle, or "strand," of small wires having the same aggregate sectional area of metal, it will be found to have gained greatly both in flexibility and life, and usually, also, in tensile strength. These reasons account for the large amount of steel-strand wire rope that is used as haulage rope, hoisting rope, hawsers, guys, etc., and for such purposes as oil-well drilling, ships' rigging, tramways, elevators, and so on. These ropes consist of iron or steel wires ranging in tensile strength from 85,000 to 280,000 pounds per square inch or even more, the wire of very high strength being made from high-carbon steel. Wire for ropes used for guy or standing purposes is generally galvanized.

Fabrication of Wire Rope—A full description of the fabrication of wire to form the various types and kinds of rope would be very lengthy, indeed. Consequently, only a few of the simplest examples can be cited here to illustrate a few of the principles involved in steel wire ropemaking. The first wire rope was made of straight wires held together by fine wires which were wrapped about them at intervals, but this construction was unsatisfactory for obvious reasons. The first great improvement consisted in twisting a number of wires together to form what was called a strand, and then twisting a

number of these strands together to form a rope. At first both the strands and the rope consisted only of wire, and the strands in the rope were few in number, with the result that it lacked flexibility and wearing qualities. But this lack was soon remedied by the introduction of the hemp core, which, together with an increase in the number of strands and the number of wires in a strand, marked the beginning of really successful wire rope manufacture—from the standpoint of the user, at least. So, a steel wire rope, as it is made today, may be defined as a group of strands, usually in one layer, twisted together helically and symmetrically, with a uniform pitch and direction, around a central core of hemp or wire, each strand in turn consisting of a group of steel wires similarly twisted, in one or more layers, around a central core of wire or hemp. The principal operations of ropemaking are two in number and are known as **stranding** and **laying**.

Stranding—Stranding consists of bringing the individual wires together in a predetermined pattern, and twisting or "stranding" them uniformly in concentric rings about a central wire, in either a clockwise or counterclockwise direction, to form the completed strand (Figure 40—35).

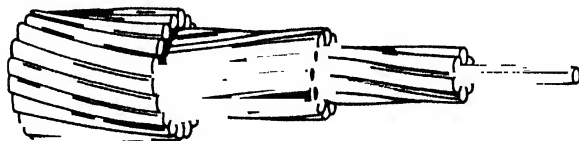
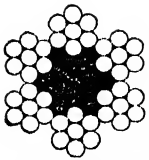


FIG. 40—35. Arrangement of wires in a galvanized bridge strand. Note the increasing number of wires, starting at the right, are 1, 7, 19, and 37, respectively.

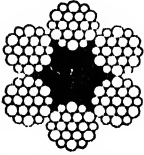
Many types and geometrical patterns of wire-rope strands are possible and are made for special uses. Most wire-rope strands, however, fall into three classifications or basic types: 7-wire, 19-wire, and 37-wire. The 7-wire strand is made by covering a center wire with a layer of 6 wires stranded or twisted about it in uniform helices. Strands of this kind made of uncoated wires are closed about a fiber core to form a 6x7 haulage rope (Figure 40—36, Diagram A). When made of galvanized wires, this construction is a 6x7 guy or standing rope. If a cover of twelve wires is twisted about a 7-wire strand, a 19-wire strand is formed. There are several modifications of the basic 19-wire strand used in the fabrication of 6x19 hoisting ropes, the most widely used of all wire ropes. Adding another cover of 18 wires to the 19-wire strand produces the 37-wire strand, which is also fundamental in design. Modifications of this basic pattern are used by manufacturers in wire ropes of special flexibility, such as crane ropes and other wire ropes which operate over relatively small sheaves.

Laying or Closing—The final operation in rope making is the closing of the strands around the core. More wire ropes are composed of six strands than all the others combined, as the 6x19 and 6x37 classifications are the most common types. Eight-strand ropes, 8x19 classification, are extensively used on passenger elevators. There are ropes of three, five, and other numbers of strands, but these are comparatively few in number. The largest number of strands in any standard wire rope is eighteen in 18x7 non-rotating hoisting rope.

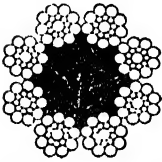
The cores around which the strands are closed are usually either fiber or wire. Fiber cores are standard for wire ropes which are not subjected to heat, crushing and/or heavy loads. Wire cores, usually independent wire-rope cores, are employed where greater strength or greater resistance to distortion and/or heat are required. Other substances, such as plastics, are used to very



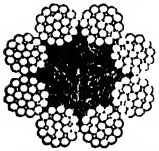
A. 6 x 7 Classification Haulage Rope, consisting of 6 strands of 7 wires in each strand and one fiber core.



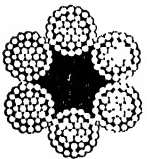
B. 6 x 19 Classification Hoisting Rope, with 6 strands of 19 main wires in each strand and one main fiber core.



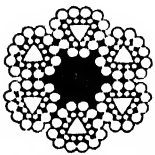
C. 8 x 19 Seale Elevator Hoisting Rope, consisting of 8 strands of 19 wires in each strand - arranged in Seale construction - and one fiber core.



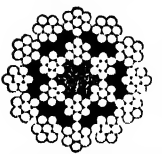
D. 8 x 19 Classification Hoisting Rope, consisting of 8 strands of 19 main wires in each strand - plus 6 filler wires and one fiber core.



E. 6 x 37 Classification Hoisting Rope, with 6 strands of 37 main wires in each strand and one fiber core.



F. 6 x 25 Type B Flattened Strand Lang Lay Hoisting Rope consisting of 6 strands, each containing a single triangular-shaped core wire and 24 round wires and one fiber core.



G. 18 x 7 Non-Rotating Hoisting Rope consisting of 12 strands of 7 wires each over a core of 6 strands of 7 wires each and one fiber core.



FIG. 40-36. Different types of steel wire rope.

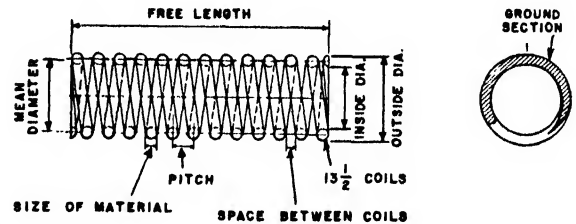
limited extents to meet specific service requirements.

The direction of lay of a wire rope may be either right or left, although practically all wire ropes are right lay. In a right lay wire rope, the strands form a helix about the core similar to the threads on a right-hand screw. If the direction of the lay of the wire in the strands is opposite to the direction of lay of the strands in the rope, that rope is regular lay. If the lay of the wires in the strands and the strands in the rope are of the same direction, that rope is Lang lay. Regular lay ropes are in greater demand than are the Lang lay ropes because of greater inherent stability. Lang lay ropes, however, possess greater resistance to abrasion than do regular lay ropes because of the greater distance that each outer wire in a Lang lay rope is exposed to surface wear.

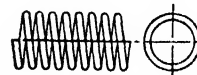
Types of Wire Rope—By changing the number and arrangement of wires in each strand, by varying the number of strands in the finished rope, by making ropes both with fiber cores and with independent wire-rope cores, by closing the ropes so that they are regular lay and Lang lay, right lay and left lay, by producing wire ropes in several grades of steel and in sizes from $\frac{1}{16}$ -inch diameter through 4 inches in diameter, the number of possible wire ropes becomes legion. However, only a small fraction of the possible constructions are sufficient for industrial purposes.

WIRE SPRINGS

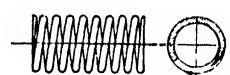
There is a wide diversity in types, shapes and sizes of springs, ranging from the tiny watch hairspring to the large buffer springs used for railroad equipment. The majority of springs are manufactured from medium-



COMPRESSION SPRING

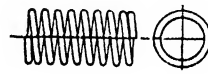


PLAIN ENDS
COILED RIGHT HAND

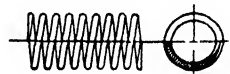


SQUARED OR GROUND ENDS
COILED LEFT HAND

$$\text{TOTAL COILS} = \text{ACTIVE COILS (N)} \quad \text{TOTAL COILS} = \text{ACTIVE COILS} + 2$$



SQUARED OR CLOSED ENDS
NOT GROUND
COILED RIGHT HAND

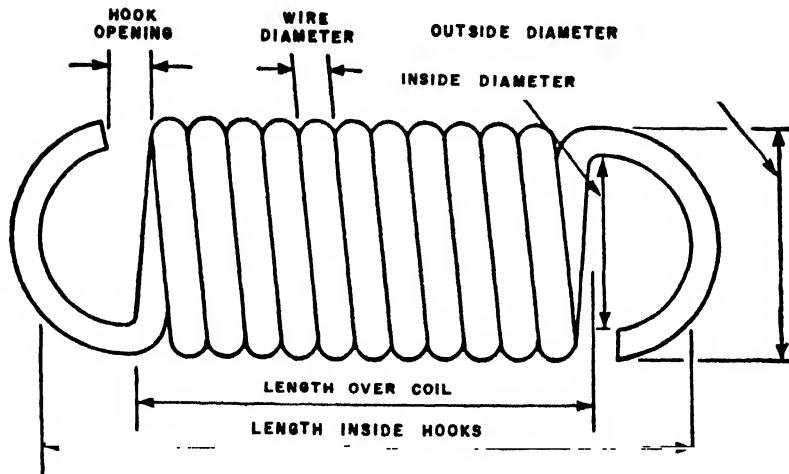


PLAIN ENDS - GROUND
COILED LEFT HAND

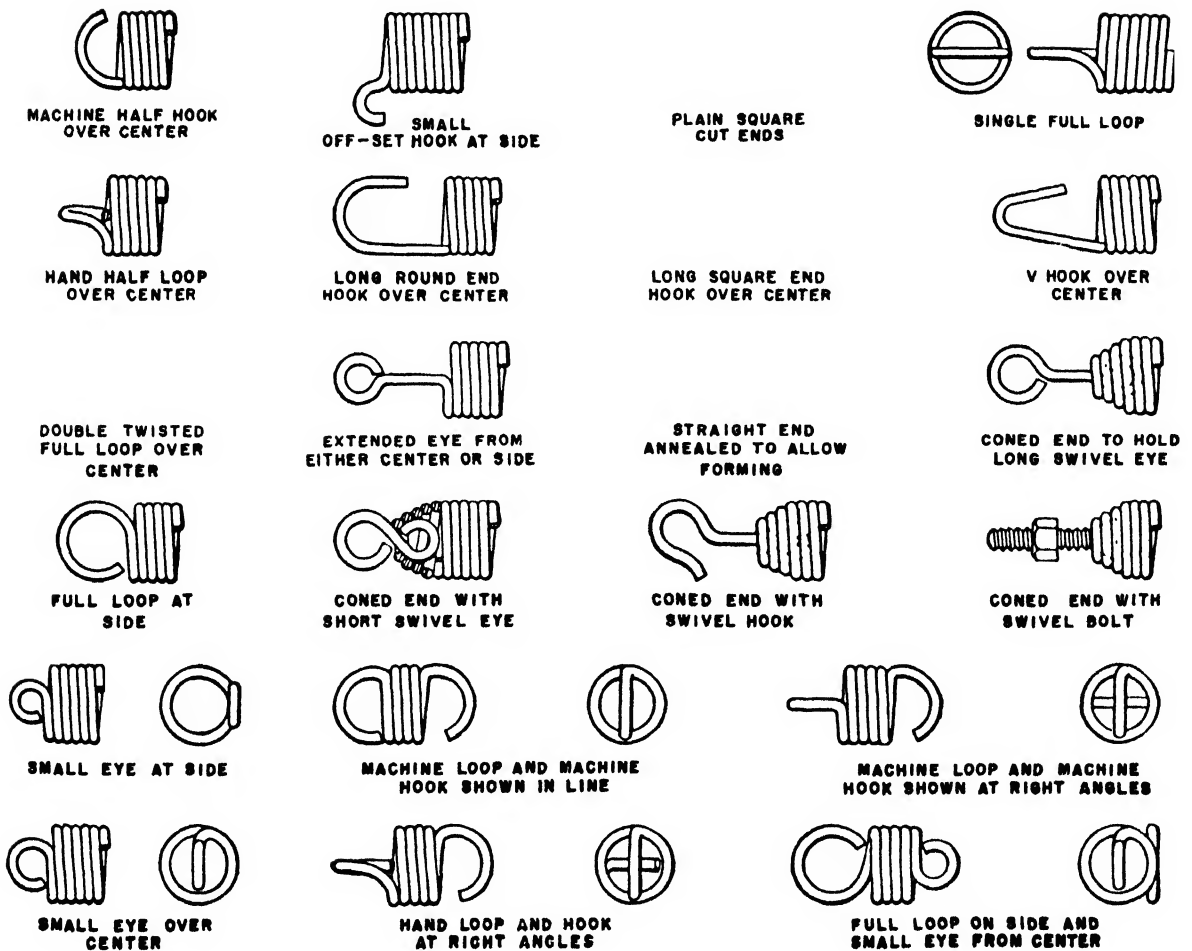
$$\text{TOTAL COILS} = \text{ACTIVE COILS} + 2 \quad \text{TOTAL COILS} = \text{ACTIVE COILS (N)} + 1$$

TYPES OF END FINISHES

FIG. 40-37. (Above) Illustrative sketch of a compression spring. (Below) Types of end finishes for compression springs.



EXTENSION SPRING



TYPES OF ENDS USED ON EXTENSION SPRINGS

FIG. 40—38. (Above) Illustrative sketch of an extension spring, giving nomenclature of parts. (Below) Types of ends used on extension springs.

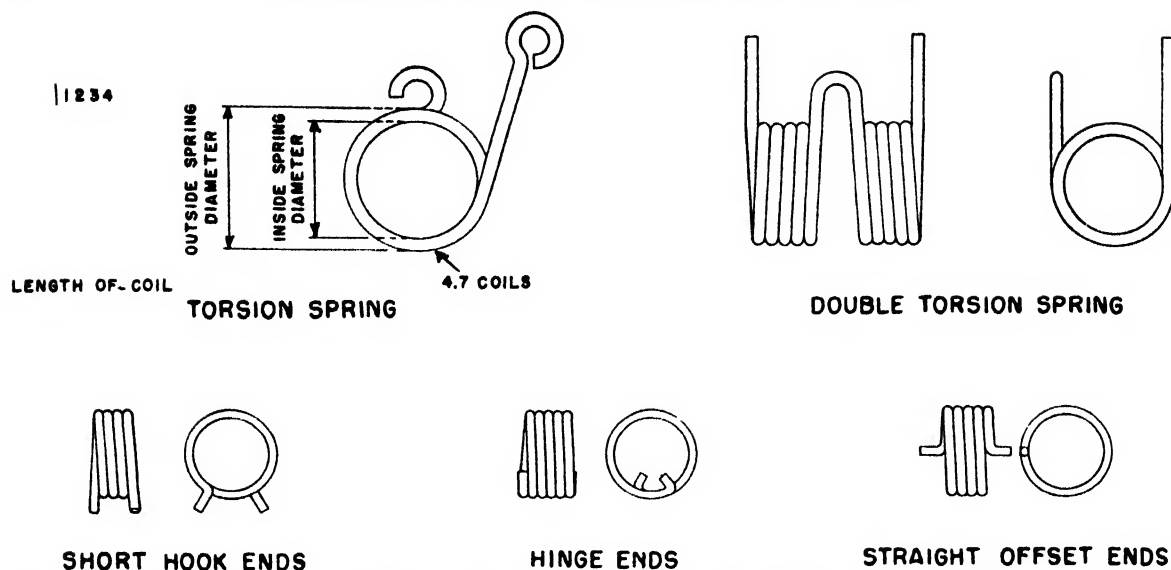


FIG. 40—39. (Above) Illustrative sketches of (left) a torsion spring and (right) a double torsion spring. (Below) Types of ends for torsion springs.

carbon or high-carbon steel; such springs are furnished in the annealed, hard-drawn or pretempered state. As to shape, they may be round, flat or rectangular. Special applications may require stainless steel, and many non-ferrous metals such as brass, phosphor bronze and the nickel alloys are likewise employed for certain classes of use. The following classifications indicate the springs in most common use.

1. **Compression Springs**—These springs are open wound with varying space between the coils and are provided with the ends plain, plain and ground, squared, or squared and ground. They are furnished in a wide range of wire sizes, diameters, lengths, shapes and each and every one must be designed to meet certain load requirements. Figure 40—27 illustrates typical springs of this class.

2. **Extension Springs**—Generally, extension springs are close wound, usually with specified initial tension and, as they are used to resist pulling forces, they are provided with hook or loop ends to fit their varied applications (Figure 40—38). These ends may be an integral part of the spring itself, or may be specially inserted forgings, castings or wire forms to meet individual requirements.

3. **Helical Springs**—The term "helical" springs, although literally applying to any spring which is helically wound, is generally considered as a trade name for extension springs used in great volume by manufacturers of spring beds, couches, and cushions. Many sizes and shapes of this particular category are in general usage.

Any one of the three foregoing types of spiral springs may be strengthened by increasing the wire size, or by decreasing the mean diameter of the helix, but in any case the ability of the material to withstand the load must be of prime consideration.

4. **Torsion Springs**—This type of spring (Figure 40—39) is one in which work is required to wind up the spring about its own axis—never to unwind it. The ends may be finished in a number of ways, the type of end being generally dependent upon the particular requirements.

5. **Upholstery Springs**—These are used to serve as cushion or bed springs. They are normally made in two styles, known as full springs and half-springs and, in

either, one or both ends may be "knotted," that is twisted about the wire at the beginning of the second coil.

6. **Clock and Motor Flat Springs**—These springs are made of cold-rolled high carbon strip steel of proper width and thickness to meet specifications and are tempered, tempered and polished, or tempered, polished and colored. The ends are formed as desired, and the coils are wound spirally and held in compact shape by a retaining clip or ring.

7. **Jig Springs and Specially-Formed Wires**—These represent a group, covering an endless variety of odd shapes and sizes, made or formed in jigs, on automatic wire-forming equipment, or with small hand tools. Springs of this type are generally used to serve as more or less simple parts of mechanisms.

SPRING TERMS

The manufacturers of springs, in common with every other industry, have used terms frequently misunderstood. The most common of these are:

Bluing—A low-temperature heat treatment of springs after fabrication, to relieve the cold-working strains created by coiling and forming.

Tested Spring—A spring which has had its set removed as a result of one or more compressions to solid height.

Scale Testing—A spring which has been tested for load at a given height or heights.

Pitch—Center to center distance between adjacent coils at free height.

Active and Inactive Coils—Two types of coils are present in compression springs, namely, active and inactive. This is prevalent in squared springs where the end coil squared on the next adjacent coil becomes inactive due to its contact with said coil and, consequently, the inactive coils are omitted when computing spring load and stress data.

Initial Tension—Indicates the force or load necessary to cause the coils to start to open. This force or load must be considered as part of the total spring load for stress calculations. This initial tension factor applies only to the close-wound extension type of spring.

BRIDGE WIRE

Bridge wire is the term commonly applied to high-grade rope wire used in the construction of cable sus-

pension bridges, in which the main cables consist of parallel wires compacted to act as a unit. These bridges are remarkable for their graceful beauty, the absence of heavy superstructures, and their extraordinarily long spans. Some of the largest bridges in the world are of this type, including the George Washington Bridge, New York, and the Golden Gate and San Francisco-Oakland Bay Bridges, California. Interesting data on the cable wire used in the San Francisco-Oakland Bay Bridge are: Length of longest spans (West Bay) center, 2,310 ft.; side, 1,160 ft.; vertical clearance, 200 ft.; number of cables, 2; diameter of cables, 28¾ in.; number of wires in each cable, 17,464; diameter of each wire, 0.195 in.; total length of cable wire, 70,815 miles; total length of suspender ropes, 43 miles. Total tonnage of wire in both crossings amounted to 18,200 tons.

Bridge wire requires steel of high quality and extreme care in every step of its manufacture from melt to finished wire. Specifications for the finished wire have required: (1) The wire to be heavily coated with zinc to protect it from corrosion—galvanized to withstand 5 immersions in the standard Preece test. (2) A minimum tensile strength of 225,000 lb. per sq. in. (3) A minimum elongation of 4 per cent in 10 inches. (4) A coating that will permit wrapping without peeling about a round mandrel of a diameter equal to 1½ times the diameter of the wire.

At one time, specifications required the steel to be made by the acid open-hearth or the crucible processes, but it is now generally accepted that steel equally as satisfactory or better can be made by the basic open-hearth process. The wire is made from selected high-carbon steel, the composition of which is held within close limits. The composition of the steel used heretofore has usually been held within the following limits: carbon, 0.75 to 0.85 per cent; manganese, 0.55 to 0.75 per cent; phosphorus, under 0.03 per cent; sulphur, under 0.03 per cent; silicon, 0.15 to 0.30 per cent, with maximum limits for other elements that may be present in small amounts. Some of the more important requirements of the manufacturing practice are listed as follows: The manufacturing processes in the production of bridge wire are very closely controlled in order to produce the required mechanical properties of high tensile strength, toughness, and resistance to fatigue. The bright wire is produced in two common wire sizes, No. 6 and No. 8, the former being more frequently used. The bright wire is usually hot galvanized. In order to prevent undue bending, and to supply wire which is straight, as required for the subsequent spinning of the cables, the wire is finished in five-foot diameter coils. Coils are coupled together to form long, continuous lengths of wire, and are shipped to the bridge site on large reels, from which the wire is unreel during the process of spinning the cables.

The following notes cite one example of metallurgical, rolling, drawing and coating practice. Heats must be kept separate, from casting of the ingots to the finished wire. The steel should be made in heats of about 150 tons or less, cast into hot-top ingot molds, and held till completely solidified before stripping. The ingots should be soaked to a uniform temperature throughout, and rolled on the blooming mill at a temperature of 2200° F, with reduction in each pass limited to avoid cracks and other rolling defects. The crop from the top of the bloom should not be less than 12 per cent, and billets (usually 2 in. by 2 in. by 30 ft.) should be carefully inspected on the ends for pipe and segregation and on the surface for other defects and chipped and surfaced to remove seams, laps, slivers, etc. The billets should be rolled at temper-

atures between 1900° F and 2000° F to a rod of specified diameter with a minimum amount of water used on the rolls and none on the reels, to avoid hard spots on the rod. The rod should be carefully inspected before it is treated. For the latter operation, the rod is uncoiled continuously and passed through a 50-foot or longer continuous furnace held at 1625° F to 1675° F to heat the rod above the critical temperature of 1325° F, after which it is passed into an 18-foot lead pan held between 900° F and 950° F, the exact temperature to give the desired structure being determined by preliminary tests on each heat. Lead adhering to the rod must be removed, a result usually accomplished by a header of coke breeze on the exit end of the lead bath.

To clean the rod, it is pickled in a bath, the acidity of which is maintained between 5 and 8 per cent with 66° Baumé sulphuric acid, and the temperature of which is held at not over 110° F to avoid acid embrittlement that cannot be removed by the subsequent baking. A suitable inhibitor is used to reduce the amount of acid consumed, prevent overpickling, or pitting, and help avoid acid embrittlement.

The rod is then washed with water, and sprayed 30 or 40 minutes to obtain a complete silt coat; submerged repeatedly in a water suspension of slaked lime; and baked 10 to 12 hours at 400° F to 450° F.

The rod is then drawn on a specially-built 6-draft continuous machine or on powerful 30-inch diameter blocks at a speed of 16.5 r.p.m. or 130 feet per minute, using powdered soap mixed with grit-free hydrated lime as a lubricant and uniform light-reduction drafts of 0.384, 0.334, 0.290, 0.252, 0.219, and 0.192. Each wire should be inspected front and back and tested. Requirements are: minimum tensile strength, 240,000 lb. per sq. in., 40 per cent reduction of area; and 2 per cent elongation in 10 inches. It has been demonstrated that wire cold drawn to finish in this way is far superior for bridges than a heat-treated wire of about the same strength.

Galvanizing operations are continuous, the wire passing through the following baths: (1) molten lead at 850° F, momentarily, to remove lubricants, (2) a cleaning box containing a 4 to 6 per cent solution of hydrochloric acid at 150° F to 170° F, (3) a hot water wash at 200° F, (4) a flux box of zinc chloride (ZnCl₂) or 1 per cent hydrochloric acid at 150° F to 175° F, (5) the galvanizing tank filled with especially pure zinc at 880° F to 900° F, equipped with an exit sinker of the roller type and a charcoal header filled with a mixture of charcoal and beef tallow.

Bridge Strands—For cable suspension bridges where the size of the main cables does not justify the stringing of individual wires, parallel bridge strands are used to form the main cables. These strands are compacted in the same manner as are the main cables composed of individual, parallel bridge wires.

Chapter 41

MANUFACTURE OF STEEL TUBULAR PRODUCTS

SECTION 1

HISTORY AND CLASSIFICATION OF STEEL TUBULAR PRODUCTS

Events Leading to Development of the Butt-Weld Process—About the year 1815, William Murdock, a Scottish inventor, introduced at London the use of coal gas for lighting purposes. For conveying this gas, Murdock collected old musket barrels and screwed them together to form continuous tubes. The popularity of this lighting system created a demand for tubes, and stimulated inventors to seek some means of producing the tubes more rapidly and at a lower cost. The first to succeed in this undertaking was James Russell, who filed patent papers describing his process as "an improvement in the manufacture of tubes for gas and other purposes" in 1824. In his method, the tube was formed by butting the white-hot edges of a bent plate together. The initial welding was done with a tilt-hammer provided with round grooves in the head, and the rough tube thus formed was finished by reheating it and passing it through a round groove in a rolling mill and over a mandrel which was supported in the pass, or opening, between these rolls. The next year, however, Russell's work was overshadowed by the invention of Cornelius Whitehouse, who succeeded in forming a commercially perfect tube by merely drawing the flat plate, heated to a proper temperature, through a "bell" or die. This invention, which became the basis of one of the present-day butt-weld processes, made it possible to produce tubes of superior quality much more cheaply than before. Shortly after this invention, about 1832, the first shop for making butt-weld pipe in the United States was established in Philadelphia by Morris, Tasker and Morris. Four years later, this firm built the works afterwards known as the Pascal Iron Works. Following their success, other plants appeared in Eastern Pennsylvania, Eastern New York, New Jersey and Massachusetts, but no plants were built west of the Allegheny Mountains until after 1860. The idea that pipe could be butt-welded continuously was conceived by John Moon in 1911. Later, with S. F. Fretz, Jr., he built the first experimental mill. This equipment proved successful and in 1921 and 1922 the Fretz-Moon Tube Company was formed and continuously butt-welded pipe was made on a production basis.

First Seamless Tubes—Following these developments in making welded tubes, inventors turned their attention to the production of seamless tubes. At first they attempted to duplicate with iron or steel the method (extrusion) used to produce tubes of lead and other ductile metals, namely, forcing the hot metal through an orifice formed by a mandrel or punch located and supported in the center of a circular die. In 1836 such a process was patented by Hanson in England, but this method proved impracticable. Two other methods were brought out in 1840 and 1845. They involved the cupping of a plate or the piercing of a round billet in a press, and subsequently elongating the rough tube thus formed by drawing or rolling. While practicable, these methods were costly,

and for fifty years the use of seamless tubes was restricted on that account. The modern developments beginning about 1890 are described in Section 5.

Innovations by Briggs and Riverside Iron—Besides these developments in methods of manufacture, two other events should be mentioned because of their far-reaching effects on the industry as a whole. About 1862 Robert Briggs, then superintendent of the Pascal Iron Works, formulated the dimensions of pipe (tube) threads, and compiled a table giving the nominal sizes, the exact diameters and the number of threads per inch for all sizes of pipe and tubes up to 10 inches. These formulae and tables were subsequently adopted as standard for the manufacture of all tubes and pipes up to 15 inches in size and are widely known as "Briggs Standards," but the name now officially adopted is American Standard (A.S.A.) Pipe Thread. In 1887 the Riverside Iron Works, Wheeling, W. Va., began making butt- and lap-welded pipe of soft Bessemer steel. Up to that time, wrought iron had been the only material used for welded pipe and tubes. Riverside proved that steel was not only equal to wrought iron for this purpose, but actually superior to it in many respects.

The Present Importance of the Steel Tubular Industry and of steel tubular products is apparent to all observers. This growth was due to many things, chief of which is the diversity of uses to which steel tubular products may be and are applied. The mere mentioning of the names, oil, gas, air, water, plumbing, heating, ammonia, dry-kiln, greenhouse, boiler tubes and bedstead tubing are sufficient to call to mind industries almost wholly dependent upon steel pipe or tubing. In addition to these uses, we find it in service as trolley poles, flag poles, lighting poles, telephone and telegraph poles, columns, conduits, automobile axle housings, airplane fuselage, motor mounts, and many other items too numerous to list here.

Classification of Steel Tubular Products—The many uses to which steel tubular products have been applied have led to a great variety of products and to the use of a large number of more or less descriptive terms in designating the products used for different purposes. Use, therefore, may form a basis for classifying steel tubular products, as shown in Table 41—I.

Another classification is based on methods of manufacture. On this basis all steel tubular products may be classified under the two main headings of welded and seamless, with subclasses, as shown in Table 41—II.

Modern Methods of Manufacturing Welded Tubular Products—A butt-weld pipe or tube is made from a hot-rolled strip, with square or slightly beveled edges, called skelp, the width of which corresponds to the circumference of the pipe, and the gage to the thickness of the wall. By heating this skelp to the welding temperature and drawing it through a suitable die or

Table 41—I. Steel Tubular Products Classified According to Use

1. Standard Pipe.
 - a. Standard Weight Pipe (Black and Galvanized)
 - b. Extra Strong Pipe (Black and Galvanized)
 - c. Double Extra Strong Pipe (Black and Galvanized)
2. Bedstead Tubing (Special Light Weight Pipe).
3. Conduit Pipe.
4. Drive Pipe.
5. Dry-Kiln Pipe.
6. Refrigeration and Ammonia Pipe.
7. Large-Diameter Pipe.
 - a. Water Mains
 - b. Gas Lines
 - c. Penstocks
8. Line Pipe.
 - a. Gas Lines
 - b. Oil Lines
 - c. Water Transmission Lines
 - d. Slurry Lines
9. Nipple Pipe.
10. Tubular Piling.
11. Tubular Poles.
 - a. Line Poles
 - b. Signal and Safety Poles
 - c. Flag Poles
 - d. Masts and Booms
12. Rolls and Roller Piping.
 - a. Paper- and Textile-Mill Rolls
 - b. Printing-Press Rolls
 - c. Conveyor Rolls
13. Pressure Piping.
 - a. High-Pressure Steam Lines
 - b. Hot-Oil Lines
 - c. High-Pressure Chemical Lines
14. Signal Pipe.
15. Structural Pipe.
 - a. Railings
 - b. Fence Posts
 - c. Scaffolds
 - d. Columns
 - e. Bridge and Roof Trusses
16. Water-Well Pipe.
 - a. Water-Well Casing
 - b. Water-Well Pump Pipe
 - c. Water-Well Drive Pipe
 - d. Water-Well Reamed and Drifted Pipe
 - e. Driven-Well Pipe
17. Oil-Country Tubular Goods.
 - a. Oil-Well Casing
 - b. Oil-Well Tubing
 - c. Drill Pipe
 - d. Drive Pipe
18. Pressure Tubes.
 - a. Boiler Tubes
 - b. Superheater Tubes
 - c. Economizer Tubes
 - d. Water-Wall Tubes
 - e. Locomotive-Boiler and Superheater Tubes
 - f. Arch Tubes
 - g. Air-Heater Tubes
 - h. Soot-Blower Tubes
19. Oil-Still Tubes.
20. Heat-Exchanger and Condenser Tubes.
 - a. Galvanized Tubes
21. Mechanical Tubes.
 - a. Airplane Tubes
 - b. Automotive Tubes
 - c. Precision Pump Tubes
 - d. Working Barrels
 - e. Stainless-Steel Tubes (Food and Dairy Industries)
 - f. Ball- and Roller-Bearing Race Tubes
 - g. Rolls
 - h. Shafting
 - i. Bushings
 - j. Special Shapes for Structural Purposes
 - k. Upset, Swaged and Special Formed Tubes
 - l. Axle Tubing
22. Stainless-Steel Pipe and Tubes.
 - a. Ornamental
 - b. Mechanical
 - c. Pressure

Table 41—II. Steel Tubular Products Classified According to Methods of Manufacture

A. Welding Processes.

- I. Butt-Weld Process. This process is used in the manufacture of pipe, $\frac{1}{8}$ inch to 4 inches, nominal diameter.
- II. Lap-Weld Process. This process is used to produce pipe and tubes, $\frac{1}{4}$ inch to 30 inches, nominal diameter.
- III. Electric-Weld Process. This process is employed to produce pipe and containers in practically all sizes.

B. Seamless Processes.

- I. Piercing Processes—Roll-Piercing and Disc-Piercing. These processes are used to make pipe and tubes up to 26 inches in outside diameter.
 - a. Hot Finished
($1\frac{1}{2}$ inches to 26 inches, outside diameter)
 - b. Cold Drawn
(Any size up to 10 $\frac{3}{4}$ inches outside diameter)
- II. Cupping Process. This process is applied in the manufacture of tubes and cylinders 3 inches to 20 inches outside diameter.
 - a. Hot Drawn (Finish)
(3 inches to 20 inches)

roll pass while at this temperature, the skelp is bent into cylindrical shape and its edges pressed firmly together into a **butt-weld** or **jump-weld**, thus forming a pipe. Recent innovations which have improved this weld are described in Section 2.

In the **electric-weld** process, hot-rolled strip or plate, of a gage corresponding to the thickness of the wall of the pipe desired, but of an overall width slightly greater than its circumference, is first edge-trimmed to insure parallelism and accurate width. To produce **fusion-welded pipe**, the plate is then bent into cylindrical shape with the beveled edges abutting to form a "V" into which the electrode is melted. Fusion welding, which is particularly applicable to large-diameter pipe, is a term used to distinguish this method from electric-resistance welding used in the manufacture of smaller sizes of tubing. By the latter process union of the seam is effected by the application of pressure and heat, the heat being generated by the resistance to current flow (either transformed or induced) across the seam during the welding.

Seamless Tubular Products are made by two basic processes, known as the **piercing** and the **cupping** processes. The piercing process, in which a solid round bar or billet is heated, pierced and afterwards wrought to the desired diameter and thickness of wall, is employed generally on sizes up to 26 inches, outside diameter,

while the cupping process, in which the tubing is formed by pressing a preheated solid round plate through cup-shaped dies, may be employed for all sizes between three and twenty inches. The reasons for these restrictions in the use of the five methods cited will be made apparent in the detailed descriptions that are reserved for succeeding sections.

Pipe—Applied in a general sense, pipe is a term used to designate any long hollow body used for conducting gases or liquids, and may be of clay, cement, wood, lead, brass, cast iron, or steel. Restricted to the steel industry, the term is one that is applied to all tubular products intended for the purposes for which such products are ordinarily used, as for conducting water, fuel, gas, steam, air, oil, etc. The term "wrought" distinguishes forged iron or steel pipe from cast-iron pipe.

Varieties of Pipe—Since the pipe should be adapted to the kind of service it is expected to give, there are different varieties of pipe. Thus, welded pipe is regularly made in three separate classes, which are distinguishable by characteristic differences in the wall thickness of each class of pipe as a class. These classes of pipe are known as **standard weight**, **extra strong**, and **double extra strong**. **Standard weight pipe** is pipe made and threaded to the American Standard (A.S.A.). The weights or wall thicknesses of this pipe are suitable for all ordinary purposes, such as water and gas lines for plumbing and heating, etc. **Extra strong pipe** is characterized by heavier walls in all sizes than the standard weight pipe, and is used where higher internal pressures or greater column loads are to be sustained. **Double extra strong pipe** is used for extremely high-pressure purposes, such as hydraulic lines for operating hydraulic machinery or apparatus. **Line pipe**, used for the higher pressure oil, gas and water lines, and **drive pipe**, used in drilling oil

and water wells, differ from each other and from standard pipe mainly in the manner in which the different sections of pipe are coupled together. **Air-line pipe**, used for steam train and electric car air-brakes, and **drill pipe**, used for oil-, gas- and water-well drilling, are further examples of the different varieties of pipe. In addition to these varieties, pipe is also made to conform to the British Standards. Pipe may be used for many structural purposes, such as columns, railings, piling and poles for telegraph, telephone and electric power lines, also railway signal and safety devices.

Sizes of Pipe—As a whole, pipe is made in sizes that range from $\frac{1}{8}$ -inch nominal inside diameter to 96 inches outside diameter. Up to a diameter of 12 inches, pipe is commonly known by the inside diameters, and is spoken of as three-quarter-inch pipe, one-inch pipe, two-inch pipe, etc. These terms are but nominal, because, in order that different pieces of pipe may be coupled together conveniently, the outside diameters must be standardized, and any variation due to differences in gage, or thickness of the wall, occurs in the bore. However, care is taken to make these differences for any one kind of pipe as small as possible, so that in the case of standard pipe these nominal diameters approximate as closely as possible the inside diameters. Exceptions may be cited in the case of 8-inch, 10-inch, and 12-inch standard line pipe, which are made in different weights to adapt the pipe to different purposes. For example, 8-inch standard line pipe is made in two standard weights, namely 24.696 pounds per foot and 28.554 pounds per foot, the difference in weight being represented by a difference in the wall thicknesses. Standard line pipe, drive pipe and pipe over 12 inches internal diameter, and practically all pipe over $12\frac{3}{4}$ inches in diameter, are known by their outside diameters

SECTION 2

BUTT-WELDED PIPE

The National Tube Division of United States Steel manufactures butt-welded pipe by the furnace-weld and continuous butt-weld methods. Following is a description of these two processes.

BUTT-WELD PROCESS (FURNACE-WELD)

Rolling the Skelp—Skelp for butt-welded pipe may be rolled on any mill adapted to the rolling of long thin flats. For the smaller sizes of pipe, $\frac{1}{8}$ inch to $\frac{1}{2}$ inch, the skelp sizes will range from about $1\frac{1}{8}$ inch to $2\frac{3}{4}$ inches in width and from 0.068 inch to 0.109 inch in thickness. For the larger sizes of butt-welded pipe, the skelp will range in width from $3\frac{1}{8}$ inches to 12 inches, and in gage from 0.113 inch to 0.216 inch for the standard, or from 0.154 inch to 0.300 inch for extra strong pipe. While it is evident that this material may be rolled by tongue and groove passes and plain rolls, much in the same manner that sheet bar is rolled, on many different types of mill, yet, on account of the introduction of special features aiming at improvements in the quality of pipe, skelp is best rolled on a mill specially designed for the purpose.

Example of a Skelp Mill—As an example of a modern skelp mill, the No. 5, 13-inch mill at the Lorain Works of National Tube Division may be cited. This mill consists of thirteen stands of horizontal rolls and one stand of vertical rolls arranged on the continuous plan, but the roll stands are so placed that the piece will clear the first, second, fifth, seventh, and eleventh passes before it enters the next succeeding passes, respectively. In order to secure this arrangement, the first ten stands

are set up in four tandem groups composed of two, three, two, and three stands, in the order named. The stands in each group are geared together and driven through one shaft. The eleventh stand, in which the rolls are vertical, and the twelfth, thirteenth and fourteenth stands, which are horizontal rolls, are separately driven. The mill is served by two continuous heating furnaces and is provided with a hot shear between the fifth and sixth passes, that is, between the second and third groups of rolls. Its product is received on a two-way cooling bed. An end-cutting shear is located at each of the two delivery corners of the cooling bed, and between these two shears are two other shears, known as the clipping shears.

Operation of the Mill—The bloom, which is rectangular in shape and of the proper dimensions to produce skelp of the desired size, is heated in the continuous furnace to a rolling heat. The first stand of rolls is placed far enough from the furnace to permit the switching of the bloom to the mill floor in case there should be a rolling accident on the mill or the bloom should be overheated or underheated. The first two or four stands in the mill, corresponding to roughing rolls in other mills, are used to spellerize the metal. In the usual manner of carrying out the spellerizing process, the working surfaces of the first and second stands, called **knobbling rolls**, are provided with regularly shaped projections and depressions, while the surfaces of the other passes are plain. When the bloom passes through these rolls, the roll-knobbling, or kneading, process to which its sur-

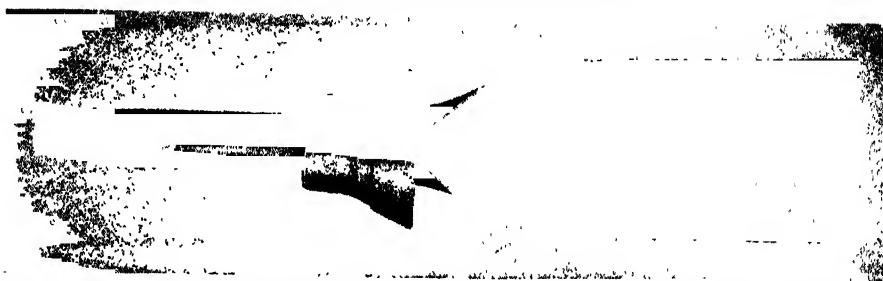


FIG. 41—1. Showing how the skelp is bent and welded as it is drawn through the welding bell.

faces are subjected is said to give a pipe surface that is better adapted to resist corrosion in the form known as pitting than that of skelp rolled on plain rolls only. Upon leaving the third set of rolls, the bloom, now much reduced in thickness, is edged by a twisting guide in the second group of rolls, in order to control the width. Next, the bloom is cut into two or more pieces by the hot shear, to adjust the length of the finished strip to the cooling bed. These pieces then pass flat-wise, or flat-and-edgewise, if desired, through the next five stands to the vertical rolls. These rolls are grooved to roll the edges of the skelp, which must be slightly beveled, so that, when the skelp is bent to form the pipe, they will fit squarely together and form a perfect joint. In this way, the defects in pipe, known as a *falling*, or *sunken*, seam, a depression along the weld on the outside of the pipe, and a *deep seam*, a depression on both the outside and inside, are avoided, for both may be due to round edges on the skelp. To secure the required edge on the finished skelp, the bottom of the grooves in the vertical rolls are made slightly convex, thus forming concave edges on the skelp, so that the spreading of the steel in the succeeding passes through the three stands of plain rolls will be just sufficient to fill the concave surface.

Shearing the Skelp—From the finishing rolls of the mill, the skelp passes over the cooling bed to the shears, where the crop ends are cut off, and the remainder of the strip is cut into the lengths desired, the standard lengths being approximately twenty-one feet for single lengths and forty feet for double lengths. The skelp is then passed to the *clipper*, a machine that performs the

double function of a shear and a press. It (1) shears two small triangular shaped pieces from one end of the skelp, one from each corner, (2) shapes, or cups, the pointed end to start the curve for welding, and (3) bends the skelp transversely just back of the curve, so that when the skelp lies on the flat surface of the welding furnace hearth the pointed end will be elevated, thus making it easier to grasp with the welding tongs. Care is required in adjusting the end of each strip in this machine, so that the clipping and cupping will be symmetrical, as otherwise the skelp will be liable to stick in the welding bell, which will be described later. The skelp is now stacked in a small pile, loaded on railroad cars with a crane, and then taken to the butt-welding furnaces.

The Butt-Welding Furnace—In the pipe mill, the skelp is charged directly into a reheating furnace preparatory to welding. To obtain uniform heating of the material (a condition absolutely necessary to effect good welding) and economize in time and labor (factors that vitally affect the cost of pipe) the heating furnace, charging apparatus, welding appliances, and finishing machines must be of special design, and constitute a single unit. The heating furnace is the central figure in this unit. These furnaces are of the gas-fired regenerative type, and, depending upon whether they are designed for single or double lengths, are 22 or 40½ feet long inside, but vary in width to suit the different sizes of pipe, in order to accommodate a certain number of skelp strips, which lie side by side upon the hearth, or bottom, of the furnace. Special care must be taken

FIG. 41—2. Butt-welding furnace, showing a pipe in the process of being welded.



in making up the hearths of these furnaces to prevent the material of the bottom from sticking to the hot skelp and thus ruining the pipe. The materials used for this purpose are high-grade sand and gravel. The gas and air for combustion are admitted through ports arranged along the sides of the furnace, so that the flame must pass transversely over the hearth, which is covered with a low, arched roof. The ends of the furnaces are constructed so that only narrow openings, one extending across each end just above the level of the floor, remain to provide an inlet and outlet for charging and drawing the skelp and repairing the hearth. When the furnace is in operation, these openings are partially closed by adjustable shields suspended by proper supports from above. Opposite the inlet is placed the charging machine, while the welding machine is opposite the outlet. Both machines are mounted on wheels. Those under the charging machine run on tracks laid across in front of the furnace, and the machine is propelled by electricity back and forth as desired. Usually, only the end of the welding machine next to the furnace is movable, and it is moved back and forth by a steam or air cylinder. The charging machine for the larger sizes of pipe is provided with a platform for the operator to stand upon, a table for supporting a pile of skelp at a convenient level, and, alongside, an electrically-operated endless chain pusher or magnetic rollers for charging the skelp into the furnace.

The Welding Machine very much resembles a draw bench, with the end next to the furnace mounted on wheels and the farther end pivoted. It consists of a carriage, on which is mounted an endless chain for drawing the skelp from the furnace through the welding die, and a stop for holding the welding die as the skelp is drawn through it. The draw chain is actuated by a motor-driven sprocket wheel on the pivoted end. The

welding die, called a welding bell on account of its form, is so shaped inside that, as the skelp is pulled through it, the strip is gradually bent into the form of a tube, and its edges are forced together, thus forming the weld. The bells are made of a special grade of copper alloy.

Operation of the Butt-Welding Furnace—The skelp from the rolling mills, in either single or double lengths, depending upon the size of the furnace, and of a width and gage to make the pipe desired, is unloaded from the flat car, bundle by bundle, and placed upon the charging machine by an overhead crane. Starting at one side of the furnace, the skelp is slid by hand from the pile to the charging table with the clipped ends foremost, one strip at a time, and charged into the furnace by the charging machine with the convex side of the clipped end up. After charging each strip, the machine is moved over a distance corresponding to the width of the strip before pushing in another. In this way, the bottom of the furnace is soon covered with the strips lying side by side, and the charging machine is ready to repeat the cycle. Since the size of the furnace and the rate of charging are so regulated that, when the last strip has been charged, the first one charged will have reached the welding temperature of about 2600° F, the welder now begins drawing the skelp in the same order it was charged. This he does with specially-designed tongs, having short jaws for grasping the pointed end of the skelp and very long handles with a locking bend in the middle and a knob on the end for engaging with the lugs, or the dog, on the draw-chain. No section of these tongs can be larger than the outside diameter of the pipe, as they must pass through the welding bell. Preparatory to drawing each strip of skelp, the welder grasps the pointed end of the skelp with the jaws of the tongs. He then slips, large end first, a welding bell of the proper size over the handles of the tongs and drops the handles

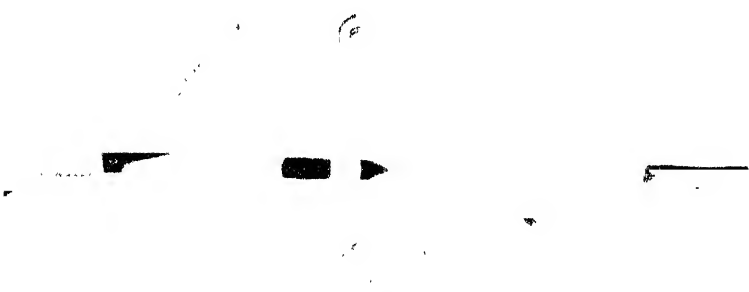


FIG. 41—3. Pipe passing through the sizing rolls.

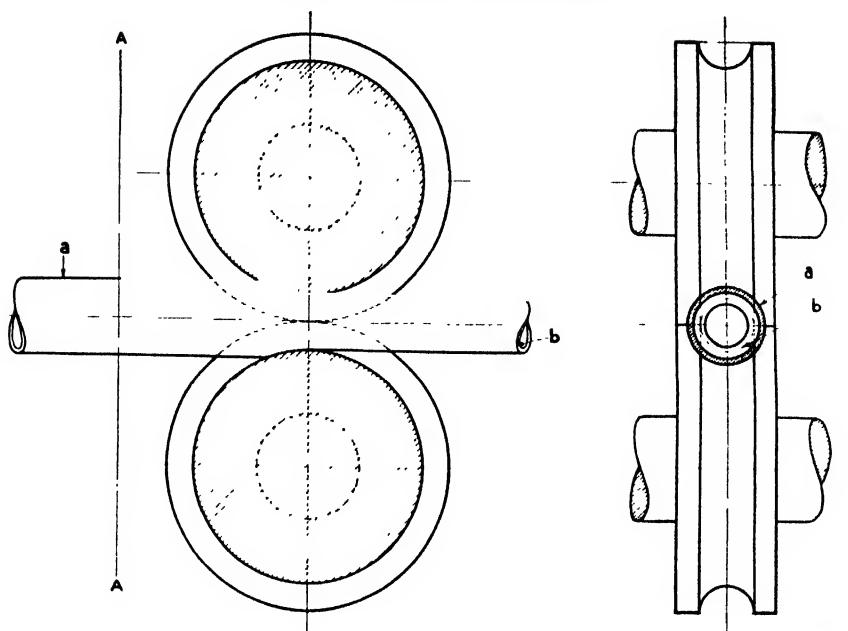


FIG. 41-4. Sizing rolls. Illustrating how the size of the pipe is reduced to the diameter desired.

between the two prongs of the bell stop in such a manner that the dog on the draw-chain will engage the knob on the handle. Then, the chain, traveling continuously away from the furnace, draws the tongs, bell and skelp out of the furnace. When the bell reaches the stop, it is held there until the skelp has been drawn through for its full length, when the bell drops from the stop by its own weight. The flat skelp is thus bent into the form of a pipe, and its edges are forced together and welded, all in the one operation. In this way the welding requires but little time: one-inch pipe, for example, may be welded at the rate of 750 twenty-one-foot lengths per hour. For the larger sizes of standard pipe, two welding bells, instead of one, are employed.

Although the metallurgical features and mechanical principles used in manufacturing butt-weld pipe from skelp have remained the same for over fifty years, there have been numerous slight changes and innovations introduced with the idea of improving the weld made. Since the temperature of the skelp edges to be welded must be very closely controlled, various means have been developed for regulating this temperature after the skelp leaves the furnace and before it enters the welding bell. The use of a strong air blast on the edges of the skelp just before it enters the bell was practiced for many years and is still in use. An improvement on the air blast resulted from the application of an air blast enriched with oxygen which, impinging on the edges of the skelp, raises the surface temperature and acts more effectively in maintaining the skelp edges to be butted together at the proper welding heat. This process is still used in many plants. A further application of this same principle consists in applying an oxy-acetylene jet to the skelp edges, and this method of local heating has two distinct advantages. In the first place, the temperature of the skelp edge can be better controlled by the adjustment of the oxy-acetylene flame, making it possible to maintain a definite welding temperature which will produce a uniform and superior weld. The second advantage of the oxy-acetylene gas flame is that it may be made reducing and thus decrease the formation of an oxide film on the skelp edges just prior to welding.

While the control of all these adjuncts must be very precise, equipment and adjustments have been gradually developed, which have led to improvement in the quality of weld made. In fact, pipe now being made with the oxy-acetylene gas-welding practice has produced welds which may be justly placed in the fusion class, enabling the use of welded pipe in applications to which the ordinary furnace-welded pipe is not adaptable.

The Hot Finishing of Pipe—The Sizing Rolls—When the pipe has been drawn through the welding die, the tongs are removed, and the pipe at once moves down a slightly-inclined table in front of the sizing rolls. These are electrically driven horizontal rolls of cast alloy iron, 18 inches to 24 inches in diameter, depending upon the size of the pipe to be made. They are grooved to form a perfect round, when properly adjusted. The rolls contain usually but a single groove, hence are very short and are supported in a single two-high housing. In passing through these rolls the pipe, which was purposely drawn slightly oversize, is compressed and elongated to an exact uniform outside diameter, and the weld is also improved to some extent thereby.

The Stretch-Reducing Mill—As the name implies, this mill is used for reducing the diameter of and simultaneously applying tension to the tube being produced. By this method and without the use of supporting mandrels, the wall thickness of a tube can be maintained or even decreased while the diameter is being reduced. The finished tube is considerably lengthened by this process. It can be used in the manufacture of both welded and seamless tubes.

The mill (Figure 41-5) consists of a series of pairs of rolls, mounted in housings which are spaced as close to one another as design considerations will permit. Each consecutive pair of rolls is driven at a speed in excess of that required to deliver an equal volume of metal per unit of time from each roll stand. Tension is thereby developed in the tube between stands, which permits larger diameter reductions than can be obtained without tension and, under certain conditions, a reduction of wall thickness. The close spacing of the housings is necessary to minimize the so-called "end-effect" or

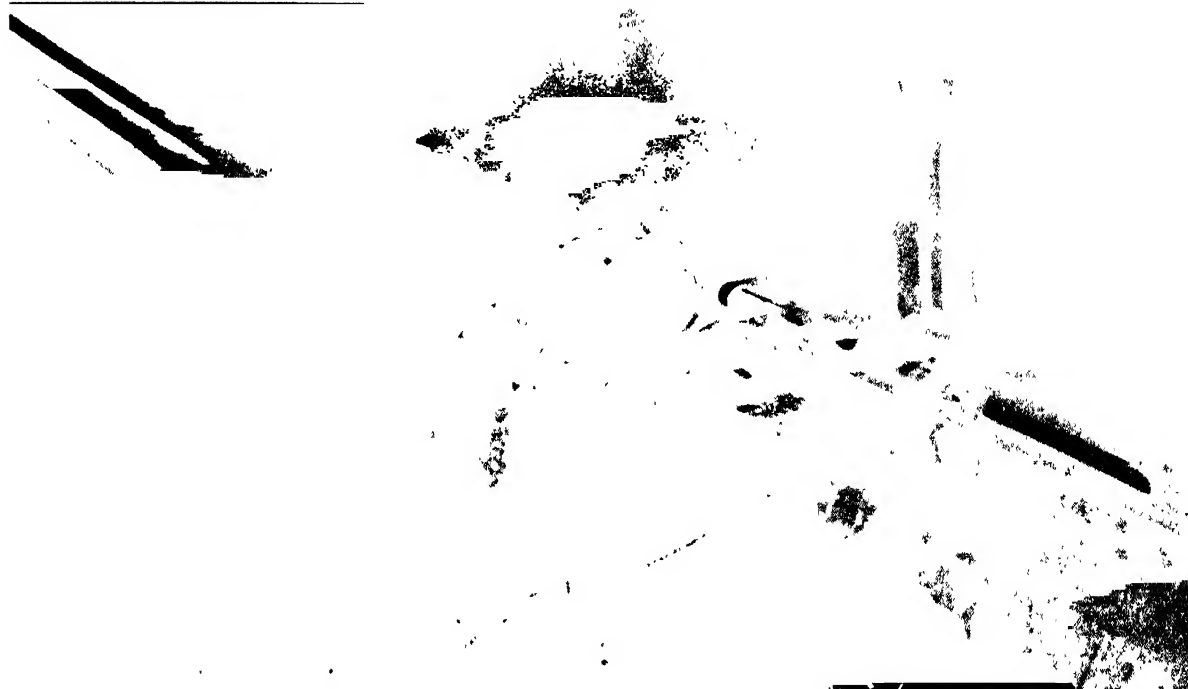


FIG. 41—5. Stretch-reducing mill.

thickening of the wall at both ends of the pipe, where no tension can be applied.

The proper speed relationships are obtained either by selecting suitable change gears on mills driven by one motor or by adjusting the motor speeds on mills having individually driven housings.

The number of pairs of rolls used ranges from eight to sixteen, varying with the design and purpose of the mill.

Straightening Rolls—From the stretch-reducing mill, the pipe is transferred over a cooling table and allowed to drop into a rack. In the case of double-length furnaces, the pipe is middle-cut by a hot-saw while on the cooling table, before dropping into the racks. After being quenched in a water bosh to complete the cooling, the

pipe is then transferred to the straightening machines.

Two types of straightening machines are used. One type, known as "cross-rolls," consists of two electrically driven rolls, from two to four feet long and from ten to twenty inches in diameter, depending upon the size of the pipe to be straightened, set with their axes askew so that they cross each other to form an angle of about 30° , their center points being directly opposite each other. The body of these rolls is machined with a concave surface so that a pipe lying between them will be in contact with the rolls for their entire length. The end of a pipe that is to be straightened is inserted between the rapidly-revolving rolls which move it forward between the rolls, as it rotates, due to the axes of the rolls



FIG. 41—6. Pipe passing to cooling bed. This bed is designed to cause the pipe to cool practically straight.

TYPICAL CONTINUOUS WELD PIPE MILL

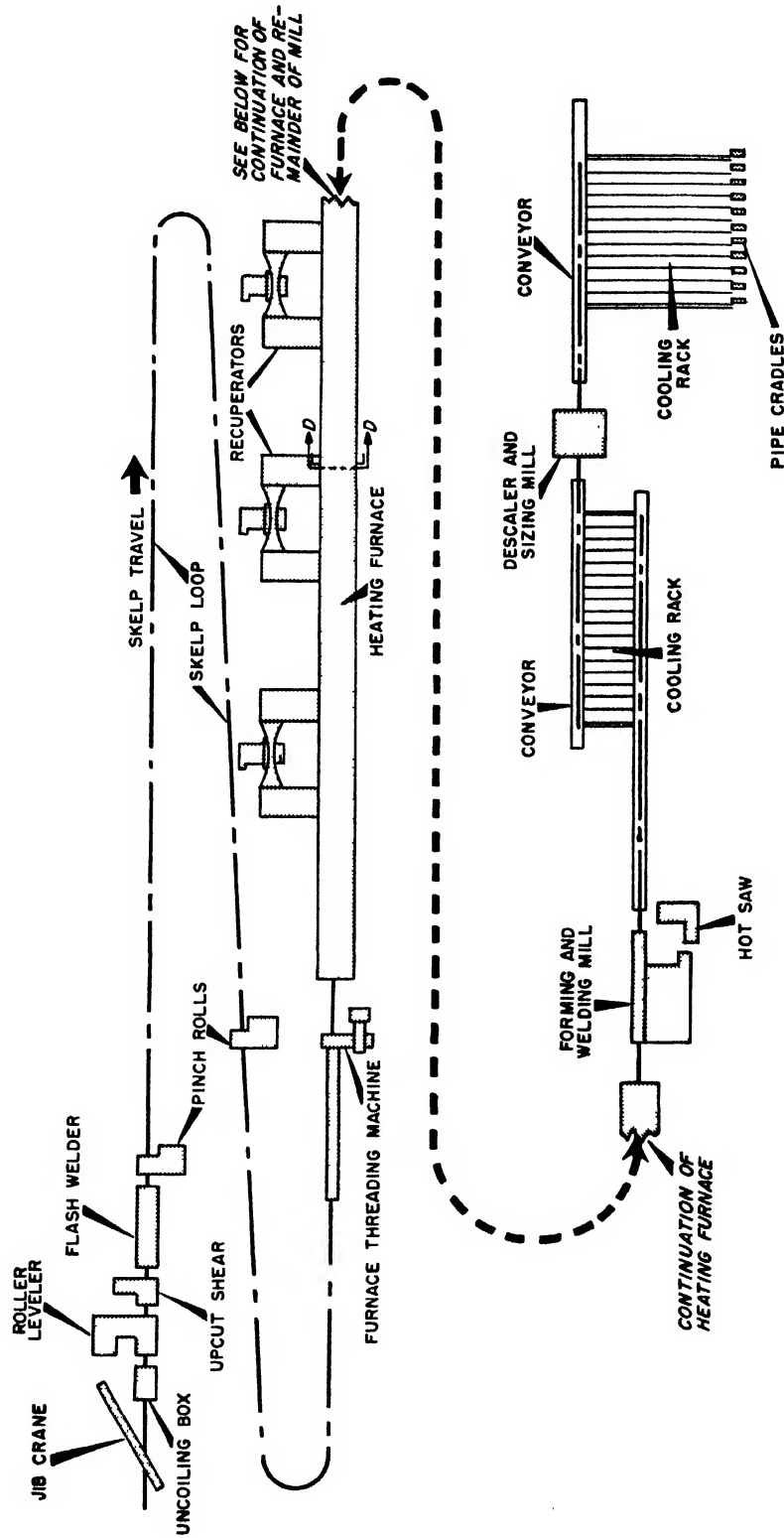


FIG. 41-7. Schematic plan of the layout of a typical continuous butt-weld pipe mill. Details of the uncoiling box are shown schematically in Figure 41-8. Figure 41-9 illustrates diagrammatically the steps performed in the operation of the flash welder. The furnace threading machine is shown schematically in Figure 41-10. Section D-D of the heating furnace is given in Figure 41-11. Operation of various parts of the forming and welding mill is explained in Figure 41-12.

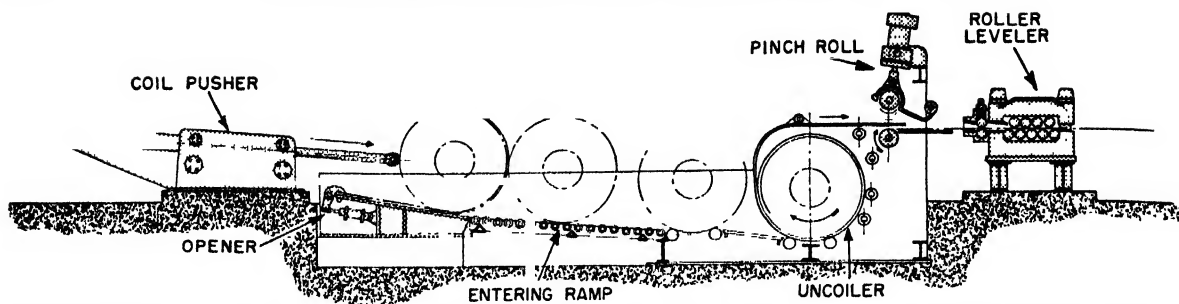


FIG. 41-8. Schematic elevation of coiled-skelp handling equipment at the entry end of a continuous butt-weld pipe mill.

being askew. When the straightened pipe emerges from the straightener, it moves forward a short distance and is discharged laterally into a cradle or rack. From the cradle, the pipe is transferred by an overhead crane to the finishing equipment. The other type of straightening machine, known as the "rotary" straightener will be discussed under the seamless process.

BUTT-WELD PROCESS (CONTINUOUS)

The continuous butt-weld mill is a true continuous process starting with the coiled skelp and ending with finished pipe. Figure 41-7 shows an arrangement which is typical of many of the continuous butt-weld mills in use today.

Production of Skelp—Skelp for the continuous butt-weld process may be produced in the same manner as skelp for the conventional butt-weld process described previously. However, instead of shearing the strip into twenty-one and forty-foot lengths, only the leading and trailing ends of each strip are trimmed square and the skelp is coiled on vertical-type reels.

Uncolling the Skelp—Coiled skelp is loaded onto the uncoiler feed table and advanced by the coil pusher to the uncoiler (Figure 41-8). The end of the coil is fed into a pinch roll which is released after the skelp enters the roller leveler. The function of the roller leveler is to straighten or flatten the skelp for further processing. A typical roller leveler consists of two entering pinch rolls and five to nine leveling rolls. As the trailing end of one skelp coil is reached, the leading end of the next coil is being made ready for welding to form a continuous strip. Before the welding can be undertaken, both ends of the skelp must be trimmed square and true. If this has not been done at the skelp mill, or if the ends have become damaged, they are cut on an upcut shear which is provided ahead of the end welder (Figure 41-7). The shear is provided with incoming and outgoing guides to align the skelp properly for cutting. When the ends have been properly prepared, they are advanced through the end welder.

Welding Unit for Joining Ends of Coils—The leading end of the coil is fed into the flash welder where it is welded to the trailing end of the preceding coil to deliver an endless piece of skelp to the mills. The welding unit consists of a flash welder, stripper, pulling and gripping mechanism, side trimmers, skelp-centering device and side guides. The welding cycle for a typical end welder is shown in Figure 41-9. During the welding cycle, the ends of the skelp are, of course, held stationary.

"Looping" the Skelp—Upon completion of the welding cycle, the skelp is guided to the floor (on its edge) and formed into a large skelp loop. This loop provides sufficient skelp storage to assure continuous operation of the mill while the uncoiling unit is stopped and the front end and trailing end of the two coils are welded to-

gether. Four magnetic-roll units are provided to assist the leveler in feeding the skelp into the storage loop.

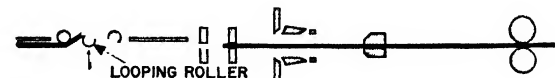
Furnace "Threading"—Vertical pinch rolls located near the end of the loop pull the skelp out of the storage loop and feed it into the furnace. The speed of the pinch roll is varied by an automatic loop control located on the back loop approximately 70 feet from the furnace. The skelp, which has been traveling on edge, again assumes a horizontal position and is fed up an inclined roller ramp to the furnace. The furnace threading machine (Figure 41-10) has driven pinch rolls, one convex and one concave, to give the skelp a dished shape to provide sufficient stiffness to permit it to be pushed through the



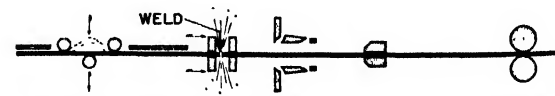
A. TRAILING END OF SKELP STRIP IS HELD STATIONARY BY HOLDING JAWS, WHILE LEADING END OF INCOMING STRIP IS BROUGHT AGAINST GAGE BAR.



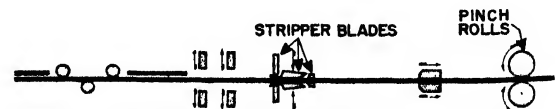
B. WELDING AND CLAMPING ELECTRODES GRIP BOTH ENDS OF SKELP GAGE BAR IS REMOVED.



C. LOOPING ROLLER MOVES UP TO ALLOW SLACK IN SKELP FOR NEXT OPERATION.



D. POWER IS APPLIED TO ELECTRODES. LEFT HAND ELECTRODES MOVE TO RIGHT BRINGING ELECTRIFIED ENDS TOGETHER FORMING ARC, UPSETTING AND WELDING.



E. WELDING AND CLAMPING ELECTRODES RELEASE. SKELP IS PULLED THROUGH FLASH STRIPPER. STRIPPER JAWS RELEASE AND PINCH ROLLS PULL WELDED AND CLEANED SKELP TO CONTINUOUS LINE.

FIG. 41-9. Welding cycle for a typical flash-type skelp

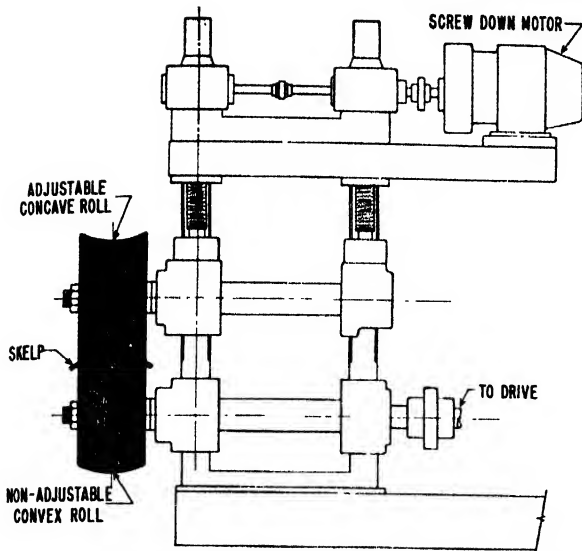


FIG. 41-10. Schematic elevation of a furnace threading machine.

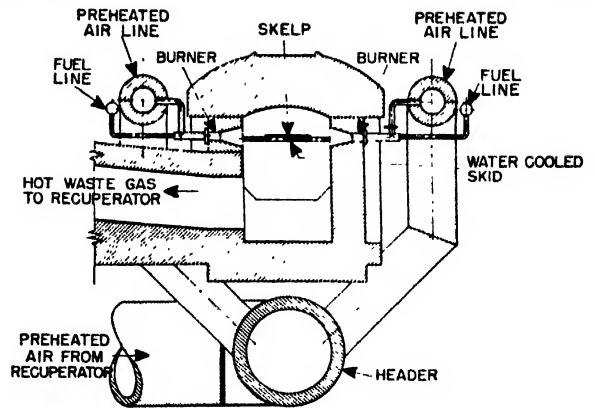
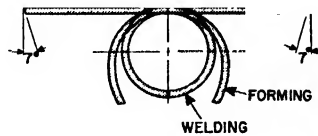
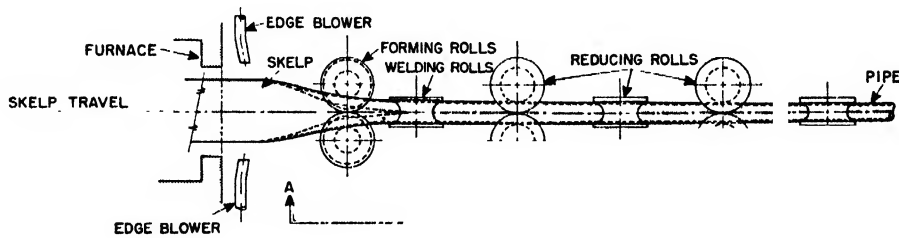


FIG. 41-11. Cross-sectional diagram showing design of a furnace used on a continuous butt-weld pipe mill for heating the edges of the skelp prior to welding. (This sketch corresponds to Section D-D of Figure 41-7.)



SKELP AT VARIOUS STAGES IN THE FORMATION OF PIPE
(NOTE BEVELED EDGES OF SKELP)

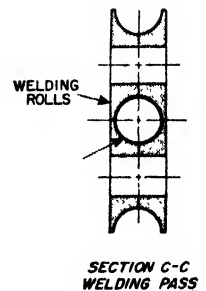
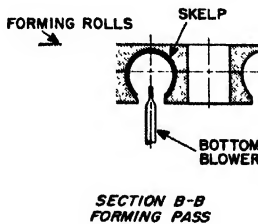
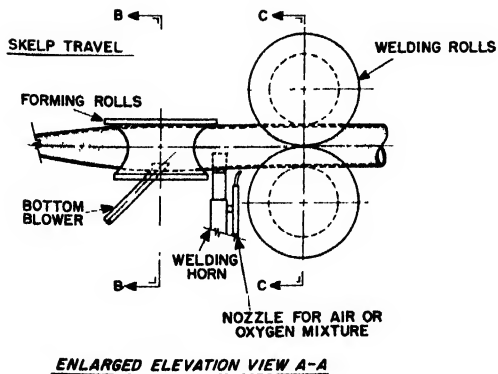


FIG. 41-12. Diagram depicting schematically the operations performed in a continuous forming and welding mill.

furnace during the furnace-threading operation. As soon as the skelp is entirely through the furnace and is being pulled by the forming and welding unit, the threading rolls are opened up and become inoperative until the mill must again be threaded.

The Butt-Welding Furnaces—The furnaces used on the continuous weld mills are long and narrow and usually have several firing zones, individually controlled. The furnaces may be gas- or oil-fired by burners set in the side walls and firing directly at the edges of the skelp passing through the furnace as shown in Figure 41—11. The skelp is supported on water-cooled skid bars. To secure a high production rate and good fuel economy, the furnaces usually are equipped with metallic recuperators which heat the combustion air to about 1000° F. The furnace must have sufficient length to heat the skelp passing through it to a temperature of approximately 2450° F. A blower usually is provided to furnish a blast of air on the edges of the skelp as it leaves the furnace. This raises the temperature of the edges to approximately 2600° F for welding.

Forming, Welding and Reducing Mills—Upon leaving the furnace, the skelp passes through the forming rolls which are designed to bend the skelp into an open-sided oval with the opening downward (Figure 41—12). An air- or water-cooled welding horn, which is located directly below the bent skelp and ahead of the welding stand, guides the skelp edges. A nozzle applies a jet of air, or of air-oxygen mixture, onto the skelp edges at this point to maintain welding temperature. The air blast also helps in freeing the edges from scale.

The hot skelp then passes through the second or welding stand, where the edges of the skelp are squeezed together to form the weld. A reduction of 4 to 6 per cent produces a satisfactory weld. The pipe passes directly from the welding rolls to a series of reducing passes where the diameter of the pipe is further reduced. The forming, welding and reducing stands, with individual

motor drives for each stand, consist of alternate vertical and horizontal passes.

Stretch reducing also is employed in the continuous weld mills. This principle was discussed previously under the butt-weld process.

Hot Sawing and Preliminary Cooling—A rotary-type flying hot saw is used for cutting the sized pipe to length. The saw is of the continuous running type with the length of cuts determined by the length and r.p.m. of the crank. The saws will cut lengths from 18 to 54 feet at maximum mill delivery speeds. Individually motor-driven roller conveyors which operate at greater than the mill delivery speeds are provided after each saw to carry the pipe away from the saws to cooling beds. Screw-type kickouts are provided to discharge pipe from the saw delivery conveyor to the screw-type cooling beds where the temperature is lowered enough to permit descaling and further sizing.

Sizing, Cooling and Inspecting—In a typical mill (Figure 41—7), the pipe passes from the first cooling rack onto a conveyor that carries it to the sizing machine. The sizing machine, which consists of three roll stands, performs, as its name implies, the final sizing operation.

The pipe is then taken to a second and much larger cooling rack where its temperature is lowered so that the pipe may be inspected and straightened. If the pipe has been cut to multiple lengths at the flying saw, a parting saw, usually located at the second cooling bed, cuts the pipe to standard lengths. One or more crop saws also located at the cooling bed may be utilized if required. A water bosh sometimes is incorporated in the cooling bed to provide adequate cooling capacity without excessively long cooling beds.

After being inspected, the pipe is taken to the finishing department where it is faced, threaded, tested, coated, bundled, or stenciled, as required. Pipe to be galvanized is straightened and usually faced prior to being galvanized.

SECTION 3

ELECTRIC-RESISTANCE-WELDED TUBING

Small-diameter tubing of light wall thickness, primarily for mechanical and structural purposes, is sometimes fabricated from strip which is formed into shape and welded electrically. Rimmed or killed steel may be used and such tubing is furnished either cold rolled or hot rolled, pickled and oiled according to the desired product.

Steps in the Manufacture of Electric-Resistance-Welded Tubing—The sequence of operations required in the fabrication of electric-resistance-welded tubing are slitting the strip, forming, welding, sizing, cutting and finishing.

Slitting—Strip is received at the pipe mills in coils 36 inches wide weighing up to 14,000 lbs. This wide strip must be slit into the correct width required to form the finished tube size.

The wide coils are loaded onto a ramp from which they are permitted to roll, as required, onto a charging buggy which moves the coils to the pay-off reel of the slitting machine. The buggy is provided with an elevator by which it is possible to center the coil on the mandrel of the reel which is then expanded and the retaining bands on the coil removed. The reel, which is motor driven, is rotated to slowly unwind the strip which is threaded into a set of three breaker rolls which flatten it sufficiently to facilitate threading through pinch rolls into the slitter knives. These knives, which are tool-

steel discs, are mounted on arbors above and below the strip and are spaced with rings to slit to the desired width. From the slitter the strips pass to the recoiler where they are wound between thin, steel disc spacers. The narrow strips of scrap from the edges of the wide strip pass over the recoiler onto the scrap winder.

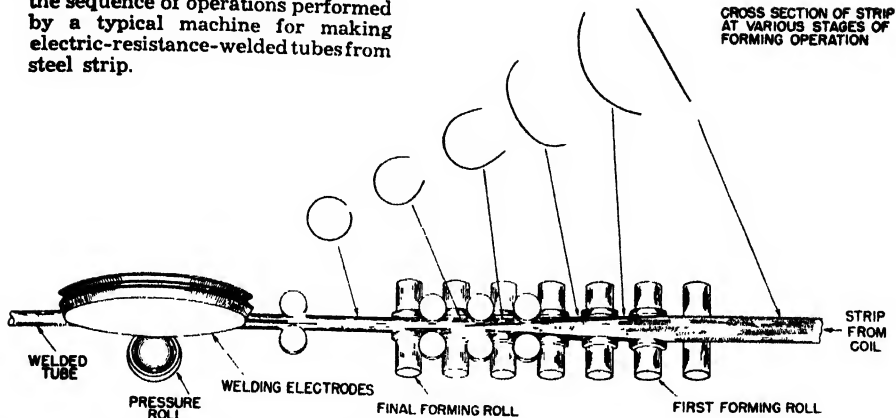
After the strip has been threaded completely through the line, the recoiler and slitter motors are adjusted to pull it through the slitter knives with any desired tension. The slitter, recoiler, scrap winder, breaker rolls and pay-off reel are each driven by separate motors. However, the pay-off reel and breaker rolls are not synchronized with the others as they are used only when threading the strip.

When the entire coil has been slit and wound on the recoiler mandrel, steel bands are placed around each of the narrow coils. An unloading buggy is elevated under the coils which, after the mandrel is collapsed, are pushed off hydraulically onto the buggy. The coils are then banded, weighed and are ready for the tube-welding machine.

Forming—Although the range of sizes handled by each mill is limited, the process of forming and welding the tubing in the several mills is identical except for the number of rolls and the power required.

Coils from the slitting line are fed either directly into forming rolls or into a "looper" to permit continuous

FIG. 41—13. Schematic representation of the sequence of operations performed by a typical machine for making electric-resistance-welded tubes from steel strip.



forming of strip welded end-to-end in the smaller sizes. The strip first passes through an edge conditioner where the edges are shaved by cutting tools to present a smooth, clean surface for welding. The strip then passes through a series of from six to nine pairs of horizontal and three or four pairs of vertical forming rolls (Figure 41—13). The first five pairs of horizontal rolls are driven from a line shaft. The other rolls are not driven. The lower rolls are on fixed centers and the upper rolls, driven through link gearing, are adjustable so that the strip, first shaped into a shallow "U," is progressively made deeper. Finally, the sides are bent in and the strip formed into a circular shape as it leaves the last stand

of rolls. In forming wider strip, which is not looped, additional stands of rolls are used and a pair of pinch rolls provided to assist in feeding the first end of the strip into the mill.

Welding the Tube—The open tube passes directly from the forming rolls to the welding electrodes (Figure 41—14) where the seam is welded by the electric-resistance method. These electrodes are copper discs mounted on a revolving transformer so that one disc, or electrode, is on each side of the opening to be welded. The tube is guided under the electrode and pressure maintained at the weld by vertical rolls.

The welded tubes then pass under a cutting tool which

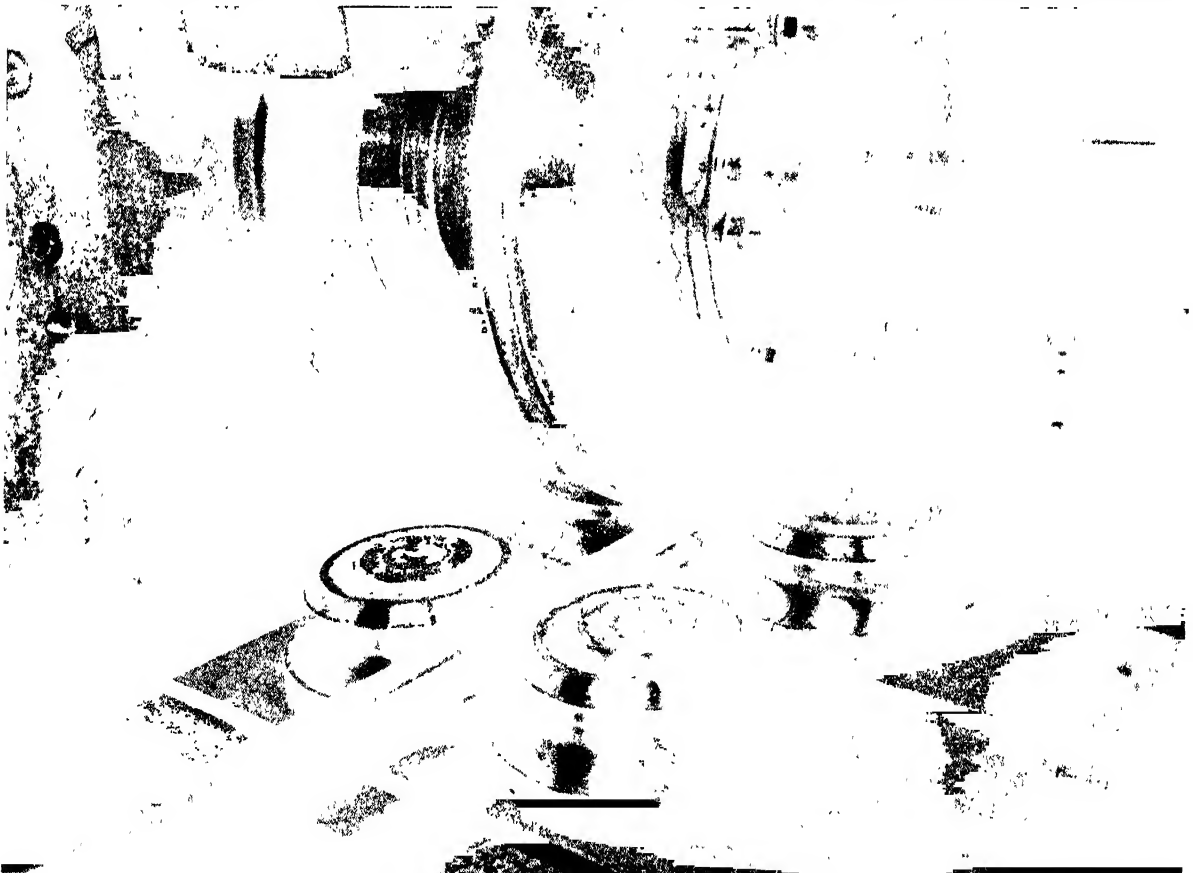


FIG. 41—14. Tube passing under welding electrodes.

removes the outside flash which resulted from the pressure during welding. After removing the flash, the tube is cooled by passing through a spray of soluble oil.

Sizing the Welded Tube—The sizing mill consists of four pairs of driven horizontal rolls and four pairs of idle vertical rolls all of which are mounted in the same manner as the forming rolls and are made of tool steel. Sizing is performed cold.

Finishing—The outside flash from the welding operation is removed by a cutting blade as was described under welding the tube. To remove the inside flash or burr on sizes over $1\frac{1}{4}$ inches, a roller-type mandrel can be placed in the tube a short distance from the welding electrode. To perform this operation an additional roll stand is required to support the outside of the tube. When the burr must be removed from tubes $1\frac{1}{4}$ inches

in diameter or smaller, the operation is performed by an air hammer located in the middle of the sizing mill. Hammering is done on the seam with a solid mandrel on the inside of the tube which is also supported underneath by a grooved die.

After the tube leaves the sizing mill it is cut to determined lengths by a rotary-head cut-off machine which travels with the tube during the cut. Either disc cutters or parting tools may be used.

The cut tubes are then transferred to the finishing floor where they are straightened in rotary straighteners. The tube is then cropped or cut to length by rotary-head cut-off machines and processed through a battery of profilers which face the ends and chamfer the O.D. and I.D. after which they are inspected and packed for shipment.

SECTION 4

ELECTRIC-WELDED LARGE-DIAMETER PIPE

Applications of the Process—Large-diameter pipe in sizes beyond the practical limits of the seamless process is fabricated by electric welding. This type of pipe, which is employed for water lines, gas mains, oil lines, tanks, pressure vessels, etc., may be made from rolled-steel plate of any weldable quality. The size of the pipe which can be made by this process is practically unlimited, since, when the circumference of the desired pipe exceeds the plate width capacity of the rolling mill, two or more plates may be welded together longitudinally to provide the necessary width or, if only a short length is desired, the plate may be bent lengthwise, permitting it to be formed with only one longitudinal weld. Where long-length, large-diameter pipe is required, the desired length may be made by welding together two or more pieces circumferentially.

Steps in the Manufacture of Electric-Welded Pipe—The sequence of operations required to make plates into pipe by the electric-weld process as practiced by National Tube Division are shearing, planing, crimping, bending, welding, expanding and finishing.

Shearing and Planing—Plates employed in the manufacture of electric-welded pipe are rolled on either a plate or strip mill as described elsewhere in this book. The plate is transferred to the edge-planing machine where it is aligned so that the two edges will be parallel and square with the ends after planing. A clamping bar, hydraulically operated, holds the plate during the edging operation. Along the full length of both sides of the table, lead screws drive carriages carrying a series of cutting tools which trim the edges of the plate.

The series of cutting tools are arranged so that no single tool will be required to remove more than a reasonable amount of stock from the rough edges of the rolled or sheared plate.

Crimping—Forming plate into the circular shape required for pipe is usually performed in three operations. The first operation, called crimping, consists of bending the edges of the plate so as to avoid a flat surface near the longitudinal seam of the pipe.

Crimping may be performed in a large crimping press which deforms the edges of the plate for a distance of approximately 6 inches in a hydraulically operated press. Crimping may also be performed by crimping rolls which roll the edges to the desired radius as the plate is drawn through the roll pass.

Bending—The crimped plate is then conveyed to what is called the "U"-ing machine. In this operation, the plate is centered over a series of parallel rocker-type

dies which lie along the axis of the plate. A large "U"-shaped die, which is as long as the longest length of plate fabricated and which is operated by a 2000-ton press, is moved down on the plate, forcing it between the dies which automatically conform themselves to the operation and assist in forming the plate into the "U" shape. The plate is then transferred to what is called the "O"-ing machine. This machine consists of two semi-circular dies which are as long as the plate to be formed. Rollers mounted on vertical spindles prevent the "U"-shaped plate from falling and keep it in correct alignment as it enters the "O"-ing machine. The "U"-shaped plate rests in the bottom die, and the top die, operated by an 18,000-ton hydraulic press, is forced down, deforming the plate until it is the shape of an almost closed circle which is then ready for welding.

Welding—Welding may be performed by any one of the numerous methods. However, the most common is the submerged-arc method. In this method, coalescence is produced by heating with an electric arc or arcs between bare-metal electrode or electrodes and the work. The welding is shielded by a blanket of granular, fusible material or flux on the work. Pressure is not used and filler metal is obtained from the electrode and sometimes from a supplementary welding rod.

It is extremely important that the gap in the formed pipe be properly positioned for welding. Two steel archways supporting a longitudinal guide are located over the conveyor approaching the welding machine. As the pipe moves over the conveyor, the guide enters and continues along the gap, guiding it into the welding machine.

The welding operation is started by striking an arc beneath the flux on the work. The heat produced melts the surrounding flux so that it forms a sub-surface conductive pool which is kept fluid by the continuous flow of current. The end of the electrode and the work piece directly under it become molten and molten filler metal is deposited from the metal electrode onto the work. The molten filler metal displaces the flux and forms the weld.

The specially designed welding heads used for this process perform the triple function of progressively depositing the flux along the joint, feeding the electrode, and transmitting welding current to the electrodes. The flux is supplied from a hopper connected to the head by tubing. The bare electrode is fed into the welding head from a coil mounted on a reel.

After the pipe is welded on the outside, the weld is inspected and conditioned on the ends when required. At this time, square pieces of metal are welded to the

FIG. 41-15. The "U"-ing machine forming the crimped plate into a "U" shape.

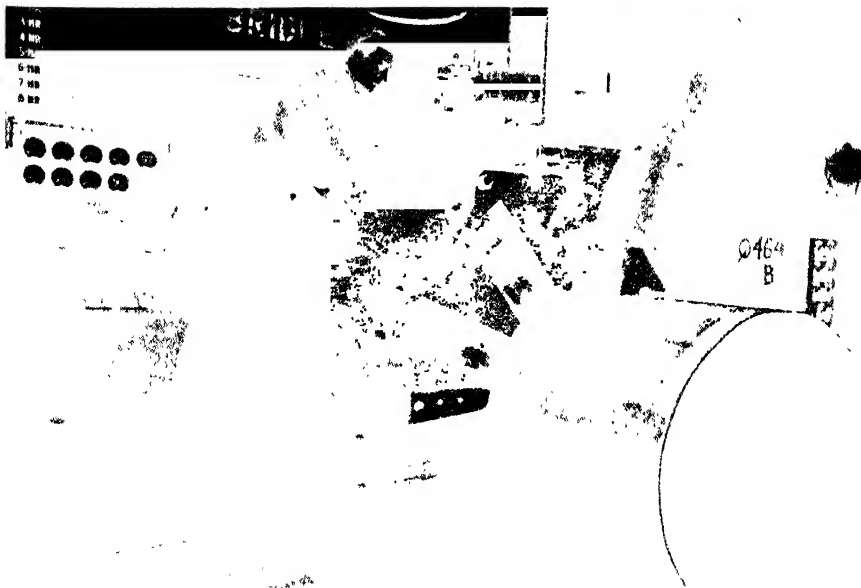


FIG. 41-16. 30-inch O.D. pipe emerging from the outside-welding machine.



FIG. 41—17. Expanding a 40-ft. length of 30-inch O.D. pipe in the hydraulic expanding machine.

pipe at the ends of the weld to enable the inside-welding machine to start at the end of the pipe.

The automatic machine which does the internal welding is similar in design to the outside welder, and is mounted on the end of a long cantilever arm and the pipe is drawn over this arm by a carriage.

After welding, the scaly deposit left from the flux must be cleaned out. This is accomplished by running the pipe over a series of rollers and, at the same time allowing the entry of a cantilevered tube which is attached to a vacuum system. The weld is then carefully inspected and any defects corrected with manual welding equipment.

Sizing or Expanding—The final pipe diameter is obtained by hydraulically expanding the welded shell against a retaining jacket. The pipe is placed in an ex-

panding machine and mandrels forced into each end expanding the pipe to the required diameter at the ends only. Retainer jackets then encircle the body of the pipe which is filled with water and expanded with hydraulic pressure to the limits of the jackets. The pressure is then reduced and serves as a hydrostatic test. In addition to obtaining close diameter control, this method cold works the metal, obtaining higher mechanical properties, and is a good test of the weld.

Finishing—Following the rounding-up or expansion operation, the pipe is placed in special machines which face the ends. This operation insures that the ends will be smooth and accurately within a plane at right angles to the longitudinal axis of the pipe. If the pipe is being prepared for welded joints, the ends are beveled in this operation.

SECTION 5

SEAMLESS STEEL TUBULAR PRODUCTS

Historical—Although the modern developments for manufacturing seamless or weldless steel tubes did not begin until about 1890, they are, nevertheless, the result of nearly a century of experimentation. As mentioned in the introduction to this chapter, the first attempts to make seamless tubes of steel that were commercially successful involved the two basic processes known as cupping and piercing. In the cupping process, which was first used about 1845, a circular sheet or plate was forced by successive operations through several pairs of conical dies, each pair being deeper and more nearly cylindrical than the previous one, until the plate took the form of a tube, or cylinder with one end closed. This method is still in use for certain sizes. In the piercing process, a round hole was first made along the central axis of a round billet which was then rolled and drawn over mandrels to form a tube. The mill for lengthening such hollow billets was first patented in England by Church and Harlon about 1841. The rolling mill and the draw

bench afforded simple and comparatively cheap methods of elongating the hollow billet, but the development of a method for piercing it proved a difficult matter. The oldest and simplest method was to heat the billet to a high forging temperature and hydraulically force a punch through its center while in this hot state. As it was essential that the hole be exactly concentric with the billet throughout its entire length, a feat that is hard to accomplish with a billet more than a foot or so in length, recourse was had to drilling a small hole in the cold billet, then heating the billet and enlarging the hole by piercing in a press. About 1888 a patent was granted for a process whereby a small ingot was cast about a core of refractory material, which hollow ingot was to be treated as described for hollow billets.

In the decade preceding the twentieth century, the bicycle became a very popular vehicle, and the growth of the bicycle industry created a demand for high-class seamless tubes in very large quantities. The early

methods of seamless tube manufacture were slow and tedious, and orders aggregating millions of feet of tubing for bicycle construction could not be filled. In 1895 the first American seamless tube plant, using rotary piercing, was constructed in Ellwood City, Pennsylvania, and operated successfully for many years in the manufacture of bicycle tubing. The equipment at this plant consisted of a **Stiefel disc piercer**, a **Pilger rolling mill**, and cold-drawing benches on which the tubes were finished.

After 1900 the automotive industry began to grow and had a remarkable influence on the seamless tube industry. In the first place, the popularity of the automobile soon caused a great decrease in the manufacture of bicycles, with a consequent loss of seamless tube business. On the other hand, the great increase in the use of automobiles created an immense demand for motor fuels and lubricants, which greatly stimulated expansion in the oil industry and thus developed a new and enormous market for seamless tubes. In the transition period, the seamless tube manufacturer turned to the boiler industry, and the next step in the development of the seamless process related to the production of all forms of stationary and locomotive boiler tubes. The seamless tube soon displaced the wrought-iron and lap-welded steel tubes which had previously been standard in boilers.

After World War I the discovery of enormous oil pools with flush production created a demand for pipe to be used in the wells and in the transmission lines. The later discovery, after 1920, of huge natural-gas reservoirs in remote districts, from which the gas could be transported to consuming markets only in steel pipe lines, sometimes over 1,000 miles long, resulted in another large market for seamless pipe. Some idea of the size of the oil and gas industry in the United States may be gained from the fact that there are over 500,000 producing oil wells and over 550,000 miles of pipe lines in existence.

The seamless tube mills prior to 1920 were scarcely able to make pipe over 6 inches in diameter and longer than 25 feet. In 1925 National Tube Division developed the so-called **double-piercing process**, which consisted of piercing a heavy-walled shell and then expanding this in a second piercing operation to a larger diameter tube with a lighter wall. This development made possible the production of tubes up to 16 inches in diameter. The Pilger process had also reached a production stage at this time, but a thorough study of this method caused its rejection for American manufacturing conditions because of the low production per man-hour. Further study of the **double piercing and expanding process** later led to the rotary-rolling process, by which it is now possible to produce seamless tubes up to 26 inches in diameter and in lengths exceeding 40 feet.

Scope and Requirements of Seamless Tube Products—The expansion of the seamless tube industry described above was also greatly influenced by the adaptation of the piercing process to the many different types of steels which were developed during this period. Steels melted by many processes can now be successfully converted into seamless tubes. In general, killed open-hearth and electric-furnace steels are used. However, improvements in the technique of Bessemer-steel manufacture have resulted in the production of large quantities of deoxidized acid-Bessemer steel which is converted into seamless tubes with excellent results.

Because of the severity of the forging operation involved in piercing, the steels used for seamless tubes must have good characteristics with respect to both surface and internal soundness. A study of the character of the steel as revealed by deep-etching the cross-section tends to show that a sound, dense cross-section, free

from center porosity or ingot pattern, is the most satisfactory for seamless tubes. For this reason steels of the thoroughly killed types are to be preferred to strongly rimming steels or semi-killed steels. Although rimming steels and semi-killed steels have been successfully converted into seamless tubes, the results are not always uniform, either because of a poor inside surface, due to internal porosity, or because of high losses from external seams, caused by surface defects. Metallurgical developments since 1920 have contributed greatly to the improvement of steels for seamless tubes and are discussed in detail in another section.

As a result, the seamless process has been extended to include practically all of the regular and alloy grades of steel. At the present time all of the ordinary carbon steels, even those containing as much as 1½ per cent of carbon, are processed in commercial quantities. Many of the high-sulphur steels developed for machining are also manufactured into seamless tubing.

All of the intermediate alloy steels, such as those listed in the specifications of the American Iron and Steel Institute and the Society of Automotive Engineers, are available in tubing. As a consequence the application of seamless tubing to automotive parts has been extended until there is now between 150 and 200 lbs. of tubing in each car manufactured. The alloy steels used by the ball and roller-bearing industry are now available in seamless tubing. This tubing is made from steel of special cleanliness.

In the last twenty years, numerous steels of special composition for heat and corrosion resistance have been developed. Seamless tubes are now satisfactorily made from material of most of these steels. These include the chromium steels containing from 1 to 30 per cent chromium and numerous other alloyed steels containing chromium with additions of such elements as molybdenum, nickel, manganese, columbium (niobium), silicon, and titanium. In general, it can be stated that all of the alloys which are ferritic or pearlitic in structure are satisfactory for piercing. The alloy steels which are generally of the austenitic type can be successfully pierced if the austenite remains stable at forging temperatures. A few of the austenitic steels which have a considerable proportion of ferrite at forging temperatures have not, as yet, been satisfactorily pierced. For example, an alloy such as the 25 per cent chromium—10 per cent nickel, which is partially ferritic, will not pierce, whereas an alloy with 25 per cent chromium and 20 per cent nickel can be pierced satisfactorily.

Seamless pipe of numerous special properties has been developed to meet the needs of the oil industry. In this field the pipe employed for rotary drilling must possess great torsional strength and high resistance to fatigue stresses. An idea of the stresses to which the pipe is exposed in drilling service may be gained from the realization that the power for the rotation of the drill bit on the lower end of the pipe is applied at the upper extremity, which, in some cases, is more than three miles distant. The string of drill pipe itself may weigh up to 370,000 lb., and when rotating at 200 r.p.m. or more, only material of the highest strength and toughness will perform satisfactorily. Drill pipe is usually made with heavy upset ends, and the full lengths of pipe are normalized on the completion of the forging operations.

The casing for such deep oil wells requires seamless pipe having a high resistance to collapse to withstand the high external pressures which exist. Casing has been set to depths of nearly 20,000 ft. under very high hydrostatic heads. Other classes of seamless pipe and specialty tubing made for the oil industry include oil-well tubing, pump tubing, and line pipe.

The expansion of the oil-refining, chemical, and high-pressure steam industries has also demanded special seamless pipe. Satisfactory alloy-steel tubing and pipe are made to withstand temperatures up to 1200° F coincident with pressures as high as 3000 lbs. These industries also use high-pressure pipe lines at temperatures as low as -150° F, and steel pipe which is tough and strong at these low temperatures is now produced in considerable quantities.

Steelmaking Practices—As indicated previously, the steel for seamless tubes must be made with special forging properties. The major portion of the tonnage used consists of straight carbon steels, ranging from 0.05 per cent to 0.50 per cent carbon and 0.35 per cent to 1.50 per cent manganese. These grades are made in standard, stationary open-hearth furnaces and special tilting open-hearth furnaces employing the duplex melting process. Many of the special alloy steels are necessarily melted in basic electric furnaces to obtain the particular metallurgical requirements which are often specified. The open-hearth furnaces tap between 150 and 180 net tons and are of the mixed-fuel type, burning tar or oil and coke-oven or natural gas.

The stationary open-hearth furnaces making these steels at National Tube Division operate on approximately a 40-per cent cold-scrap, 60-per cent hot-metal (basic iron) charge, while the tilting furnaces are charged with a mixture of blown metal from the Bessemer converter and the requisite amount of molten pig iron with varying additions of cold steel scrap, depending upon the conditions at the plant. Regardless of the initial charge in either the stationary open-hearth or the duplex melting furnace, the slags employed and the final working and refining periods in the two types of furnace are identical.

As stated previously, seamless tubes are also produced from deoxidized Bessemer steel. This is an acid-Bessemer steel which has been treated in the vessel to remove a considerable portion of the dissolved oxygen as a gas and further treated in the ladle to develop a silicon content of 0.10 to 0.30 per cent. Deoxidized Bessemer may also be produced by treatment in the ladle with sufficient aluminum or its equivalent to insure practically no evolution of gases during solidification. Deoxidized acid-Bessemer steel is a killed steel with a homogeneous ingot structure as compared to the conventional rimmed or capped Bessemer steels. The importance of having good forging properties can best be realized by the fact that previous to the development of deoxidized Bessemer steel, it was commercially impractical to produce seamless pipe, by the piercing process, from Bessemer steels.

Deoxidized acid-Bessemer steel has a high ratio of yield strength to tensile strength. Its machining and welding properties are similar to open-hearth steels of equivalent mechanical properties. The combination of high yield strength and good ductility makes it peculiarly suited for seamless tubular products. Over a period of years, a considerable amount of experience has been gained by the National Tube Division in applying deoxidized acid-Bessemer steel to many types of seamless tubular products.

Since the surface quality of the steel for seamless tubing is of the utmost importance, molds are thoroughly cleaned, hot dipped, and coated on the inside surface with pitch or other carbonaceous material before the steel is teemed. The ingots used are of corrugated contour and range in weight from 10,000 to 22,500 lb. Various forms of hot tops are used for special requirements.

Rolling Mills—The rolling mills consist of blooming mills and bar mills. The practice employed at the rolling

mills in preparing the semifinished round used in producing seamless pipe has a definite influence on the quality of the finished product. The time elapsing between pouring of the ingots and charging in the soaking pits, the time in the soaking pits, and the heating of the ingots at carefully regulated temperatures for the various grades of steel are some of the factors which receive close attention and are made a matter of supervision by the metallurgical departments. The use of rolls without ragging and the practice of rotating the ingot through 90 degrees after two drafts in collared passes have proved advantageous in eliminating mechanical surface defects.

The uniformly heated ingots, varying in size from 23½-inch by 26½-inch to 32-inch by 32-inch depending on the bloom size and billet requirement, are rolled in 40-inch two-high reversing mills according to definite schedules of light drafts in the initial passes and uniform reductions in succeeding passes. When the ingot has been reduced to the required bloom section, it is sheared for the top and bottom discard, the extent of the discard being regulated by the type of steel, the character of the heat, and the appearance of the blooms produced.

The major portion of the blooming-mill product is delivered direct after shearing and without reheating to two-high reversing bar mills which are located in tandem with the blooming mill. The bar mills are provided with a number of extra housing sets, which permits a roll change to be made in less than 10 minutes. Rounds 4¾ inches to 13¾ inches in diameter are thus rolled direct from ingots. Smaller-sized rounds are produced by the reheating of small blooms and rolling on additional bar mills.

The bar-mill rounds are hot-sawed to length and centered for the seamless mills. Seamless-tube rounds must be rolled to an accurate diameter and cut to accurate lengths, as each round produces a given length of pipe and accurate round weights are necessary. All rounds are hot-stamped for identification and racked in cradles for the various tube orders.

The solid rounds used for making seamless tubes must undergo a thorough surface-conditioning operation. It is now general practice to peel all rounds 4¾ inches and larger, and then by chipping or by scarfing with an oxy-acetylene torch to remove any remaining seams or other defects evident to the inspectors. Rounds smaller than 4¾ inches in diameter are converted from small blooms from which the surface defects have been removed by chipping or scarfing. The most careful attention to the thorough removal of surface defects is necessary to eliminate seams and other pipe-mill losses.

Sequence of Operations—The sequence of operations in seamless-pipe mills varies slightly from mill to mill. At the Lorain Works of National Tube Division it follows three general patterns determined by the size of the pipe to be produced. The round billets are first uniformly heated to piercing temperature in special gas- or liquid-fuel-fired furnaces. They are then processed through the various operations in the following order, depending upon the diameter of tube to be produced:

Pipe Size—2-inch to 4½-inch diameter
Piercing Mill
Plug Rolling Mill
Reeling Machine
Reheating Furnace
Sizing Machine

Pipe Size—3½-inch to 16-inch diameter
First Piercing Mill
Second Piercing Mill

(Continued on Next Page)

(Continued from preceding page)

Reheating Furnace
Plug Rolling Mill
Reeling Machine
Sizing Machine

Pipe Size—14-inch to 26-inch diameter

First Piercing Mill
Second Piercing Mill
Reheating Furnace
Plug Rolling Mill
Reheating Furnace
Rotary Rolling Mill
Reeling Machine
Reheating Furnace
Sizing Mill

The operation of each of the above units will be described in detail in the remainder of this section. In the overlapping sizes of the range 3½-inch to 4½-inch diameter, the heavier-wall pipe is processed through only one piercing operation whereas the two piercing mills are employed in producing the light walls. Also, in the overlapping sizes of the range 14-inch to 16-inch diameter, the lighter-wall pipe is rotary rolled, the heavier-wall pipe by-passing the second reheating operation and the rotary-rolling mill.

The Mannesmann Machine for piercing round billets for making seamless tubes was patented in 1885. In this machine, the principle of helical rolling is employed. The two steel rolls, which bring into play the forces used to produce the cavity in the work piece, are positioned side by side and have their axes inclined at opposite angles 6 degrees to 12 degrees with the horizontal center-

line of the mill. These rolls measure from 20 inches to 30 inches in length and from 32 inches to 48 inches in diameter. The roll surfaces are contoured so that, in the horizontal plane through the centerline of the pass, the space between the rolls converges toward the delivery side for a length of from 5 inches to 15 inches to a minimum, called the gorge, and then diverges to form the pass outlet. The converging and diverging angles formed by the roll surfaces vary from 2 degrees to 12 degrees. The shafts of these rolls are mounted in bearings which can be adjusted laterally in the housing to permit the space between the rolls to be properly set for the size of work piece being rolled. The inlet end of each roll shaft is fitted with a universal coupling which, through long spindles, connects with a common reduction gear powered by an electric motor. The size of this motor and reduction unit depends on the size range to be produced on the mill, varying between 700 and 3,500 h.p. These motors are designed for a pullout torque of 300 per cent. The rolls are cooled by water sprays. The elevation of the centerline of the pass is determined by two guides, one of which is mounted above and the other below the center of the mill in the space between the rolls.

Between these guides in the pass outlet a projectile-shaped piercing mandrel is held in position on the end of a water-cooled mandrel-support bar, located on the delivery side of the mill. The opposite end of this bar is mounted in a thrust bearing which is carried in a reciprocating carriage that is latched stationary during the piercing operation. The pointed end of the piercing mandrel extends just beyond the gorge toward the entering side of the rolls.

The Operation of Piercing—A solid round bar or billet of the proper length and diameter to make the size and

FIG. 41-18. Mannesmann piercer in operation.



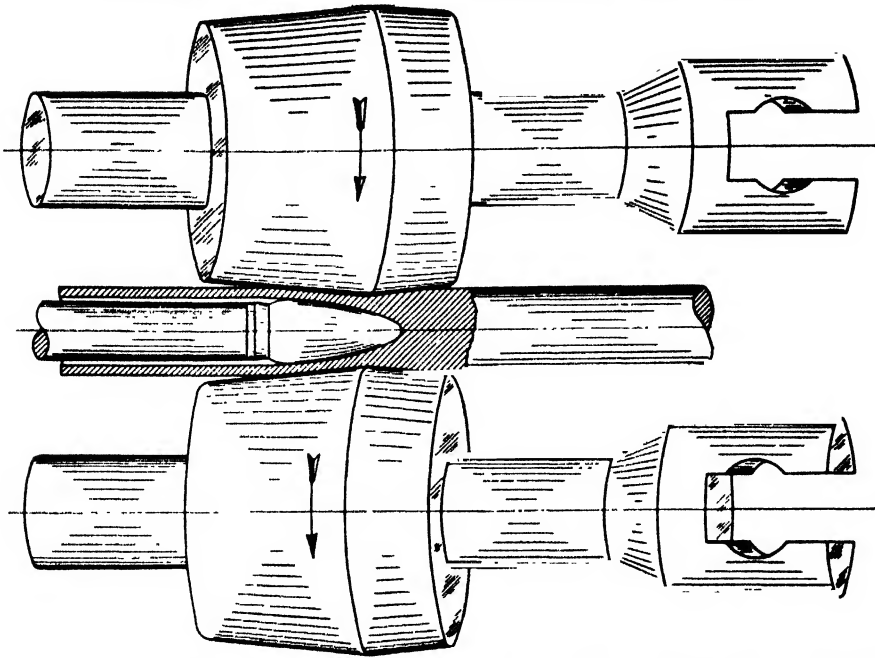
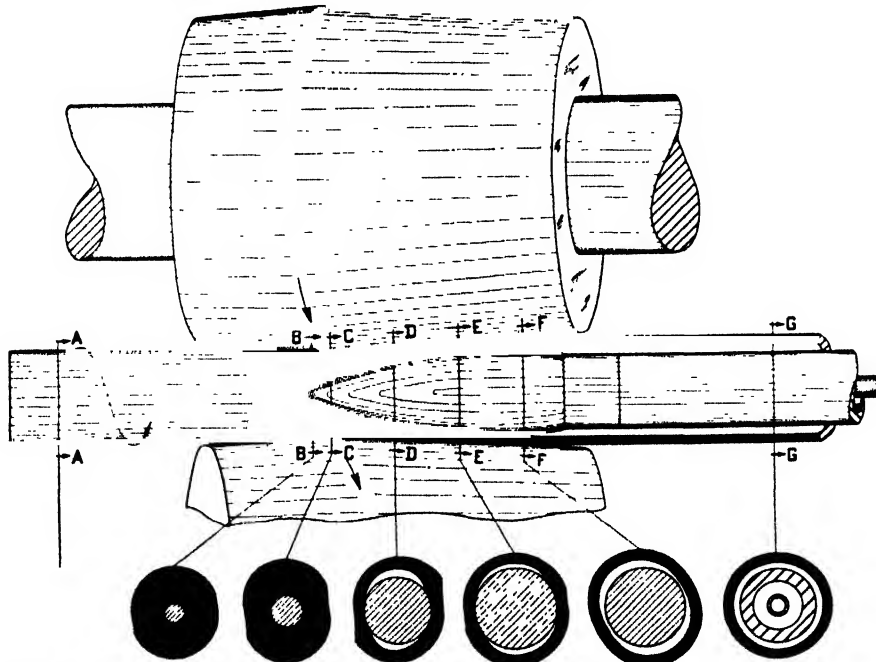


FIG. 41-19. Sketch illustrating action of different parts of Mannesmann piercer in the piercing of a solid billet.

weight of tube desired, is heated uniformly to the usual temperature for rolling light sections. With the rolls revolving at constant speed (800 to 1000 surface feet per min.), the heated billet is transferred to a horizontal trough, which positions the axis of the billet on the inlet side of the mill coincident with the centerline of the pass.

The heated billet is pushed forward into the space between the rolls, which has been adjusted so that the gorge is approximately $\frac{3}{4}$ inch less than the diameter of the billet. As soon as the leading end of the billet has

contacted the rolls, the force of the pusher is removed. Because of the obliquity of the roll axes, the motion imparted to the billet between the rolls is one of rotation and axial advance. When the leading end of the billet has advanced to the gorge, it encounters the nose or pointed end of the piercing mandrel. The grip of the rolls is sufficient to continue the advance of the work piece against the retarding effect imposed by the piercing mandrel. When the rearward end of the billet is rolled clear of the piercer mandrel, the thrust-bearing carriage



SECTION A-A SECTION B-B SECTION C-C SECTION D-D SECTION E-E SECTION F-F SECTION G-G
FIG. 41-20. Sketches illustrating action of rotary-piercing mill on the round billet.

is unlatched and the mandrel withdrawn from the billet, now a hollow shell, which is then conveyed to a reheating furnace in preparation for further fabrication into a finished tube. These shells are produced in lengths up to 26 feet in less than a minute.

The Action of the Rolls—It is evident that the forward motion of the billet is caused by the inclination of the axes of the rolls. How these two rolls, by exerting pressure only on a surface of the billet, are able to force the metal over the mandrel to form a tube from a solid billet is not so readily grasped. It is to be especially noted that the mandrel is not forced through the metal, but that the rolls cause the metal to flow over and about the mandrel. To bring about this result, the rolls must first draw metal away from the center of the billet, which action tends to form a central hole, or cavity, for the entrance of the piercer point. The truth of this statement is evident from the fact that a small, but somewhat irregular, hole may be formed in a billet without the use of the piercer point. In practice, the end of the piercer mandrel is placed sufficiently forward to prevent the formation of a cavity ahead of the point. Advantage is taken only of the tendency of the rolls to form this cavity.

The Principle Involved in Forming the Cavity—Indeed, such a hole can be opened up in the center of any solid cylindrically shaped plastic body by rolling it between, even, two flat surfaces. Steel workers, particularly hammermen, are familiar with the fact that, if a piece of steel in the form of a round be pressed or hammered into an oval form several times in succession, a rupture will occur in the center that will extend longitudinally through the middle of the bar. The reason for the formation of this rupture is plainly due to that fact that when pressure is applied to the round bar at diametrically opposite points sufficient to deform it, making one diameter shorter and that at right angles to it longer, the spreading of the metal, which takes place along the long diameter and in opposite directions, sets up a lateral tension that may cause its particles to be drawn away from the center (Figure 41—20).

Flow of the Metal in Piercing—As the billet, which is in a plastic state, enters the mill, the rolls grasp it at diametrically opposite points on its circumference. As they draw the billet forward in the converging portion of the pass, they continue to compress it at these opposite

points and, since the billet is being revolved rapidly, these points are continually changing. As the compressive rolling or cross-rolling proceeds from the point of initial contact of the piercer rolls to the gorge, the diameter of the billet is reduced and the section is changed from a circle to an oval with the long diameter in a vertical position. Since the billet is rotating, the central portion is acted on by all of the forces which are applied around its circumference during successive contacts between the billet and rolls. If a sufficient reduction is effected in this manner, a cavity will be formed in the center of the billet even without the presence of a piercer mandrel. In practice, this cavity is not permitted to form in advance of the nose of the mandrel, since the rough surface of the self-formed cavity might not permit the inner surface of the shell to be subsequently rolled smooth by the action of the mandrel. It is for this reason that the end of the piercer mandrel is positioned in advance of the gorge so that it will actually effect the opening in the center of the billet, being assisted in its function by the cross-rolling action. To avoid the difficulty of accurately centering the mandrel, the forward end of the billet is centered to insure that the point of the mandrel will penetrate the billet at or very near its axis. Centering is not absolutely necessary; however, it assists in starting the end of the mandrel in the center of the billet and reduces wear on the mandrel. If the end of the piercer mandrel is positioned too far in advance of the gorge, the cross-roll action on the center of the billet will not be great enough to reduce the resistance sufficiently to permit the grip of the rolls to advance the billet over the mandrel.

Once the end of the mandrel has penetrated the axial center of the billet, the piercer mandrel serves as a third roll so that, with the properly designed pass, the metal of the work piece is helically rolled over the piercing mandrel (rather than extruded) to produce the hollow shell. The grip of the rolls on the hollow in the diverging portion of the pass, due to the obliquity of the roll axes, tends to draw the billet forward as the reduction in wall thickness is being made, and, as a consequence, tends to increase the length of the pierced hollow shell at the expense of diameter.

Double Piercing—It is to be especially noted that the amount of metal displaced increases considerably as the O.D. size (outside diameter) is increased. In 1925, the National Tube Division developed the process known as double piercing. In this process the solid billet is

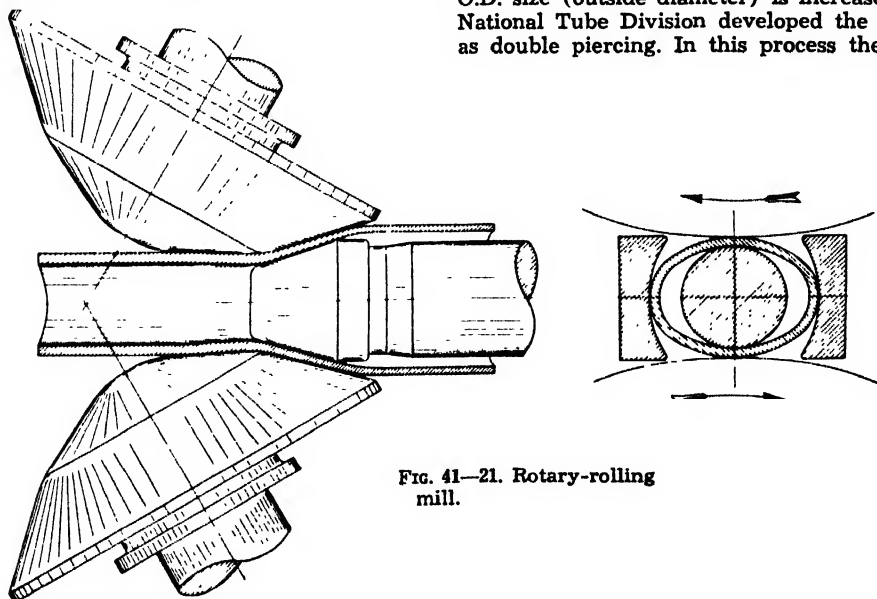


FIG. 41—21. Rotary-rolling mill.

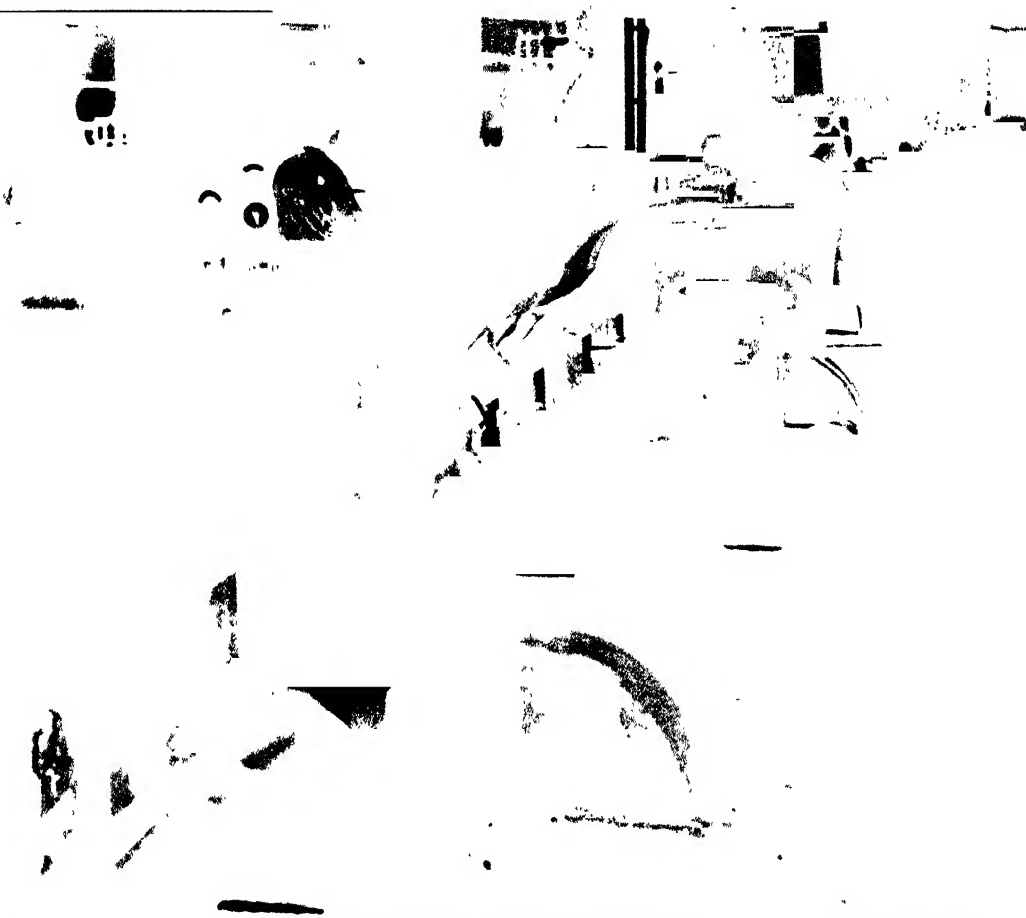


FIG. 41—22. Rotary-rolling mill in operation.

first pierced to a comparatively heavy-walled shell, after which, without reheating, it is put through a second piercing mill. The second mill further reduces the wall thickness and increases the diameter and length of the piece. This practice has extended the permissible diameter range of the automatic-mill method of producing seamless tubes by dividing the requisite work of piercing in the two stages. The double-piercing process has the further advantage that it definitely improves the concentricity of the tube, since any eccentricity produced in the first operation results in a temperature differential between the thick and thin sections which favorably influences the displacement made in the second piercing mill.

Rotary Rolling—The large demand for pipe between 16 inches and 36 inches in outside diameter for the transportation of natural gas for long distances raised a serious question as to the best manner of manufacturing such pipe. The existing lap-weld processes for making such pipe sizes were both costly and slow and also unsuited for the manufacture of lengths over 40 feet. It is also not feasible to roll pipe over 16 inches in diameter on the automatic rolling mill, and this feature made it questionable whether seamless pipe in these sizes could be economically made. The National Tube Division developed the rotary-rolling mill, which has made possible the production of pipe as large as 26 inches in outside diameter in lengths up to 45 feet and with wall thicknesses as light as 0.312 inch. Other large sizes are produced with wall thicknesses as light as 0.250 inch.

The rotary-rolling mill is a modification and enlargement of the cone-type piercing mill. The shafts of this mill, which drive the 74-inch (diameter) conical rolls, are in separated horizontal planes and are at an angle of 60 degrees with the axis of the pipe being rolled. A diagram of this mill is shown in Figure 41—21. Each shaft is powered with a 1500-h.p., 200-500-r.p.m., D.C. motor, which provides peripheral roll speeds of 800 to 2400 feet per minute. In operation, the conical rolls grip and spin the pipe, feeding it forward over a large tapered mandrel, thereby effecting a decrease in the wall thickness of the pipe and an increase in the diameter. The length of the tube is substantially unchanged by the operation. The rolling action is similar to that which takes place in a tire- or ring-rolling machine, except that in the rotary-rolling mill a forward helical advance is imparted to the tube, which is supported on the inside by the tapered mandrel.

The Plug Rolling Mill is a motor-driven, non-reversing, single two-high stand, which resembles a reversing bar mill. There are, however, several differences between the two. Instead of roll tables, the mill is equipped on the entering side with a movable trough and pusher and on the delivery side with a stationary guide table and mandrel-bar support. The guides on the latter table, two or more in number, are of the double-bell type and are mounted on cross beams and lined up one behind the other directly in back of the grooves in the rolls. The water-cooled mandrel bar, which is anchored in the support at the rear of the table, projects

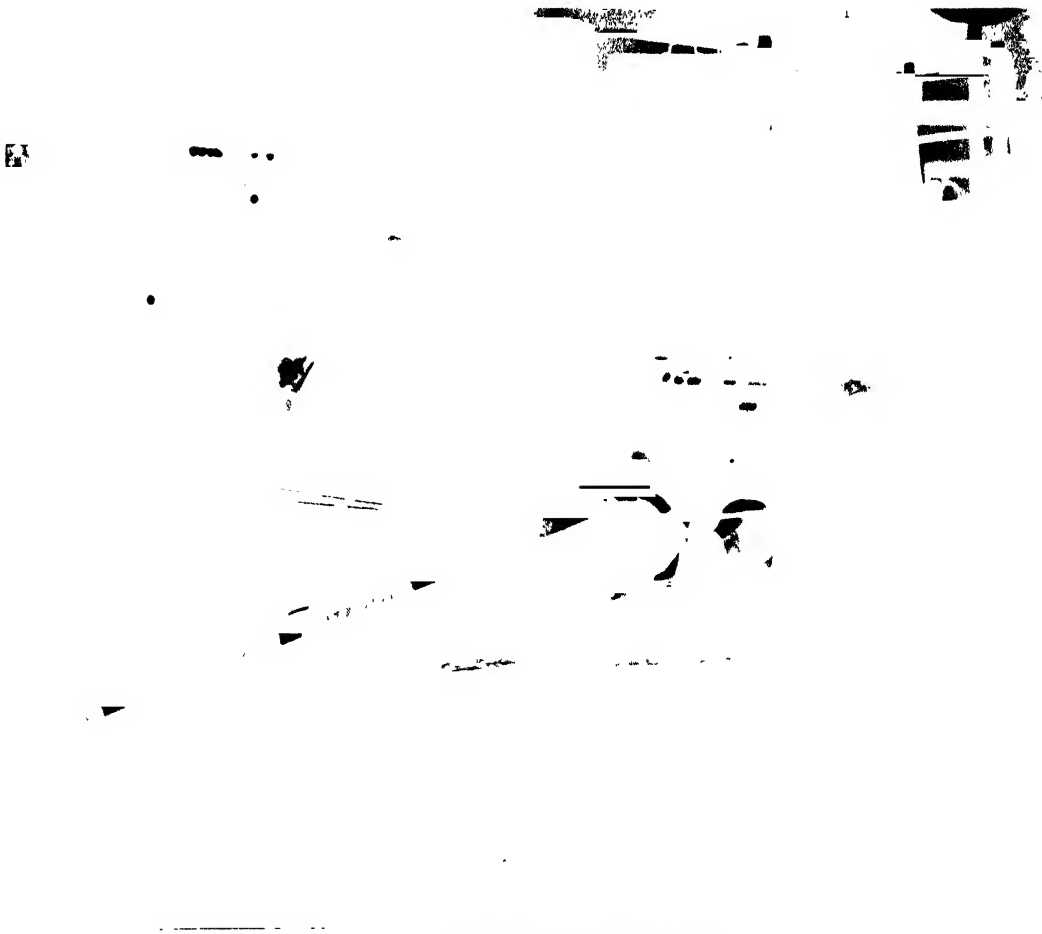


FIG. 41—23. Plug rolling mill.

through a series of these guides with its opposite end terminating about $\frac{1}{4}$ inch short of the vertical centerline of the rolls. The free end of the mandrel bar provides support for the mandrel or plug during the working cycle. The work rolls, which are from 22 inches to 38 inches in diameter, depending upon the size of the mill, have several semi-circular grooves machined in their surface. With the rolls in position one above the other, the opening formed by the groove is not a true circle but is slightly oval with the long axis in a horizontal plane. This flare of the groove at the roll surface is provided to prevent the edge of the groove from shearing the work piece. In general, only one mandrel bar and one groove are used in the rolling of a given size tube. The tube is passed through this roll stand twice, being rotated through 90 degrees between the passes so that the entire surface receives an equal and similar treatment in the slightly ovalled groove. To permit the tube to be stripped rapidly from the mandrel bar, the top roll is supported by counterweights and arranged to be elevated mechanically for a rapid opening of the pass. Stripper rolls, located just to the rear of the main rolls, are grooved to correspond with the main rolls. The lower stripper roll is also arranged for mechanical movement for rapid closing of the pass. These stripper rolls rotate in a direction opposite to that of the main rolls and function only when the top main roll is in its elevated or open pass position.

The Operation of Plug Rolling—The pierced shell, except for the smaller pipe sizes, is reheated after the

piercing operations. With the pierced shell lying in the feed or delivery trough, an alloy-steel mandrel or plug is attached to the end of the bar, the bar holding the plug at the correct position in the roll groove. The plug is somewhat larger in diameter than the support bar in order to provide clearance between the inside of the tube and the support bar. In order to start the shell over the plug and permit the rolls to secure a good bite upon it, the shell is shoved into the pass with considerable force by a compressed-air-operated ram or pusher. Once started, the force of friction due to the pressure exerted by the revolving rolls is sufficient to draw the shell rapidly over the plug, slightly reducing its diameter and wall thickness and increasing its length. As soon as the shell has passed through the groove, the mandrel is removed from the bar. The top work roll is elevated approximately $1\frac{1}{2}$ inches. The lower stripper roll is then elevated to raise the tube clear of the bottom work roll and to grip it in the grooves of the stripper rolls, which return it to the entering side of the mill. Another mandrel is then placed on the bar, and the tube is rotated through an angle of 90 degrees. The top work roll and the lower stripper roll are returned to their original position, and the pusher again enters the tube in the pass. As soon as the tube has passed through the groove for the second working pass, it is returned again to the entering side of the mill, from which it is discharged for further fabrication. In this way the wall of the tube, supported by the mandrel on the inside and subjected to the action of the rolls on the outside, is reduced in thickness to the

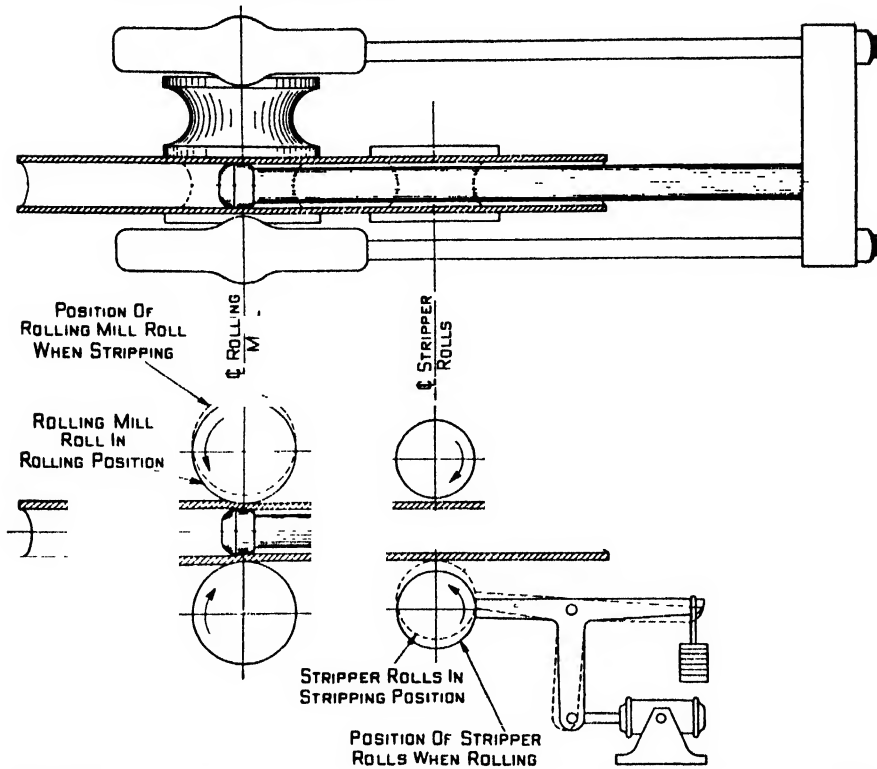


FIG. 41-24. Schematic plan view (above) and elevation (below) of a plug rolling mill.

gage desired. The pierced billet is proportionately lengthened and slightly reduced in outside diameter. The wall reduction normally made in the plug mill is approximately $\frac{1}{8}$ inch to $\frac{1}{4}$ inch. After plug rolling, the tube has a uniform wall of the desired thickness throughout but is slightly out of round or oval shaped, not perfectly straight, and still at a bright-red heat.

The Reeling Machine is similar in construction and operation to the Mannesmann piercer except that the rolls, which are about 30 inches long and 34 inches in

diameter, are almost cylindrical in form. The rolls are adjusted laterally in the same manner as that described for the Mannesmann piercer and are separated by a space a little less than the diameter of the tube to be reeled. The rolls are motor-driven and are geared together to revolve in the same direction at a surface speed of approximately 900 feet per minute. In operation, a cylindrical mandrel, which is placed between the rolls, is supported on the delivery end by a water-cooled mandrel-support bar. Like the piercing mill, the opposite

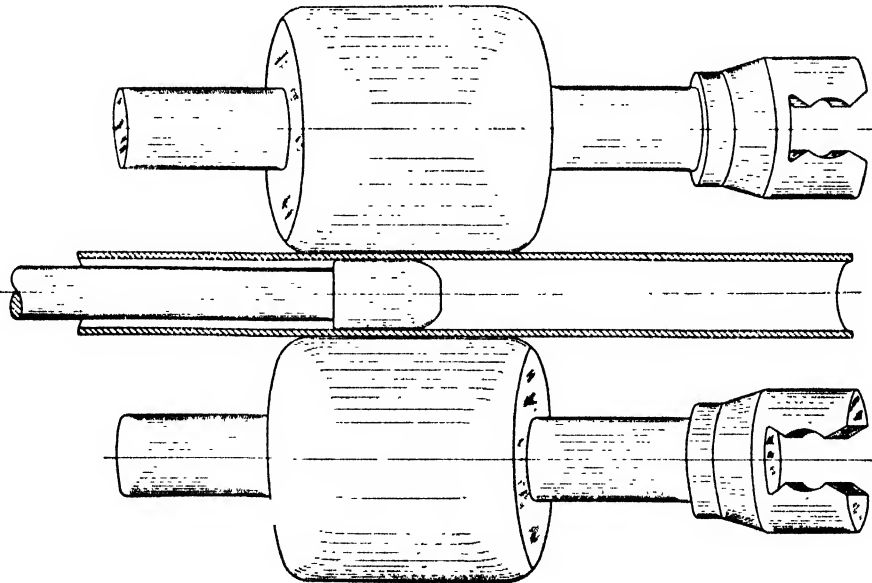


FIG. 41-25. Reeling machine.



FIG. 41—26. Reducing and sizing machine.

end of this bar is connected to a thrust bearing carried in a reciprocating carriage, locked stationary during the reeling operation. On the inlet side of the mill a conveyor carries the tube through stationary guides to contact with the rolls. Since the rolls are revolving in the same direction and with axes oppositely inclined, they cause the tube to revolve and helically advance over the mandrel. The elevation of the mandrel and tube during reeling is maintained in the proper horizontal position by stationary guides mounted between the reeler rolls above and below the pass. Owing to the fact that the total space between the reeler rolls and the mandrel is a few thousandths of an inch less than twice the wall thickness of the entering tube, a slight reduction in the thickness of the wall is effected during the reeling operation. This slight reduction made in the reeling operation has the effect of burnishing the inside and outside surfaces of the tube and slightly increasing its diameter. The function of the reeler is, therefore, to round up and to burnish the inside and outside surfaces of the tube delivered from previous fabricating operations.

Sizing the Tube—The manner of sizing the reeled tube depends on the diameter of pipe that is being produced. For sizes 3 inches and over the sizing process consists merely of passing the tube, reheated if necessary, through two or more stands of sizing rolls, the grooves in which are slightly smaller than the diameter of the tube delivered from the reeler. The diameter reduction effected in the sizing mill is made to insure uniform size and roundness throughout the length of the tube.

Since it is not economical to pierce, roll and reel tubes

of small diameter, the production of hot-finished tubes less than 3 inches in diameter requires a reducing and sizing process for which a special machine is employed. This machine is similar to a continuous rolling mill. It consists of 8 to 16 stands of two-high grooved rolls about 12 inches in diameter, arranged on the continuous plan and set about 2 feet apart, center to center. Instead of standing vertically, the housings for these rolls are inclined 45 degrees, so that adjacent stands lie at right angles to each other and the loci of the centers of the pass openings formed by the grooves, which gradually decrease in size from the first to last, are in the same straight line. The grooves in the initial stands are slightly oval in shape. However, the grooves in the last two stands on the delivery end of the mill are preferably round. As the tube from the reeling machine is too cold to be reduced in diameter, it is passed endwise into a long reheating furnace located at the entering end of the reducing mill where it is heated to a uniform temperature just below the scale-forming point. It is then pushed by a mechanical pusher directly into the first stand of the reducing rolls, being drawn continuously through the successive stands in which it is elongated and reduced to the outside diameter desired. The smallest size to which tubes are generally reduced by this process is $1\frac{1}{4}$ inches. By properly regulating the surface speed of the successive roll stands, the elongation is made to compensate almost exactly for the decrease in diameter, while the wall thickness remains substantially unchanged.

Warm Working—It has been known for some time that

the cold working of steel pipe by various drawing, rolling, or compressing methods will raise the yield strength and in some cases increase the collapse resistance. Such material worked at room temperature loses a considerable amount of ductility. If steel which has a yield point of 60,000 lb. per sq. in. is cold worked to raise the yield point to 90,000 lb. per sq. in., a considerable amount of force must be exerted. A steel having a yield point of 60,000 lb. per sq. in. at room temperature will have a yield point of approximately 30,000 lb. per sq. in. at some higher temperature. The amount of force or work required to deform the steel at this higher temperature is much less, although the increase in the yield point of the steel is quite similar to that obtained when the work is done at room temperature.

National Tube Division practice in warm working consists of making seamless pipe in the conventional manner, but slightly oversize. After hot sizing, the pipe temperature is equalized over its entire length to the desired temperature, which ranges from 650 to 1,000° F, and is quickly passed through a five- or seven-stand set of sizing rolls following which it is air cooled on regular cooling tables. The reduction in diameter in this pass is approximately 5 per cent.

The furnace used to equalize the pipe temperature is divided into two zones: a cooling, or heating zone if cold pipe is charged, and an equalizing zone. Air is heated, passed through the ducts into the furnace and recirculated. Thermocouples placed in the air ducts control the temperature within narrow limits.

As previously stated the mechanical properties are increased by warm working. Test results show that a carbon steel which, in the as-rolled condition, will average 65,000 lb. per sq. in. yield strength and 108,000 lb. per sq. in. ultimate strength with an elongation of 27 per cent

in 2 inches will, after warm working, average 95,000 lb. per sq. in. yield strength, 115,000 lb. per sq. in. ultimate strength with an elongation of 24 per cent in 2 inches. Resistance to collapse, one of the most important considerations in oil-well-casing design, is markedly increased by warm working. For example, 7-inch O.D. casing with a wall thickness of 0.317 inches will have its collapse value increased from an average of 4,370 lb. per sq. in. in hot-rolled pipe to 6,320 lb. per sq. in. after warm working, an increase of 45 per cent.

Spray-Quenched Deep Well Casing—When higher strength casing than that produced by the warm-working process is desired for deep oil wells, carbon-manganese steel pipe is heat treated by a special process developed by National Tube Division. In this process, the hot-rolled pipe is heat treated in a continuous unit by water quenching from 1650° F and tempering at 600 to 1000° F. A series of barrel-type gas-fired furnaces are used for heating the pipe. During heating and quenching, the pipe is rotated. The total length of the eight-barrel continuous furnace (Figure 41—27) is 47 feet, and the rate of heating, depending on pipe size, is 5 to 20 feet per minute. The capacity of this unit may be increased by the addition of more barrel furnaces. After heating to 1650° F, the pipe is quenched as it passes through a specially designed water-spray ring containing nozzles which spray water on the complete periphery of the pipe.

After quenching, the steel pipe is placed in a tempering furnace (Figure 41—28) and heated in a 50-minute cycle to 600 to 1000° F, depending upon the final properties desired. The pipe is rotated during heating in the tempering furnace to maintain straightness and uniformity of heating. The continuous tempering furnace is fired with gas and the pipe temperature is checked with a radiation-type pyrometer as it leaves the furnace.

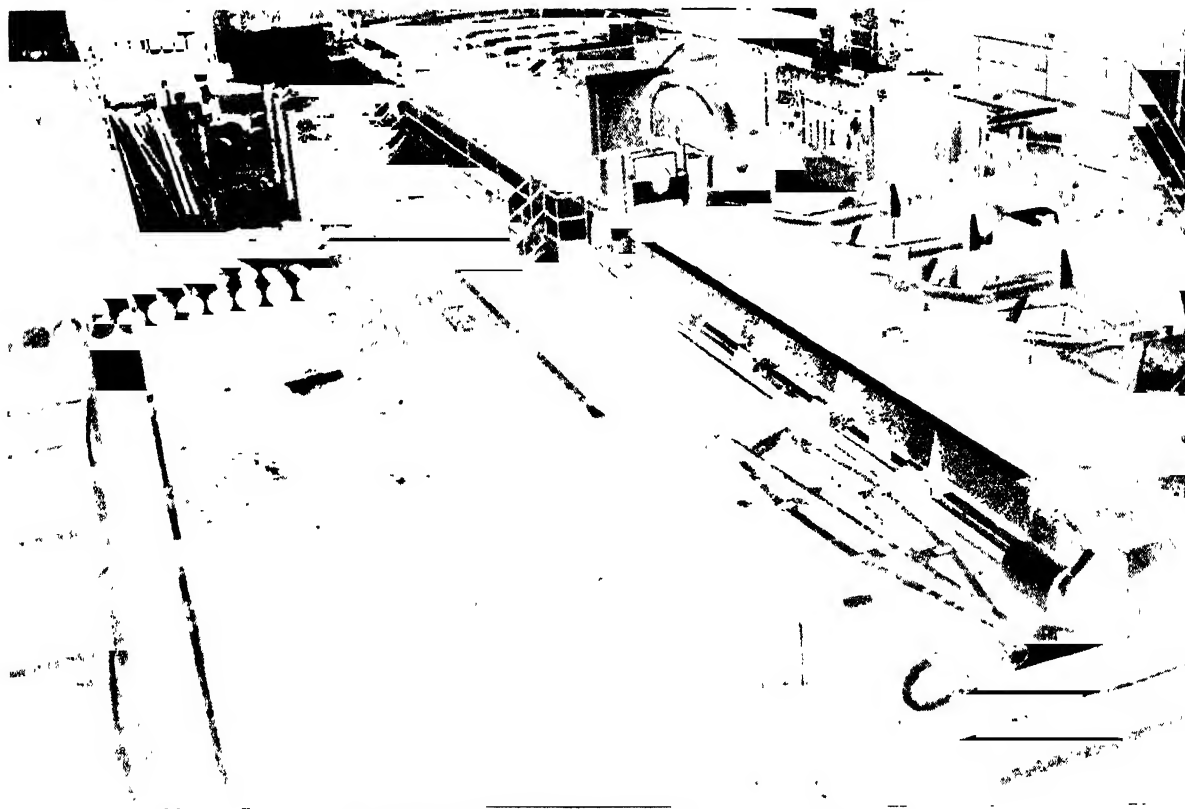


FIG. 41—27. Heating and spray-quenching equipment.

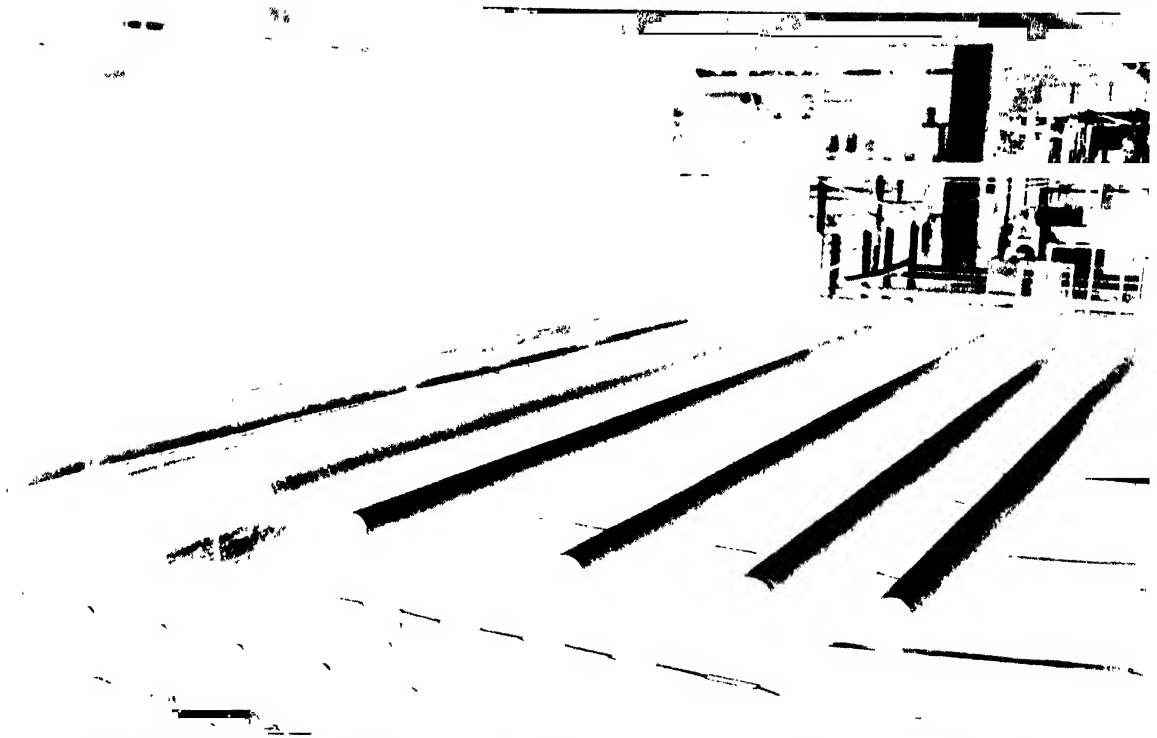


FIG. 41-28. General view of the entry side of a tempering furnace for quenched steel pipe.

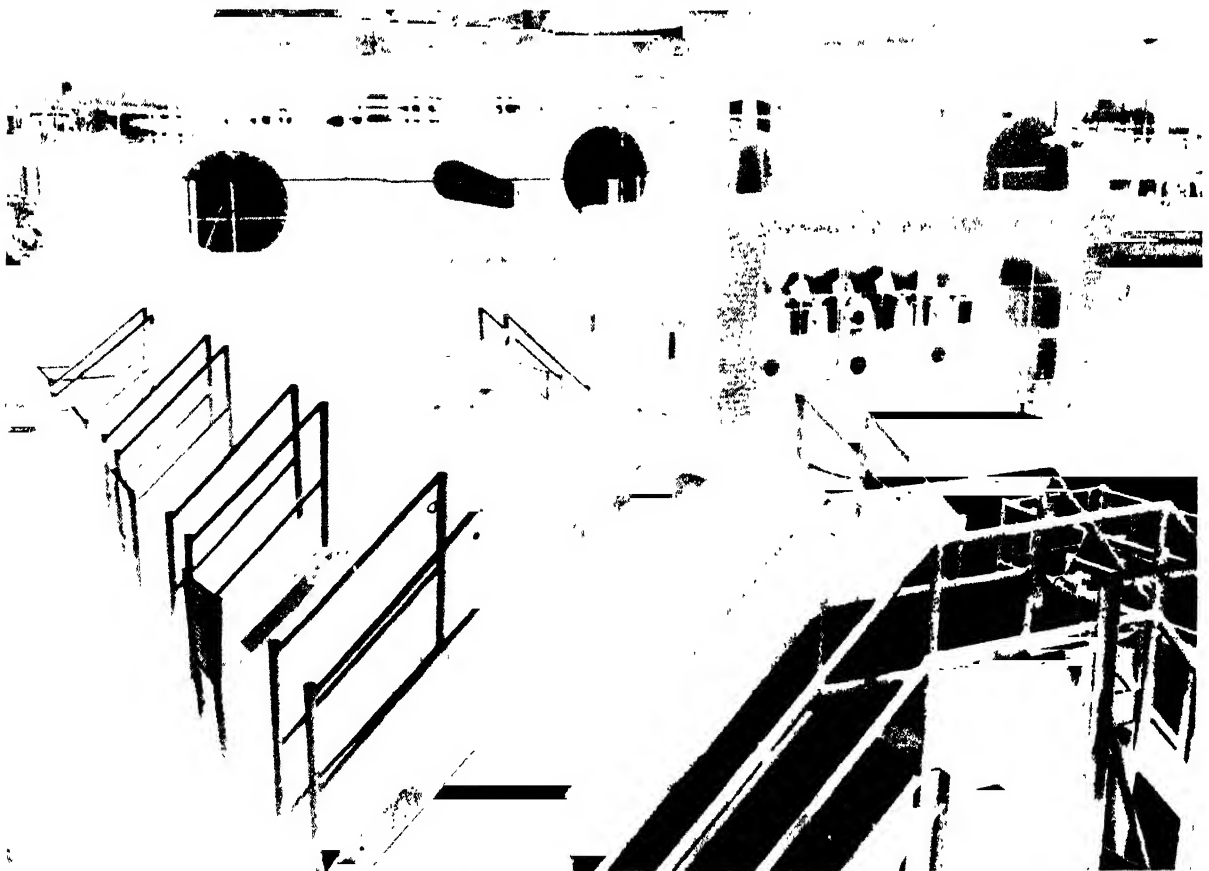


FIG. 41-29. General view of a set of five stands of sizing rolls.

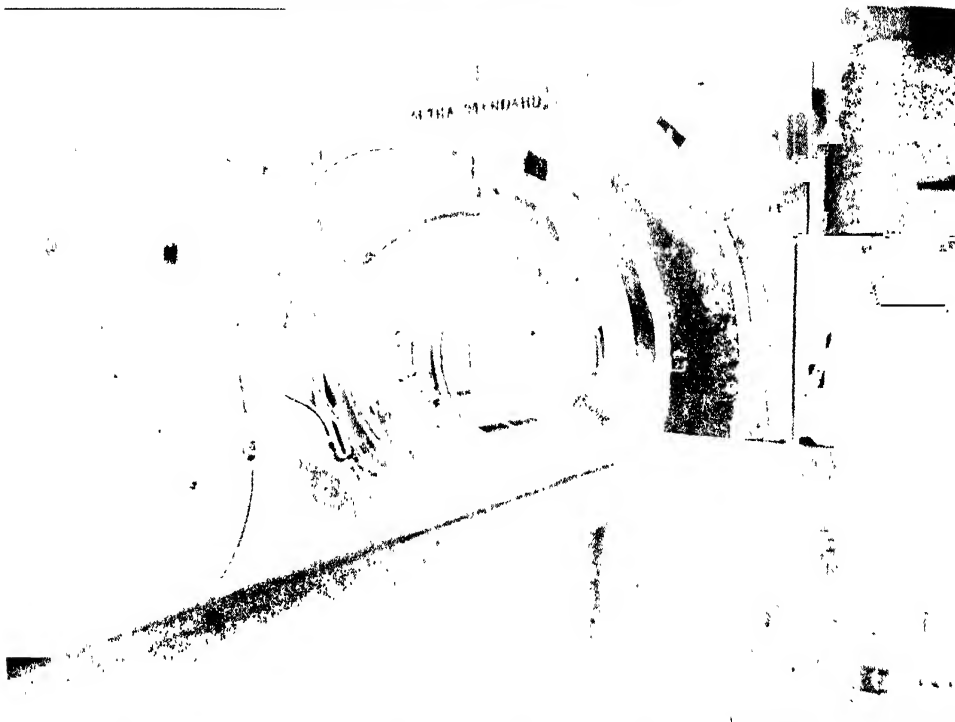


FIG. 41-30. Cut-off machine for the preparation of pipe for the cold-expanding operation.

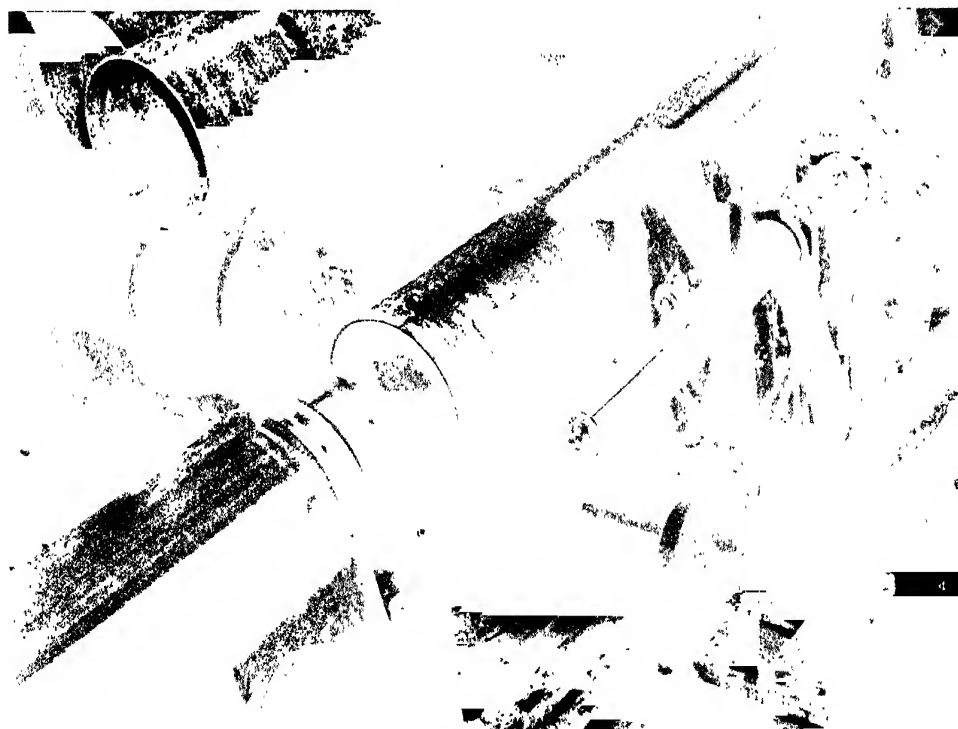


FIG. 41-31. Expander head and guide with pipe in position to be expanded (24-in. outside diameter by 0.500-inch wall pipe).

When the pipe leaves the tempering furnace, it immediately passes through a set of five stands of sizing rolls, shown in Figure 41—29. Due to the formation of martensite during quenching, an expansion of the steel occurs which increases the outside diameter of the pipe, and introduces a limited amount of ovality and out-of-straightness. The sizing operation insures uniform size in the quenched and tempered product; however, the severity of the sizing pass must be limited.

Test results show that a carbon-manganese steel with average hot-rolled mechanical properties of 64,000 pounds per square inch yield strength, 100,000 pounds per square inch ultimate strength and 30 per cent elongation will, after spray quenching and tempering, exhibit average mechanical properties of 123,000 pounds per square inch yield strength, 136,000 pounds per square inch ultimate strength and 23 per cent elongation. A marked increase in collapse resistance also results from this heat-treating process. For example, 7-inch O.D. casing with a wall thickness of 0.408 inch will have its collapse resistance increased from 6,400 pounds per square inch in hot-rolled pipe to 12,290 pounds per square inch after heat treating—an increase of 92 per cent.

Cold-Expanded Seamless Pipe—One of the new developments in the line-pipe industry is the production of seamless cold-expanded pipe in sizes 16 to 26 inches O.D. This pipe presently is being manufactured at the Lorain Works of National Tube Division.

The hot-rolled seamless pipe produced in a Mannesmann-type mill from solid billets up to 12¼ inches in diameter is expanded in a rotary-rolling mill. After the pipe leaves the rotary rolling mill, it enters a pair of cut-off machines (Figure 41—30) which trim both ends. In these machines (the first of their kind in America) the pipe is held stationary as a six-tool cutting head revolves around the end, cutting through from the outside of the pipe. The new machines are controlled automatically by electronic devices.

The pipe is next conveyed to the cold expander shown in Figure 41—31, where it drops into the expander trough and one end is held firmly against a backstop. A 60-foot ram made of 14-inch O.D. seamless pipe is positioned at the opposite end of the pipe and an expander plug is fitted on the bar cap at the end of the ram. The expander plug is then forced through the pipe by a pressure of 300,000 pounds (1500 horsepower).

At the end of the stroke, the backstop opens to permit passage of the plug, which drops from the bar cap onto an elevator. The elevator lowers the plug to a conveyor which carries it back to the entry end. Here another elevator raises it to an automatic aligner which again positions it for fitting onto the bar cap and expanding another pipe. Five plugs of the same diameter are used in sequence to permit uninterrupted operations and cooling of the plugs between use. The expander plugs, made at the Lorain Work's foundry, are built-up discs fitted with nickel-chromium-iron alloy rings. The plugs to produce expanded seamless pipe of 24-inch nominal outside diameter are under 24 inches in diameter and 8½ inches thick. They expand the pipe to its nominal outside diameter. Power is supplied to the ram through an electrically driven gear-reduction unit. Two heavy chains, one on each side, are attached to a cross head to drive the ram. The plug is lubricated through a flexible pipe which feeds from a reel and leads through the ram to the bar cap. Clamps which hold the pipe in the expander are operated by an oil hydraulic system at 2000 pounds per square inch pressure.

After cold expansion, the pipe enters a rotary straightener and then is conveyed to an inspection table where

it is rotated during visual inspection. Two rings are cut from each lot of pipe. One is flattened and prepared for a standard strip tensile test to determine ultimate strength and elongation and the other ring is hydraulically tested for transverse yield strength.

After the pipe ends are beveled, the finished pipe is again inspected for surface and end flaws, after which it is hydrostatically tested to a pressure that exerts a fiber stress equivalent to 85 per cent of the specified minimum yield strength.

Pipe manufactured by this process is cold worked in the transverse direction. The operations involved in cold expanding a seamless pipe about 40 feet long may be performed in 60 seconds, using a ram speed of 175 feet per minute. Depending upon the size, cold-expanded seamless pipe is produced at the rate of about 50 tons per hour. The diameter of the pipe is increased during cold working and, although a water-soluble oil is used as a lubricant, there is a temperature rise in the pipe during expansion. The pipe length is reduced about 1½ per cent, with less than 1 per cent change in wall thickness. Cold working seamless pipe in the transverse direction increases the yield strength to a greater degree than the ultimate strength. Excellent transverse ductility is obtained due to the nature of the seamless process.

The Continuous Seamless Process—The two most recently constructed National Tube Division seamless mills utilize equipment entirely different than that used in the conventional seamless process. The rolling mill and reeler of the conventional mill are replaced by a continuous rolling mill (Figures 41—32 and 41—33) with nine tandem individually powered stands of two-high grooved rolls. Figure 41—32 illustrates the method of reduction employed by this continuous rolling mill. The rolls in the consecutive stands have their axes at 90 degrees to each other and are driven by motors which provide a total of 8500 horsepower. The pipe mill requires an internal mandrel against which the work piece is rolled to reduce wall thickness. This cylindrical mandrel extends entirely through the pierced billet and passes through the mill with the work piece. In the first two roll stands, the diameter of the pierced billet is reduced so that the inner surface is in substantial contact with the mandrel bar. Each of the next two stands makes a reduction in wall over a portion of the circumference, the two jointly completing the first increment of reduction. The next two stands, the fifth and sixth in this mill, make a similar complete reduction but of somewhat less magnitude. The next two succeeding stands (7 and 8) are designed to effect a very slight reduction, the purpose of which is to planish the tube surface. The shape of the tube which has been oval in the preceding stands is changed to approximately circular section in the ninth stand. The rounding up operation effected by this stand frees the inner surface of the tube from the mandrel bar to facilitate withdrawal of the mandrel.

In the operation of the mill, after a billet has been pierced by a conventional Mannesmann piercing mill, a lubricated mandrel, considerably longer than the pierced shell, is inserted and both pass through the rolling mill. The tube and mandrel are then kicked out of the pass line to a stripper which mechanically removes the mandrel (Figure 41—34). The rolled tube is then further processed by one of two methods depending on the desired product.

After withdrawal of the mandrel, the rolled tubes are reheated (Figure 41—35) before being processed in either a sizing mill or a tension reducing or "stretch" mill. The stretch mill (Figure 41—36) which is similar in construction to the continuous rolling mill, consists of twelve two-high roll stands with the individual stands

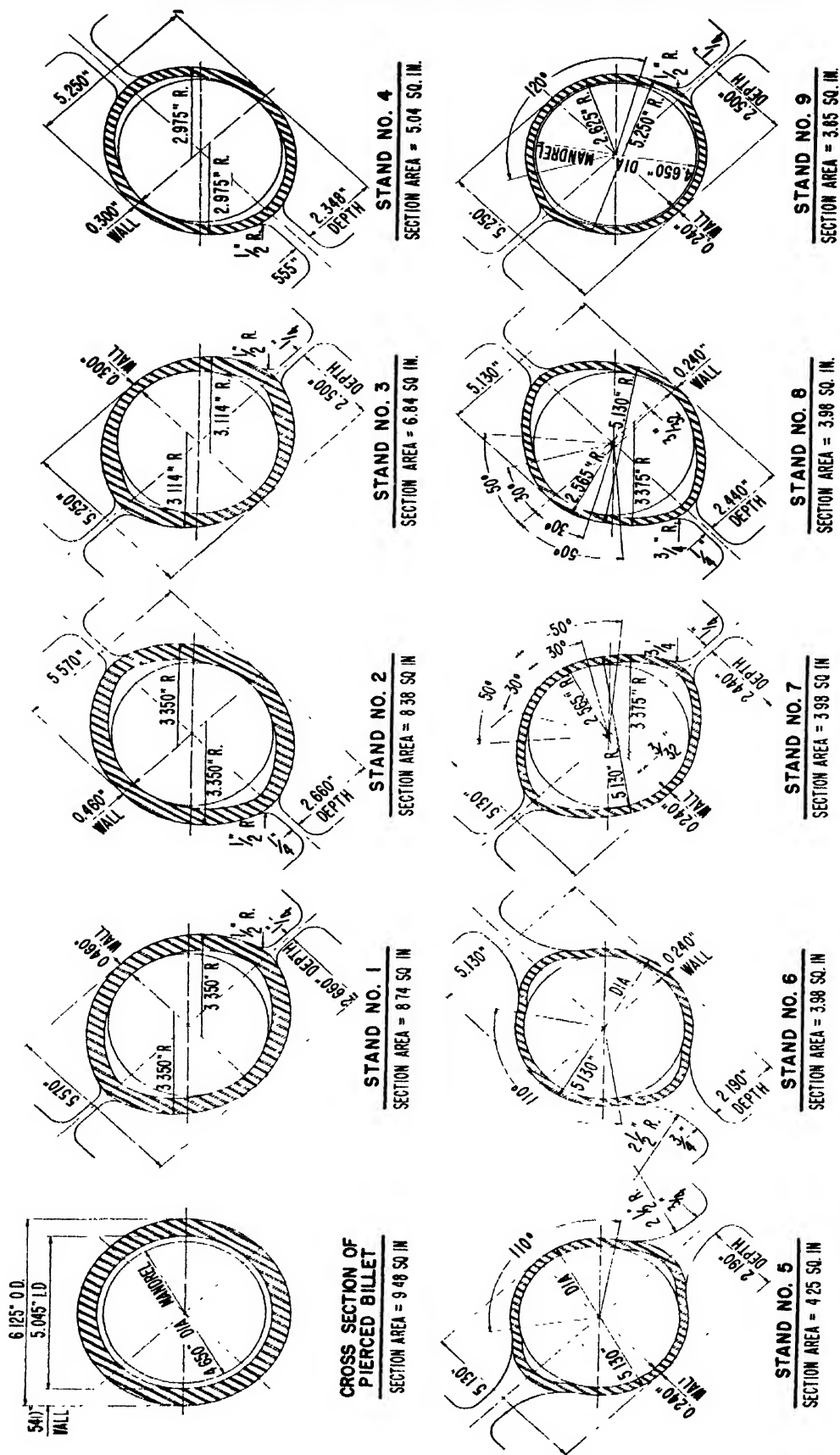


FIG. 41-32. Pass design for a nine-stand continuous tube-rolling mill when rolling 5.250-inch O.D. by 0.240-inch wall pipe shell.

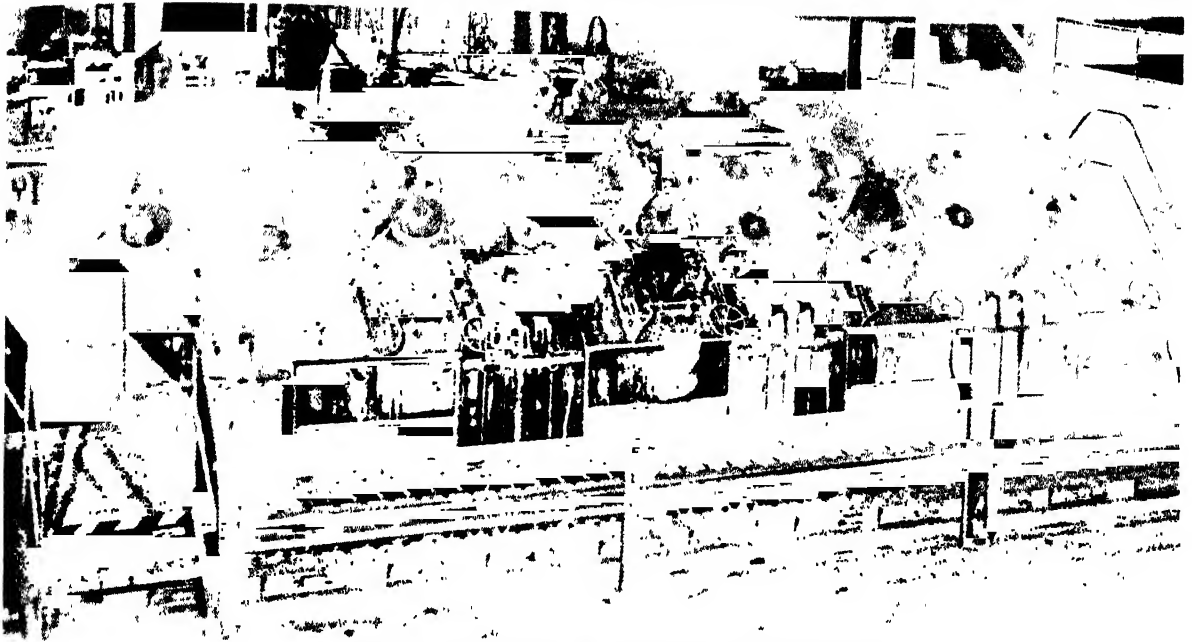


FIG. 41-33. Overall view of a nine-stand mandrel mill.

powered by 200-horsepower motors. Tension reducing, as described in connection with the butt-weld process, is unique in that without the use of a supporting mandrel the wall thickness is diminished while the diameter is reduced. This operation differs from the conventional reducing mill in which the wall thickness of the tube is increased as the diameter is reduced. In the tension reducing mill, the tension forces to which the tube is subjected between roll stands are not only effective in

reducing the wall thickness of the tube but, in addition, make possible in successive stands more than 20 per cent greater diameter reductions than can be made in the conventional mill. The faculty of permitting the entering tube wall to be maintained or reduced while large diameter reductions are being made in successive stands permits a single entering tube size to be employed for the production of all tube sizes in the size ranges up to 3½-inch O.D. This unit, therefore, permits production at



FIG. 41-34. Mandrel bar being removed by stripper conveyor from rolled tube held against stripper block.

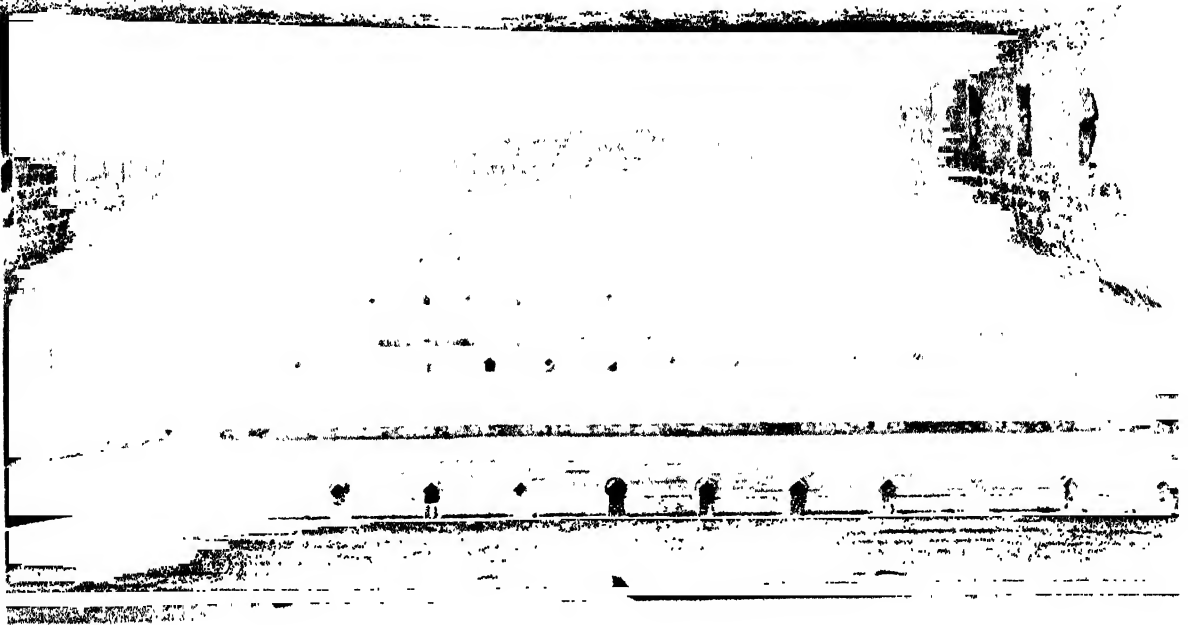


FIG. 41—35. Interior view of a tube-reheating furnace.



FIG. 41—36. Overall view of part of the twelve-stand stretch-reducing mill, with tube-reheating furnace in background.

approximately constant tonnage regardless of the diameter being produced.

To produce 4½-inch O.D. tubing, the maximum size currently being produced on this seamless mill, the rolled tube is sent through a sizing mill instead of the stretch reducing mill.

Having both a sizing and stretch mill permits continuous production since it is possible to divert tubes to either unit depending on the desired size.

The pierced billet which is 5½ inches O.D. by 0.450-inch wall by 22 feet long and pierced from a solid round 5½ inches in diameter by 8 feet long, will emerge from the continuous reducing mill as a tube 5 inches O.D. by 0.200-inch wall which is 62 feet long. This tube, after passing through the stretch reducing mill, if rolled into 2¾-inch O.D. tubing with 0.190-inch wall, will be 140 feet long.

The long tubes are cut into two sections by a rotary

saw on the cooling table. The half sections are then cut into predetermined lengths by high-speed rotary-blade cutters before going to the finishing floors.

Rolling Heavy-Wall Tubing—When rolling heavy-wall tubing, the pierced shell after leaving the piercing mill by-passes the continuous rolling mill and goes directly to an Assel mill (Figure 41—37) which is specifically designed to roll this product. It consists of three equally spaced rolls set at an angle of 8 degrees, which cross roll the pierced shell over a freely revolving mandrel which is inserted before the shell enters the mill. The roll design is similar to the Mannesmann piercer in that there is a converging inlet which grips the billet and rolls it down on the mandrel. The wall thickness is then reduced to an amount determined by the hump in the rolls. The rolls are approximately 13 inches in diameter at the gorge and are driven by a variable-speed motor through one drive. Roll positions are changed by motor-driven screw-downs which are tied together to give equal movement to each roll.

The advantage of this mill is that the tube is rolled over a polished mandrel which provides an inside surface that is smooth and free from scratches. Sizes from 2½-inch to 4½-inch O.D. with wall thicknesses from 0.250-inch to 1¼ inches can be produced. Smaller diameters can be made by further processing the rolled product through a 12-stand reducing mill.

Seamless Fabricating Practices—In the discussion of the various phases of seamless tube manufacture in this section, the role of each unit was described. It was also indicated that the number of operations is dependent on the size of the tube to be produced. To further clarify these discussions, the following examples, which typify the variations employed in the manufacture of seamless tubes, are given to develop the operations step by step.

In producing a 2¾-inch O.D. by 0.154-inch wall single-length hot-rolled tube, a solid billet 3¼ inches in diameter and weighing approximately 82 lbs. is pierced to a shell 3⅞-inches O.D. with 0.215-inch wall, about 11 feet long. This pierced shell is passed, without reheating, to the plug rolling mill where it is plug-rolled in a 3¼-inch groove to produce a tube 3¼-inches O.D. with a 0.140-inch wall and 17 feet, 6 inches long. The plug-rolled shell is then reeled to approximately 3⅞-inches O.D., 0.140-inch wall, and about 16 feet, 6 inches in length. After reeling, the tube is passed through a reheating furnace and into the reducing-sizing mill from which it emerges 2¾-inch O.D. with 0.154-inch wall and approximately 22 feet, 3 inches long. From this tube, the crop-ends, and any test pieces that may be required, are cut in the finishing operations described in a later section.

In producing a double-length hot-rolled tube 8¾ inches O.D. with 0.277-inch wall, a solid billet 8¼ inches in diameter, 6 feet, 5 inches long, weighing 1166 lbs. is pierced in the first piercer to a shell 8 inches O.D., wall thickness of 1¼ inches and 12 feet, 10 inches long. Without reheating, this shell is further processed in the second piercer to a shell 8¾ inches O.D., with 0.470-inch wall, and approximately 28 feet long. The shell from the second piercer, after reheating, is plug-rolled in an 8⅞-inch groove to 8⅞ inches O.D., 0.277-inch wall and 46 feet, 9 inches long. The plug-rolled tube is transferred directly to the reeling machine which produces a tube 8¾ inches O.D., 0.277-inch wall, and 45 feet, 5 inches long. The tube then receives two or more passes in the two-high sizing mill from which it emerges 8¾-inch O.D. with a 0.277-inch wall and approximately 46 feet long. It is then ready for crop-ending and the other finishing operations.

To produce 26-inch O.D., 0.303-inch wall double-

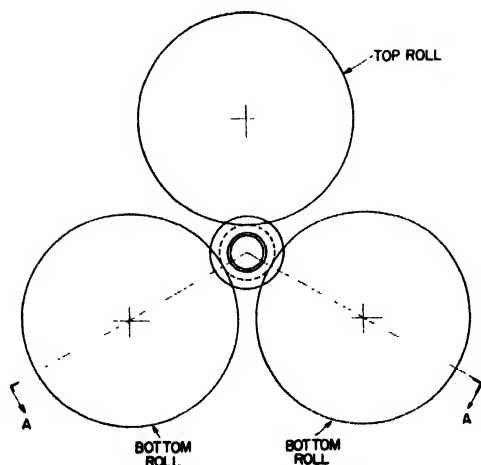
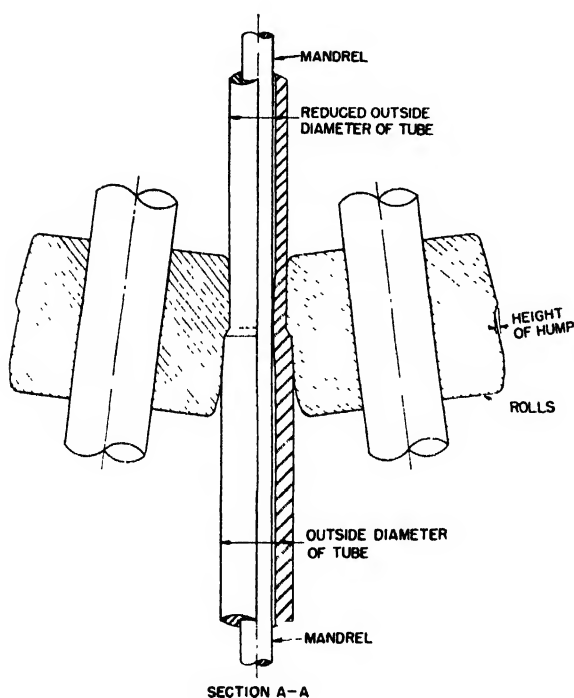


FIG. 41—37. Roll design. Assel mill.

length hot-rolled tubes, a solid billet $12\frac{1}{4}$ inches in diameter, 10 feet long and weighing 4007 lbs., is pierced to a shell 14 inches in outside diameter, 1.640-inch wall, and 18 feet, 3 inches long in the first piercer. This shell, in the same heat, is entered in the second piercer where it is rolled to 17 inches O.D., 0.750-inch wall, and 30 feet, 2 inches long. After reheating, the shell is plug-rolled to form a tube $16\frac{3}{4}$ inches O.D., 0.500-inch wall, and 44 feet, 9 inches long. The plug-rolled shell is reheated a second

time, after which it is rotary rolled to $26\frac{1}{4}$ inches O.D., 0.303-inch wall, and 45 feet, 8 inches long. Without reheating, it is then reeled to $26\frac{1}{2}$ inches O.D., 0.303-inch wall, 45 feet, 3 inches long. The tube, after reeling, is again reheated in the tunnel-type reheating furnace, after which it passes through two stands of two-high sizing-mill rolls, forming a hot-rolled tube 26 inches O.D., 0.303-inch wall, and 46 feet long, ready for the finishing operations.

SECTION 6

COLD-DRAWN, OR COLD-FINISHED, TUBES

While the ordinary requirements for pipe and tubing can be met by the hot-rolling processes just described, there are many requirements that demand greater accuracy, higher physical properties, better surfaces, thinner walls and smaller diameters than can be produced by hot-working methods. This demand is met by cold drawing the hot-rolled tubes as a finishing operation. This phase of tube production is analogous to the drawing of wire and the cold rolling of sheets described in preceding chapters.

In the early history of seamless tube production, cold drawing was employed in almost every instance as a finishing operation, but, as the art of hot rolling developed, the necessary finish and dimensional requirements have been met to a greater and greater degree in the hot-rolling operation. However, a substantial proportion of tubing is still cold drawn for the following reasons:

1. To produce tubes with thinner walls than can be hot rolled.
2. To produce tubes with smaller diameters.
3. To produce tubes longer than can be hot rolled in certain sizes.
4. To secure better surface finishes.
5. To obtain closer dimensional tolerances.
6. To increase certain mechanical properties, such as tensile strength.
7. To produce shapes other than round.
8. To produce tubes with varying diameters and wall thicknesses from end to end.
9. To make small lots of tubing of odd sizes and gages that do not justify a hot mill run.

One and five-sixteenth inches is about the minimum diameter it is practicable to produce by hot rolling, and 0.083 inch is the thinnest wall of commercial hot-rolled tube, which is available in the smaller diameters only. The diameter and wall thickness range of cold-drawn tubing produced by the National Tube Division is from $\frac{1}{2}$ -inch O.D. by 0.035-inch wall to $10\frac{3}{4}$ -inch O.D. by 2-inch wall.

The fact that hypodermic needles are seamless steel tubes that are cold drawn from hot-rolled tubing in small specialty plants conveys the idea of how far seamless tubing can be reduced by cold drawing.

Principle of Cold Drawing—Tube drawing is essentially the same as wire drawing, as explained in preceding chapters with two important exceptions: viz., the inside diameter of the tube must be supported while it is passing through the die to effect wall reduction and control the size of the hole, and comparatively short lengths are involved as in bar drawing. The process consists of pulling the tube through a die, the hole of which is smaller than the outside diameter of the tube being drawn, and at the same time supporting its inside surface by a mandrel anchored on the end of a rod so that it remains in the plane of the die during the drawing operation. Figure 41—38 shows the operation. The mandrel may be omitted if it is not necessary to make a reduction in the wall thickness, or if the dimensions and surface of the inside are not important. A modification of this method consists of drawing on a bar rather than over the mandrel, in which method the bar travels through the die with the tube and must be removed later. The resistance of the metal to passage through the restricted space between the die and the mandrel exceeds the yield strength of the metal at this section, thereby resulting in plastic flow. As a result of the reduced metal section leaving the die, the velocity is increased, the amount of increase being dependent upon the cross-sectional reduction. It is evident that the reduction or draft may be increased only up to a certain limit, depending on the ultimate strength of the section leaving the die, because, if the resistance and balancing pull exceeded this stress, the section leaving the die would break. In drawing tubes over a stationary mandrel, the maximum practical sectional area reduction does not exceed 40 per cent per pass, while, in drawing tubes on a bar which is free to move with the tube, the area reduction can be 50 per cent. As in wire drawing, every inch of metal passing through the die is subjected to stress almost up to its breaking strength and is thus given a test of its physical

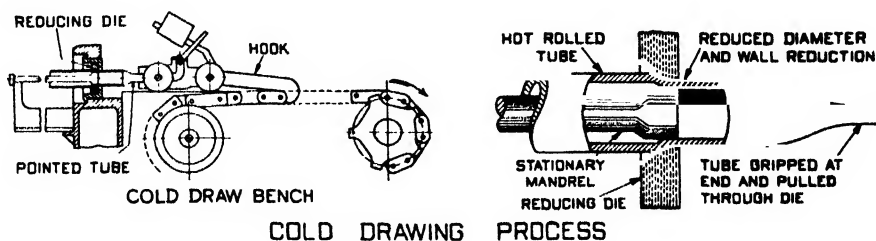


FIG. 41—38. Cold drawing a seamless steel tube.



FIG. 41—39. Draw bench with tube going through die.

fitness to withstand high stresses in its ultimate service. Any flaw or defect of consequence is brought to light under this severe treatment.

The Draw Bench—A cold-draw bench for tubes consists of a heavy steel frame or bench, in the middle of which is located a die head for holding the die. At one end of the bench is located an adjustable holder to anchor the mandrel rod. At its other end a shaft is mounted carrying a sprocket wheel over which passes a heavy, endless, square-linked chain. This chain lies in a trough on top of the bench, which extends from the sprocket wheel to the die head, where the chain passes around an idler and returns underneath the bench to the sprocket wheel. The sprocket wheel is driven by a variable speed motor through suitable reduction gearing. A carriage called a **plyer** runs on tracks on the top of the bench and over the chain that lies in the trough between the tracks. This plyer is equipped on one end with jaws to grip the tube and on the other end with a hook to engage a link of the draw chain. The plyer is connected by cable to a motor-actuated drum by which means it is returned to the die head after drawing a tube. The jaws grip the reduced or pointed end of the tube which projects

through the die about 6 inches. The closing of the jaws is effected by the motion of the hook in dropping into engagement with the chain. The whole action of gripping the tube and engaging the chain is automatic, once the operator pushes a button to return the plyer to gripping position. The mandrel-rod anchor is equipped with an air cylinder to push the mandrel into operating position inside the die after the pointed end of the tube is inserted in the die. Draw benches for small tubes are equipped with two mandrels with their supporting rods, so that a tube is being loaded on one mandrel while another tube is being drawn off the other. A motor-driven indexing mechanism places the anchor in the pass line so that the mandrel rod is in perfect alignment with the die and plyer jaws prior to the beginning of the draw. A friction-roller mechanism automatically loads the tubes on the mandrel rods. All controls for the draw benches are grouped at the operator's position near the die head so he does not have to move away from that position when operating the bench.

The total length of a bench is about 80 to 100 feet. The capacity of draw benches may be from 50,000 to 300,000 pounds pulling power. Chain speed may vary from 20

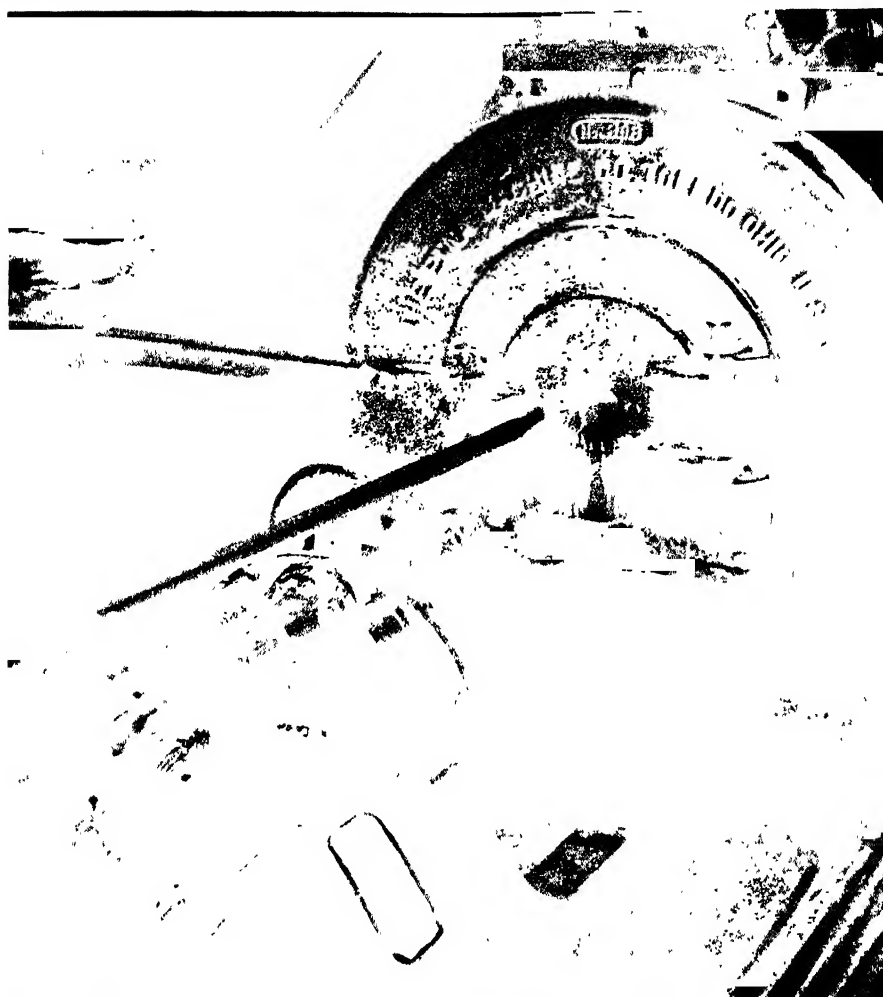


FIG. 41—40. Rotary tube pointer and feeder mechanism.

to 150 feet per minute and is automatically controlled so that the tube is started through the die at a slow speed, and, as soon as it is fairly started, the speed increases to the predetermined drawing rate. Dies are made with a conical outer surface which fits in a holder mounted in the die head. Dies up to a 3-inch hole size are made with tungsten-carbide inserts or nibs. Larger dies are made from hardened tool steel, chrome-plated on the wearing surface. Mandrels are chrome-plated, hardened tool steel and are either made in the form of a bar from 6 inches to 12 inches long with one end upset to form the working surface and the other end tapped for connection to the mandrel rod (see Figure 41—38) or, in the case of larger mandrels (2 inches and over), they are made in disc form with a central hole for engagement with the mandrel rod. In the drawing-on-the-bar method, hardened and ground bars of a diameter to correspond with the inside diameter of the drawn tube and somewhat longer than the drawn tube are used. As the bar-removal operation requires some time, three or more bars constitute a set, which permits the drawing of a tube during the interval required for inserting a bar in the next tube and extracting the bar from the tube previously drawn.

Preliminaries to Cold Drawing—The hot-rolled tubes after cooling are pointed on one end. This pointing consists of reducing the outside diameter, for a distance of about 6 inches, sufficiently to permit the reduced portion

to enter the hole in the draw die freely, so that the jaws of the pincer can grip this end of the tube. If more than one cold-draw pass is to be given the tube, the point is made slightly under the final die size, if possible. Where large diameter reductions are made on small tubes, a point that would enter the final die may be too small and weak to stand the earlier reduction, in which case the original point may be further reduced after a few passes or, in the case of very small tubes, the pointed end may have to be cut off and a new point made on the reduced diameter and wall section. This procedure may have to be repeated several times in drawing very small tubing. Tubes with diameters $2\frac{1}{2}$ inches to 3 inches and over are usually open pointed with a reduced section, 2 inches or 3 inches long, which has a rather sharp shoulder or offset joining the two diameters. A pulling pin with a cylindrical head is inserted in the tube so that the head engages the shoulder of the point and the stem of the pin projects through the die far enough to enter the pincer jaws. Pointing is done on rotary swagers or steam or air hammers after the end of the tube has been heated to a forging heat (see Figure 41—40). Tubes of certain grades of steel receive an annealing heat treatment, prior to cold drawing. This treatment, which is usually confined to high-carbon and alloy steels of the air-hardening type, is necessary to obtain additional ductility. All tubes are pickled in dilute sulphuric acid to remove scale and oxides from the outside and inside



FIG. 41—41. Pickling tubes.

surfaces. The pickling practice is quite similar to that employed in preparing wire for drawing or sheets for cold rolling. The pickle tubs are 40 to 50 feet long, about 4 feet wide, and 5½ feet deep. They are usually built of reinforced concrete and lined with vitrified brick laid in a sulphur compound mortar (Figure 41—41). Steam for heating and agitating the bath is led into the bottom of the tubs and discharged through jets or perforated acid-resisting pipes. A wash tub of similar construction is located near the pickle tubs, and after the tubes are pickled free of scale they are dipped in the wash tub to remove the acid and any sludge or loose scale. A similar tub is provided for lubricant for cold drawing, in which great care is exercised in dipping to assure that lubricant reaches the entire interior surface of each tube. This lubricant, which is an emulsion containing flour, tallow, and water in proper proportions, is used for process-drawing all tubes and for finish-drawing large-diameter tubing which requires a heavy reduction. Where tubes with bright finish are required, a special oil-base emulsion is used. After being thoroughly coated with lubricant, the tubes are placed on a conveyor chain on which they are carried through a continuous gas-fired drying oven (about 200° F) where the excess lubricant is drained and the moisture is evaporated from the lubricant. Throughout these operations the tubes are handled in bundles up to 10 tons in weight. From the drying oven the tubes are then sent to the

cold-draw benches or to stock piles near the benches.

The Cold-Drawing Operations—One cold-draw pass produces a cold-drawn tube of close dimensions, good surface, and of any mechanical property within the usual limits of cold-worked steel. Additional passes may be necessary to secure: (1) thinner walls, (2) better surface finishes, (3) smaller diameters, or (4) longer lengths. The bundle of pointed, pickled, inspected and lubricated tubes is laid on a table at the chain end of the bench, with open ends toward the mandrel-anchor end of the bench. The mandrel on its supporting rod is held so that the powered pinch rolls, into which a tube has been entered, drive the tube over the mandrel and rod until the mandrel is just back of the pointed portion. The operator then swings the tube into the pass line and operates the control for the push-up which advances the tube and mandrel to drawing position. The cable-driven pleyer-return mechanism moves the pleyer carriage toward the die head, in a direction opposite to that in which the draw chain is moving, until the carriage contacts a limit which releases the cable mechanism and permits the hook, which has been elevated, to drop into engagement with the draw chain. The engagement of the hook with the chain permits the grip jaws to close on the projecting point, and the motion of the carriage draws the tube through the die. When the open end of the tube clears the die, the pleyer hook automatically disengages from the chain. Another tube

is loaded on the mandrel and the cycle is repeated as before. When, after a series of draw passes, the inside diameter of the tube becomes too small to accommodate a substantial mandrel, the tube may be further reduced in both its outside and inside diameters by sinking it through a die without a mandrel or bar on the inside. Such a reduction results in a thickening of the tube wall which is uniform and predictable; however, the inside diameter cannot be held to the close tolerances produced with the use of a mandrel or bar support.

The Tube Reducing or Rockrite Process is used for the same purposes as conventional cold drawing; i.e., to produce tubing with smaller diameters and lighter walls, closer tolerances, better surface finish, higher mechanical properties, better machinability, etc., than may be obtained by hot rolling. Longer lengths than obtainable by either hot rolling or cold drawing—up to 100 feet or more—are possible, but such long lengths, especially in the smaller diameter range associated with Rockrite material, may not be conveniently handled through finishing operations, hence little use is made of this feature. Production is generally confined to the smaller-diameter tubes although the upper limit is established only by capacity of existing machines. The Rockrite process has a slower production rate in feet per hour than conventional drawing, but accomplishes much greater reduction per pass so that the net effect is more or less equivalent production rates, particularly where a Rockrite machine performs the work of three or more cold-draw passes, as is usually the case. Advantages of the Rockrite process over conventional drawing include improvement in the concentricity of the product, and elimination of certain intermediate operations (such as cutting, pointing, annealing, pickling, doping, etc.) required by the successive cold-draw passes used to accomplish the same total reduction; less material waste is also experienced because there is no point-crop to discard. Disadvantages include slower production rates than conventional drawing where small reductions are involved, and the time required for a size change is

much greater. Additionally, the relative die costs of the two processes greatly favor conventional drawing when small production lots of odd sizes are to be made.

Principle of Rockrite Process—The Rockrite process accomplishes simultaneous reduction of tube diameter and wall thickness by a cold-swaging action which utilizes compressive forces rather than the tensile forces employed in conventional drawing. This fact removes the reduction limitation present in the cold-draw process which is governed by the ultimate tensile strength of the reduced section. In the Rockrite process, very large reductions in one pass are found feasible—as much as 85 per cent or more; this may be compared with the 40 per cent reduction per pass which is the practical upper limit in the drawing process. This comparison is even more striking when it is understood that four successive passes of 37.8 per cent reduction each are required to effect a total reduction of 85 per cent.

Of fundamental importance in the Rockrite process are two semi-circular dies which have matching, tapering, semi-circular grooves machined into their curved faces. In operation, one die is placed on top of the other so that the matching semi-circular grooves make a circular pass. The dies are geared to each other in such a fashion that they rotate in opposite directions when they are moved laterally (in the plane of the grooves), and a converging (or diverging, depending on the direction of lateral movement) circular pass is traced by the die grooves. When a tube is held stationary on the center line of this pass, the converging path of the die grooves reduces its diameters. If now a stationary mandrel of the proper taper is also positioned in the center line of the pass, the inside of the tube is supported and the tube wall is reduced and extruded or elongated by compression between the die and mandrel. In actual practice, the dies are in constant, lateral and rotary, reciprocal motion. Modifications in die contour permit repetition of the swaging action to reduce an entire tube of given diameter and wall thickness to one of smaller diameter and lighter wall thickness, with the length being in-

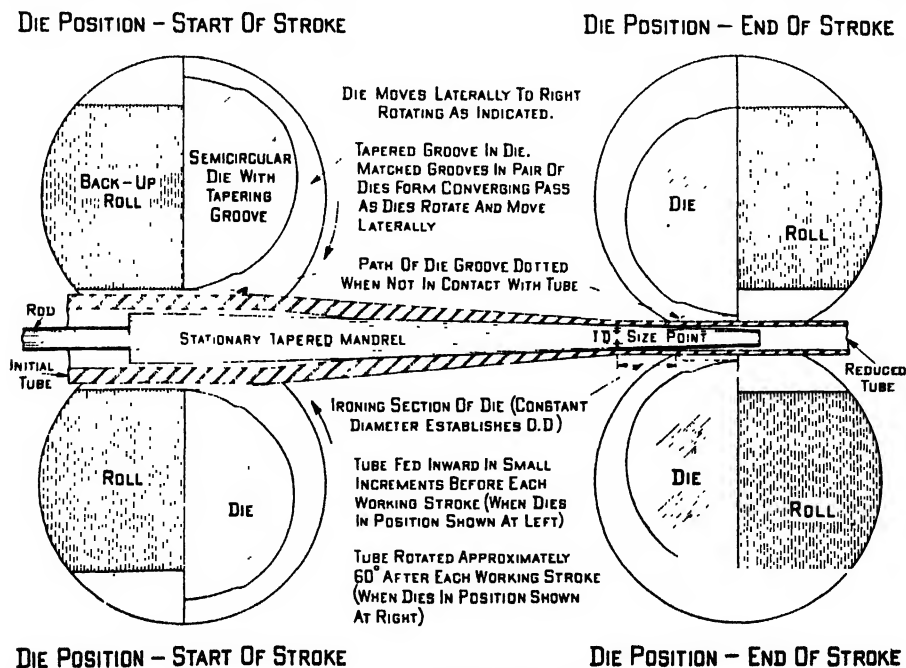


FIG. 41—42. Vertical section through tube reducer pass—showing dies at start and end of stroke.

creased in direct proportion to the amount of reduction in a cross-sectional area. Such modifications include contour-relief at both ends, and an ironing section of constant diameter at the smaller end. Contour-relief at the inlet or large end of the die grooves permits feeding the tube, in small increments, into the pass during the short interval when the dies are not in contact with the tube. Contour-relief at the outlet or small end of the die grooves permits rotation of the tube during the corresponding time interval at the end of the working stroke; this rotation serves to prevent the formation of fins and to round up the finished section. The constant-diameter ironing section is of sufficient length to permit several working strokes on each increment of reduced tube, and serves to size and round up the reduced tube. The inside diameter of the finished tube is governed by the diameter of the mandrel at the point located directly beneath the beginning of the ironing section of the dies; this point is called the "size-point." It is obvious that minor changes in I.D. size may be made by axial movement of the mandrel. O.D. size may not be altered so simply, requiring regrinding and refinishing operations in existing dies.

The Rockrite Machine—Rockrite machines are classed as continuous or intermittent, but the production mechanism is essentially the same in the two types. The machine proper consists of a heavy steel frame, some 30 to 40 feet long and 8 to 10 feet wide. The forward end of the frame contains the mechanism which actually swages the tubes. This mechanism, called the "saddle," is a housing containing the rolls into which the dies are keyed, and which slides back and forth on ways in short reciprocating strokes, being driven by connecting rods attached to cranks geared to the 40 to 100-horsepower main-drive motor. A stationary rack attached to the frame engages one of the intermeshed gears on the roll ends, and thus produces reciprocating rotation of the dies as the "saddle" moves back and forth. The tapered mandrel is located in the desired position in the pass by a long mandrel rod running back, inside the tube being processed, to a thrust block located at the rear of the machine frame.

The tube being processed is gripped at its back end in a vise carried in the "cross-head"—this last being the tube feeding and turning mechanism. A long screw—running from beyond the back end of the machine, through a suitable drive-nut in a stationary housing, up to the "cross-head"—advances the latter a predetermined amount during the time interval at the end of each back stroke of the "saddle" when the tube is not

contacted by the dies. Similarly, a splined shaft—running from the back to the front end of the machine frame, and passing through the "cross-head"—rotates the "cross-head" vise, together with the tube it grips, approximately 60 degrees during the corresponding period each time the "saddle" reaches the forward end of its stroke. This splined shaft also drives an outlet friction vise which serves to rotate the tube being processed when its back end is no longer gripped in the "cross-head" vise.

The chief difference between the continuous and intermittent machine types is the method of charging a new tube. In the so-called continuous machine, two mandrel-rod locks, or thrust blocks, located some 20 feet apart, are employed. In use, assuming both locks closed and the machine in operation, the rear lock is opened and a new tube is charged over the end of the mandrel rod up to the forward lock; the rear lock is then closed and the forward lock opened, allowing the tube to be moved forward until it touches the end of the tube being reduced—in which position it remains to be gripped by the "cross-head" on its return to initial position. Suitable automatic controls open the "cross-head" vise at the forward end of its travel, actuate a high-speed return motor, and then close the vise again when the "cross-head" has returned to its initial position at which time the regular feed stroke is resumed. The "saddle" remains in motion during this entire cycle and the production time lost is negligible. In the intermittent machine, when the "cross-head" reaches the forward end of its travel, the "saddle" is stopped, the "cross-head" vise opened, the mandrel-rod thrust block unlatched, and high-speed return motors actuated to return the "cross-head," and back out the mandrel rod and mandrel. A new tube is then placed in position, the mandrel rod and mandrel returned to position and the mandrel-rod thrust block locked, then the "cross-head" vise is closed to grip the tube, and finally the "saddle" is restarted in motion. Since reduced tubes are too long to handle conveniently through finishing operations, a flying saw has been devised to cut shorter lengths as the reduced tube emerges from the machine.

Preliminaries to Rockriting—Hot-rolled tubes, or those which have been previously cold-drawn or Rockrited, may be cold-reduced by the Rockrite process. Tube preparation generally first involves an annealing operation to insure the ductility required by the swaging action; some plain low-carbon steels are ductile enough as received from the hot mill, but hot-rolled alloy steels and all previously cold-drawn or Rockrited tubes must

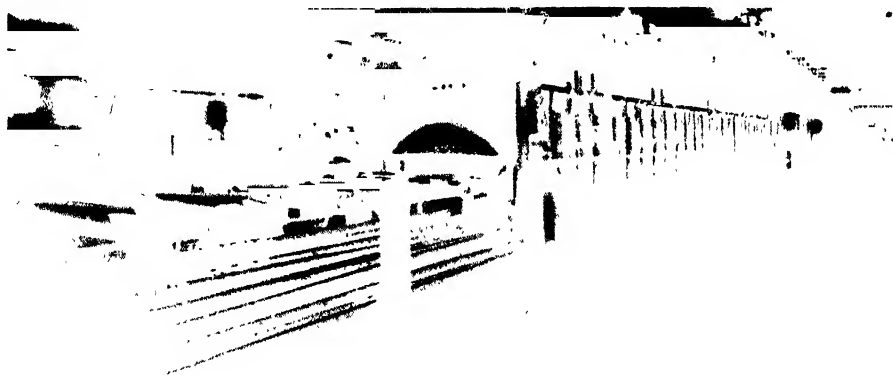


FIG. 41—43. Continuous annealing furnace.

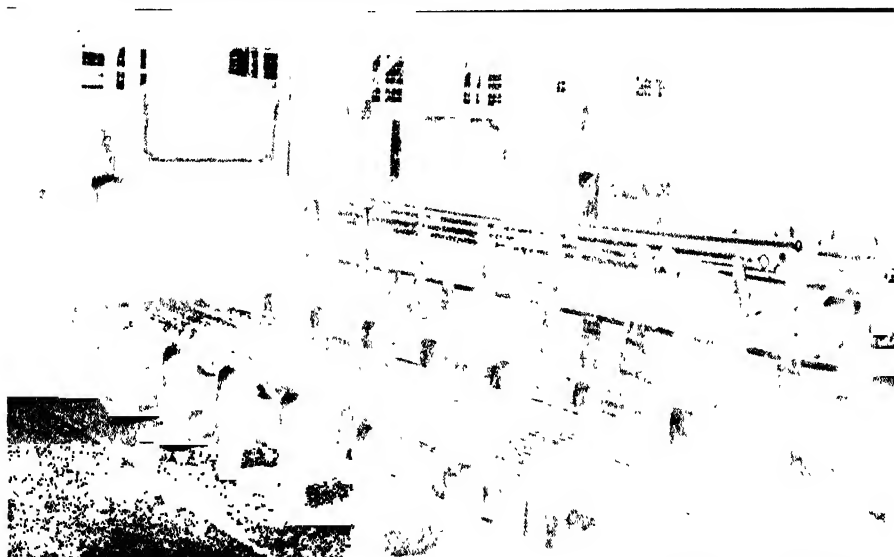


Fig. 41—44. Car-bottom annealing furnace.

be annealed. Tubes are next pickled and straightened, then both ends cropped square with the tube body, and finally they are coated with an oil base or flour-tallow-water lubricant by immersing horizontally in the desired solution—this last operation is more commonly referred to simply as “doping.” The pickling operation is employed to remove mill scale or annealing scale, and utilizes an inhibited sulphuric-acid solution, the same as employed in preparing tubes for cold drawing. The straightening operation is performed in a rotary straightener, and may be preceded by a so-called “break-down” pass through a press straightener when excessive camber necessitates this operation. Tube ends must be cut square because the trailing tube is used to feed the one preceding it when the forward tube is no longer gripped by the “cross-head” vise.

Rockrite Machine Operation—No more than one operator is required for each Rockrite machine, and in some instances one operator may handle two or more machines. The principal duty of the operator is charging new tubes and disposing of reduced material. In the case of intermittent machines, he must also start and stop the “saddle,” and manually open and close the “cross-head” vise. A coolant solution is constantly poured onto the dies during the motion of the “saddle,” and the operator must control its flow and be sure that it is adequate and continuous.

The number of strokes per minute made by the “saddle” is variable within limits (say 70 to 140 per min.), but this is generally fixed by supervision—depending on

the steel grade being processed, amount of reduction, and rate of in-feed. The in-feed is also variable—from 0 to $\frac{1}{16}$ inch or more per stroke—but this is similarly controlled by supervision—depending on the steel grade being processed, the amount of reduction, and the class of product being manufactured. The amount of reduction is governed by the die and mandrel contour, and, except for minor mandrel adjustment (which alters the size of the finished product) is beyond the control of the operator. The stroke of the “saddle” is constant for any given machine size, and increases from about 15 inches to 2 feet and longer as the rated capacity of the machine increases.

Annealing and Redrawing—While a large proportion of tubes receive only one cold-draw pass, many require a number of passes for reasons previously noted. Because cold drawing hardens and reduces the ductility of tubes, it is necessary to anneal them after each cold-drawing operation. Before further cold-drawing, the annealed tubes must be pickled and lubricated as previously described. All tubes, except bright-finished mechanical tubes, receive a final anneal or heat treatment after the last cold-draw pass. Many tubes receive a special normalizing treatment before the last pass in order to obtain the proper grain structure in the finished tube. This annealing is performed in either continuous tunnel or car-bottom batch furnaces fired with gas. The continuous furnaces are provided with heat-resisting driven rolls spaced about 3 feet apart, on which the tubes are carried through the furnace at a predetermined rate depending on the tube section, annealing

Table 41—III. Approximate Mechanical Properties of Hot-Rolled, and Cold-Drawn Low-Carbon Steel Tubing. Steel Specification AISI-C-1015

	Yield Strength (lb. per sq. in.)	Ultimate Strength (lb. per sq. in.)	Elongation (Per Cent)	Hardness	
				Rockwell	Brinell
Hot Rolled	33,000	55,000	40	B-64	107
Normalized	35,000	50,000	40	B-57	97
Soft Annealed ...	30,000	48,000	40	B-50	...
Medium Annealed	40,000	65,000	30	B-73	128
Finish Annealed .	55,000	75,000	20	B-81	149
Hard Drawn	65,000	80,000	15	B-84	159



Fig. 41—45 Abramsen rotary straightener.

temperature, time at temperature, etc. (see Figure 41—43). The car-bottom batch furnaces are arranged in a battery and are served by a special charging crane. Less than two minutes is required to discharge a 5-ton batch of annealed tubes and recharge the furnace with a new batch. The furnaces will accommodate tubes up to 50 feet long (see Figure 41—44). Both types of furnaces are fully equipped with recording pyrometers. Each furnace is divided into four zones, and the temperature is automatically controlled in each zone. When extreme softness or freedom from scale or both are requested, the tubes are enclosed in a heat-resisting sheet-metal box and annealed in a car-bottom batch furnace. The final anneal or heat treatment is varied to produce tubing with the desired mechanical properties. Table 41—III illustrates the effect of this heat treatment on AISI-C-1015 steel. The mechanical properties shown are representative expected values for all but the soft-annealed condition, where they reflect the approximate softest condition expected.

Finishing Operations on Cold-Drawn Annealed Tubes

After the final heat treatment, the cold-drawn tubes are finished in preparation for shipment. The principal finishing operations consist of straightening, cutting, inspecting and testing. Straightening is performed on various types of straighteners, viz., press, rotary, continuous and post. The rotary type consists of rolls set with axes oblique with the pass line (see Figure 41—45). These rolls are somewhat smaller in the center than at the ends to afford a line contact with the tube, which passes

between the two driven rolls on one side and the three idle rolls on the other. The tube is helically advanced by the rolls which are adjusted to bend the tube progressively as it moves through the machine. Initial bends in the tube are removed and a straight tube is produced. Care must be exercised to have the roll setting adjusted accurately, especially on light-walled tubing, to prevent crushing the tube. Some very thin-walled tubes can be straightened successfully only on a post set in the floor carrying grooved blocks which support the tube while it is sprung into straightness by hand. Where exact straightness is required, as in some mechanical tubes, gag press straighteners are used. Operators of these presses develop great skill in giving the tubes the proper deflections between the supporting dies so that the desired amount of permanent set remains. A proving table, on which the tube rotates, is equipped with dial gages, which provide a quick and accurate means of determining straightness. Each tube is spark tested as a final check to identify it as made of steel having the composition specified. The hardness is established by Rockwell or Brinell tests, and tension tests of samples indicate the mechanical properties. Microscopic examination of the grain structure is an essential phase of the inspection program. Each tube is given a visual inspection for surface defects, both inside and outside. Pressure tubing is hydrostatically tested to internal pressures in excess of service pressures that may be encountered. The usual range of test pressure is from 1000 to 5000 pounds per square inch. On pressure

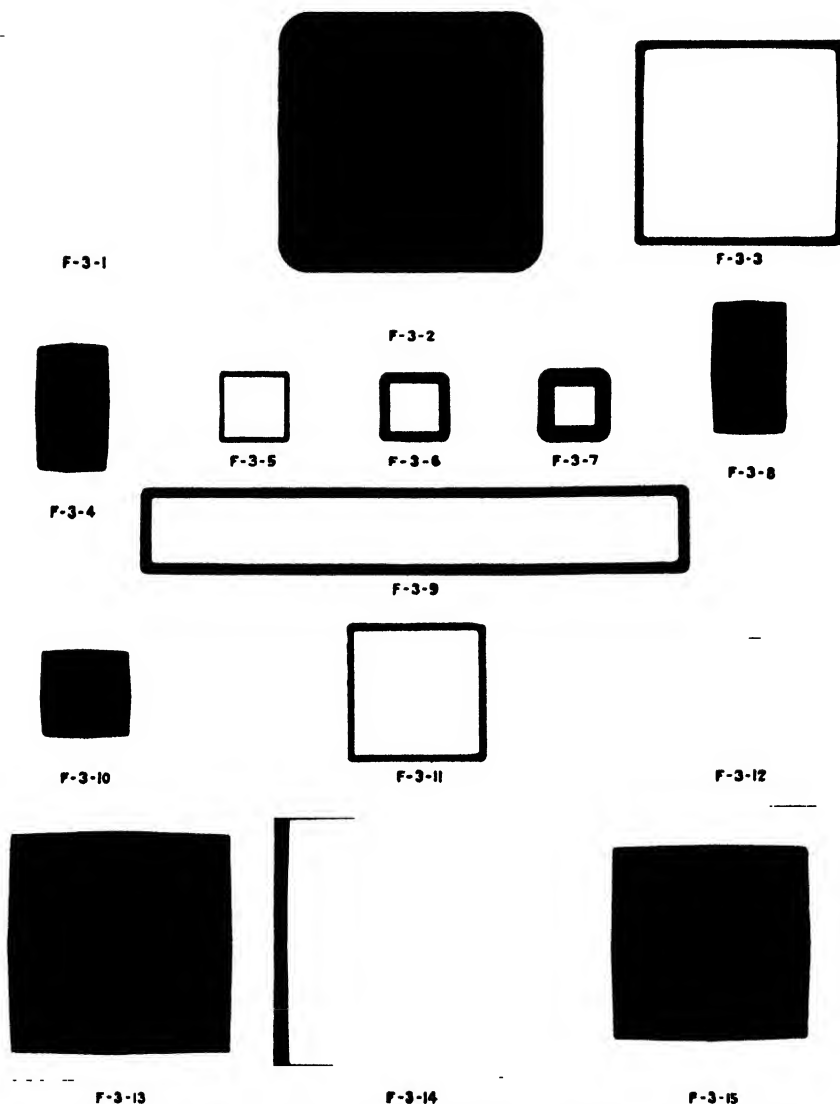


FIG. 41—46. Rectangular shaped sections of seamless tubing. Scale 1" = 2 $\frac{3}{4}$ ".

tubing, manipulation tests on coupons consist of flattening, expanding, flanging and crushing. All tubes are carefully measured for outside diameter, size and wall thickness. A protective coating of oil, or rust preventive, is usually applied before shipment. Light-gage tubes are boxed and small-diameter tubes are bundled to prevent injury during shipment.

Mechanical Tubing—Tubing for mechanical purposes is made in a wide range of sizes ($\frac{3}{8}$ -inch O.D. to 10 $\frac{3}{4}$ inches O.D.) and in many wall thicknesses in round and special sections. Some of these special sections are square, rectangular, oval, streamline, octagon, hexagon, etc. (see Figures 41—46 and 41—47). Square and rectangular sections are produced on a bench equipped with a Turk's head in place of the regular die. This Turk's head consists of a frame in which are mounted four rolls with their axes in one plane and so arranged around the pass line that the faces of the rolls form a square or rectangular hole through which the round tube of proper size is pulled, thus producing the desired section. Other special shapes are made in dies in which

the bore corresponds to the outside contour of the finished tube, or in which shaped mandrels support and control the inside dimensions. Mechanical tubing has wide application and can be found in airplanes, automobiles, agricultural machinery, electrical equipment, household equipment, etc. Specific uses include ball-and roller-bearing races, gravity conveyor rolls, bushings, separators, hydraulic cylinders and hoists, oil-well pumps, bicycle frames, metal furniture, etc.

Pressure Tubing—Tubing used to withstand internal or external gas, steam or fluid pressure in refining, chemical, or evaporator apparatus is designated as pressure tubing and is usually furnished in the full-annealed state to assure ductility under service conditions. The most common applications of pressure tubing are in boilers, condensers, heat exchangers, evaporators, cracking stills, refrigerators, and air-conditioning apparatus. Many of these applications can be met by hot-rolled tubing, but the smaller-size, light-walled tubes can be produced only by the cold-drawing process or by the Rockrite process.

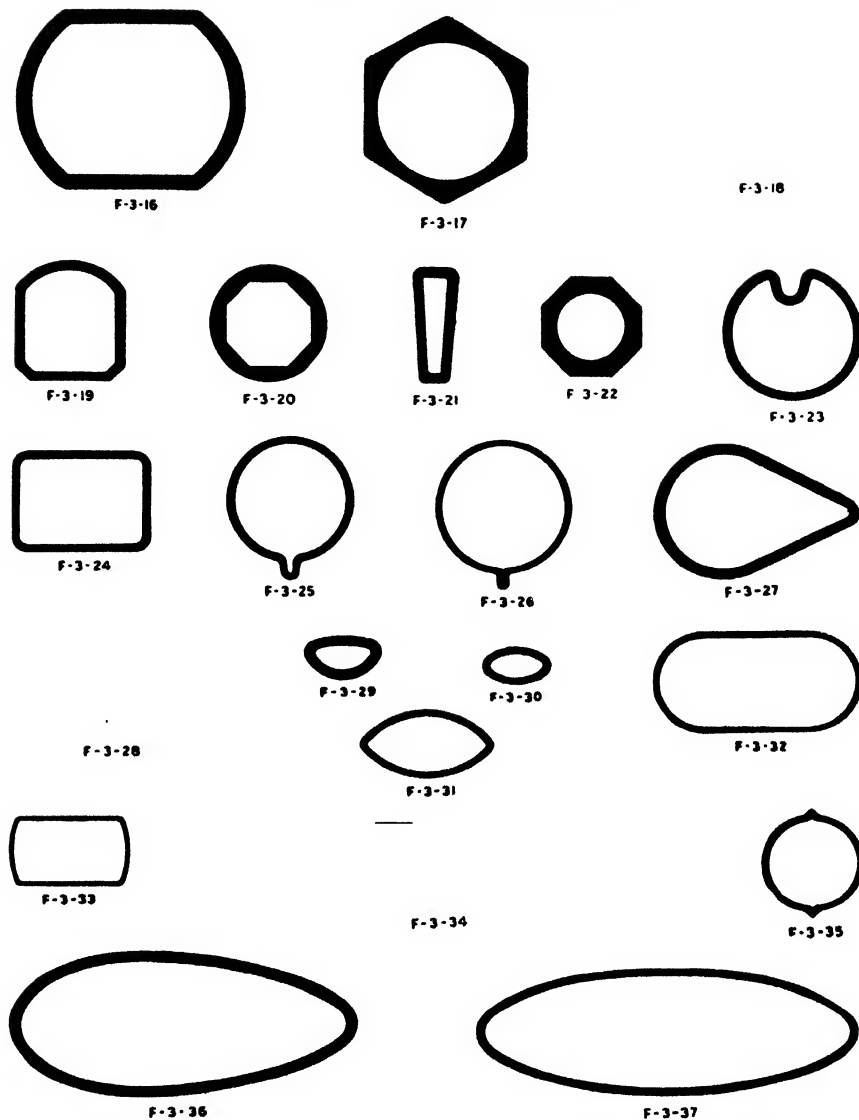


FIG. 41-47. Special shaped sections of seamless tubing. Scale 1" = 2 1/4".

Dimensional Tolerances of Cold-Drawn Mechanical Tubing—While dies for cold drawing mechanical tubing are held to the same exact sizes as wire- and bar-drawing dies, the fact that thin-walled sections will spring out of round while a solid section cannot, accounts for the somewhat greater tolerances required in tubing than with solid sections.

Surface Finishes—The various surface finishes in both hot-rolled and cold-drawn tubing are as follows:

1. Hot Finished
Hollow-forged billets and hot-rolled tubing.
2. Normalized
Hot-rolled or cold-drawn tubing.
3. Soft Annealed
Hot-rolled or cold-drawn tubing.
4. Medium Annealed
Cold-drawn tubing only.
5. Finish Annealed
Cold-drawn tubing only.
6. Hard Drawn (Unannealed)
Cold-drawn tubing only.
7. Ellwood B. F. (Bright Finish)
Cold-drawn tubing only.
8. Bright Annealed
Cold-drawn tubing only.
9. Specially Smooth (Cylinder Finish)
Cold-drawn tubing only.
10. Pickled
Hot-rolled or cold-drawn tubing.
11. Sandblasted or Shotblasted
Hot-rolled or cold-drawn tubing.
12. Polished
Cold-drawn tubing only.

1. **Hot Finished**—Hollow-forged billets have a surface appearance much smoother than hammered or pressed forgings, but may show a slight marking that will clean up with very little stock removal. Hot-rolled tubing has a surface finish comparable to plates or sheets of equal thickness. The thin-walled tubing, because of the high reduction in the rolling operation and the low finishing temperatures, will have a better surface than tubing with heavy walls. A light, tightly-adhering mill scale,

blue-black in color, is found on both the outside and inside surfaces of hot-rolled tubing.

2. Normalized—Whether hot-rolled or cold-drawn, all normalized tubing will be coated with scale, the thickness of which depends upon the thickness of section and grade of steel. Thin-walled tubing can be brought to temperature and cooled rapidly, thus avoiding long exposure to oxidizing atmospheres at high temperatures, while heavy sections require a longer exposure. Scale formation is directly proportional to time and temperature. Some alloy steels, notably those containing chromium or nickel in small percentages (1 to 5), usually are more heavily scaled when normalized, due to the time-temperature effect.

3. Soft Annealed—Soft annealing, as commonly applied to pressure tubing, or mechanical tubing that is to be manipulated cold, leaves a light scale from reddish brown to blue-black in color that is comparable in thickness to the scale on hot-finished tubes, but is usually less tightly adhering and of a more porous nature. This surface may at times resemble that caused by rusting in storage or in transit. Oil used for protective coating may be absorbed by the oxide film, and the tubing may have the appearance of not being properly oiled, when fully protected against normal atmospheric corrosion.

4. Medium Annealed—Due to the lower temperature employed, medium annealed tubing is very slightly scaled and the loose scale can usually be rubbed off easily, leaving a black oxidized surface.

5. Finish Annealed—As the temperature at which tubing is finish annealed produces an oxide film of blue color, the tubing has a blue-black appearance and the smooth surface produced in the cold-drawing operation is not disturbed. The oxide film offers a slight protection against local rusting or discoloration.

6. Hard Drawn (Unannealed)—As the tubing has no heat treatment after the cold-drawing operation, the surface is more or less bright, depending on the number of passes through the die and on the nature of the lubricant used in cold drawing. Normally, thin-gage tubing will have a smoother and more uniform surface than heavy-walled tubing.

7. Ellwood B. F. (Bright Finish)—Ellwood Bright Finish is applied to hard-drawn tubing in the lighter gages and smaller diameters and is secured by special treatment in the cold-drawing processes. The appearance of this tubing is like that of cold-drawn bar stock.

8. Bright Annealed—Annealed cold-drawn tubing may be furnished with a scale-free surface, when so specified, by annealing the material in a controlled atmosphere or bright annealing furnace.

9. Specially Smooth (Cylinder Finish)—Cylinder finish is often required in oil-well-pump tubing, hydraulic jacks and hoists, air cylinders and similar applications where a smooth, dense inside surface is desired. Due to the surface hardness and the fact that there are no circumferential scratches, as in machined or ground bores, this finish is particularly valuable in cylinders using soft plunger packing. This finish can also be supplied on the outside surface when desired.

10. Pickled Finish—Where mill scale or scale from heat treatment is objectionable, tubing can be furnished with pickled surfaces, both inside and outside. This is often desirable when the tubing is to be machined, especially in automatic machines using formed cutters, as the tool life is materially increased. Pickling also permits close surface inspection.

11. Sandblasted Finish—The method of removing scale by blasting is usually confined to heavy-wall pressure tubes, but facilities are available for treating mechanical tubing on both inside and outside surfaces. Sandblasting

gives a dull silvery finish, which is very susceptible to discoloration in handling and storage. A bright, smooth finish is obtained on heavy-walled, hot-rolled tubing, when sandblasted or pickled, and then burnished in a special finishing operation.

12. Polished Finishes—Polishing machines polish both the outside and inside surfaces of tubing. The polished finishes are classified as follows:

Grade "A" secured by use of No. 80 grit abrasive.

Grade "B" secured by use of No. 120 grit abrasive.

Grade "C" secured by use of No. 180 grit abrasive.

Grade "D" secured by use of No. 320 grit abrasive.

In the polishing operation, stock is naturally removed from standard-sized tubing, and, unless otherwise specified on the order, it will be assumed that O.D. tolerances may be under and I.D. tolerances may be over nominal dimensions.

Annealing of Plain Low-Carbon Steel Tubing—The temperature employed for annealing plain low-carbon steel tubing is dependent upon service requirements involving ductility, stiffness, manipulating properties, machinability, etc. In general, the higher the annealing temperature, the greater will be the ductility and softness of the material.

Hot-Finished Tubing—Tubing in the as-rolled condition is generally finished at temperatures just around the upper critical range of the steel which results in a structure similar to that obtained by normalizing. Hot-finished tubing, therefore, has good strength and ductility and can be used in those cases requiring good manipulating properties for bending and cold-drawing operations.

Finish Anneal—Unless otherwise specified, plain low-carbon steel tubes are given a finish anneal at 932° F, which restores moderate ductility to the metal. This anneal is used where stiffness and ease of machinability are important and ductility not so essential.

Soft Annealing—Where ductility is of paramount importance, plain low-carbon steel is soft annealed at 1300° F. This anneal will effect a high degree of recrystallization. This treatment is specified in all cases where severe bending and cold-drawing operations are necessary, also on all pressure tubing. In employing this anneal, free-machining qualities and stiffness are sacrificed to obtain good ductility and manipulating characteristics.

Cold-Drawn Tubing—Tubing in the cold-drawn condition will have maximum strength with slight ductility. Cold-drawn unannealed tubing will have good machinability and is specified where service requirements involve maximum hardness and high surface finish for plating, etc.

Medium Anneal—The temperature employed for medium annealing plain low-carbon steel is 1112° F. This anneal will correct the cold-worked structure to the extent of recrystallizing the ferrite but the carbide particles will still remain elongated in the direction of cold work. This anneal is applied to tubing required to withstand a moderate amount of cold forming and slight bending operations.

Normalizing—In order to obtain complete grain refinement, it is necessary to heat the steel to a temperature just above the critical range and cool in still air. This treatment causes complete recrystallization and refinement of the ferrite grains and the formation of lamellar pearlite. The mechanical properties obtained with this treatment on low-carbon steel are very similar to those obtained by soft annealing. Normalizing as a final treatment is generally not applied to low-carbon steel tubing except in special cases and on modified grades where such properties as impact resistance are specified.

SECTION 7

THE CUPPING PROCESS

Applications of the Process—A considerable quantity of tubular goods is made by the process of cupping a flat circular disc, and drawing out this cup to the desired diameter and length, on equipment such as that illustrated in Figure 41—48. Such is the practice at the Christy Park Plant of National Tube Division, where it is designated as the **cupping process**. Although seamless pipe was originally made by this method, the process is now employed only in the manufacture of special tubes and cylinders. Practically all of the gas cylinders used for the transportation of ammonia, carbon dioxide, oxygen, hydrogen, compressed air and other commercial gases, such as refinery products, are made by this method. In addition, special hydraulic cylinders for water and air receivers, airplane pumps, shells, and other specialties can be manufactured by the cupping practice. When first patented in 1851 by Remond, in England, it was intended for making all sizes of tubes. It is practicable now for all sizes larger than about three inches, but since tubes can be made so readily by the rotary-piercing processes, already described, it is seldom used except for the manufacture of cylinders. The greatest drawback to the method is the difficulty of making tubing in long pieces. The maximum lengths vary from ten to twenty feet, according to tube size and wall.

Chief Details of the Cupping Process—Although the cupping process requires considerable skill and experi-

ence, the principles of the process are simple and easily understood. The steel is delivered to the tube works in the form of square plates the sizes of which are calculated to contain the necessary volume of metal for the finished tube, plus a sufficient amount to compensate for normal losses in manufacture. These sizes may vary from two to seven feet square, and from $\frac{3}{8}$ inch to 4 inches in thickness. The smaller sizes of these plates are heated to a forging temperature in a suitable furnace, and placed in a press having a cutting die (Figure 41—49) where they are sheared to a circular plate, or disc form. Since this operation does not cool the plates to a great extent, they are immediately passed to a second press, placed in a position concentric with the circular opening of a die, and forced through it by a round-nosed plunger operating from above (Figure 41—50). The hollow cup thus formed is then placed in a third press where a mandrel forces the cup through a smaller die which deepens the cup, and, at the same time, reduces its diameter and leaves it practically cylindrical.

Plates for the largest sizes are cut into the disc form before heating, usually by an oxy-acetylene-torch cutting machine. These discs are then heated and cupped in the same process used for the smaller sizes.

Hot Drawing the Cupped Piece—After the final cupping process, the cup is again reheated preparatory to hot drawing on the horizontal bench. This bench (Figure

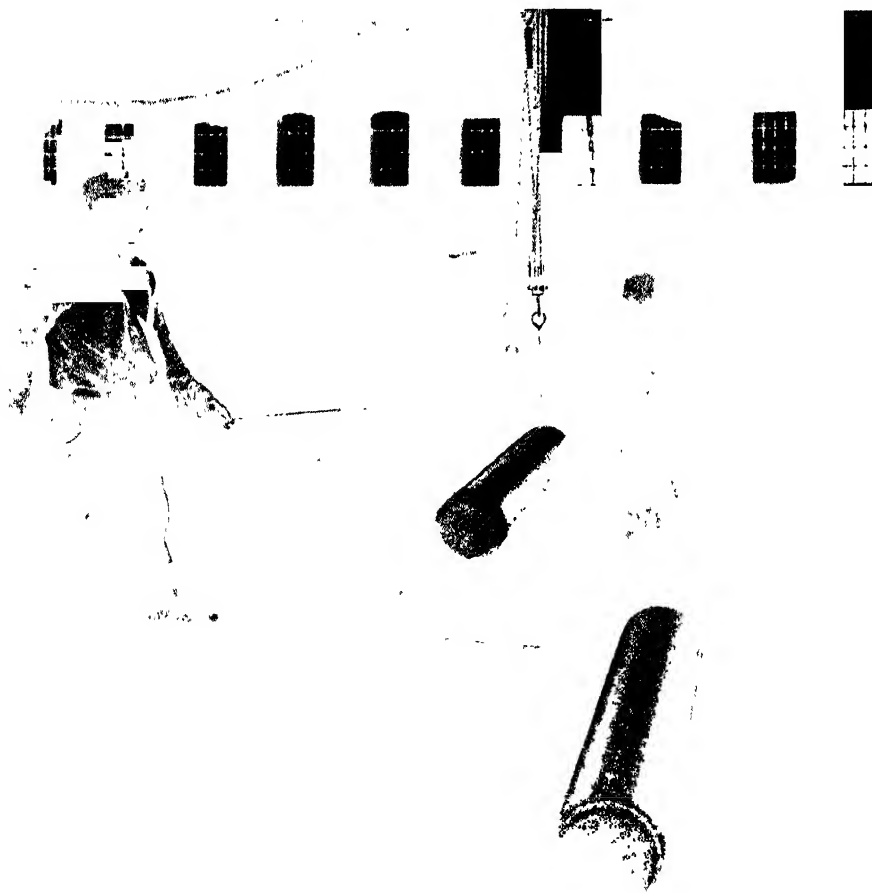


FIG. 41—48. Hot-draw bench in operation.



FIG. 41—49. View of press punching disc from square plate preparatory to first cupping operation.

FIG. 41—50. View of press starting first cupping operation.

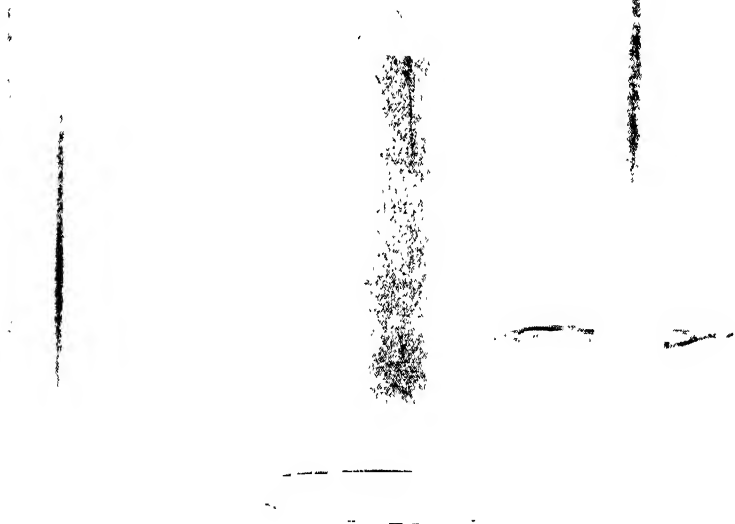


FIG. 41—52. Hammer swag-tubes being heated at open end for subsequent swaging.



41—48) consists of a heavy, cast steel, trough-like frame, provided in each side with suitable recesses for the insertion of dies and at one end with a powerful hydraulic plunger long enough to force a punch through the full length of the bench. The punch, of a size corresponding to the inside diameter of the tube desired, is placed on the end of the plunger, and a series of dies of successively decreasing diameters are dropped into the recesses of the bench-frame, which supports them in line. The reheated cup is now dropped into the bench trough, between the dies and the plunger, with its closed end facing the first and largest of the dies. The plunger advances, enters the cup, and pushes it slowly through the whole series of dies, elongating it and reducing its wall thickness and diameter at the same time. At the end of the forward stroke of the plunger, stripper tongs

are used to hold the forging while the punch is slowly retracted. As the drawing pressure is limited to a force less than that required to punch through the closed end of the cup, the operations of heating and drawing may have to be repeated to draw the tube down to the dimensions required. After the final operation, the plate which has taken the form of a tube with one end closed, may be cut off, or the closed end may be left intact to form the bottom of a cylinder. The bottom of the cylinder is frequently formed into a concave section by inserting a convex die at the end of the bench and forcing the drawn forging against this die at the end of the forward stroke. Tubes that require it are straightened in a gag press designed for the purpose. After inspection and testing, the process, so far as the tube-making is concerned, has been completed. The hot-drawn tubes under



FIG. 41—51. Hot-drawn ing one end of a tube to form the open end of a cylinder.

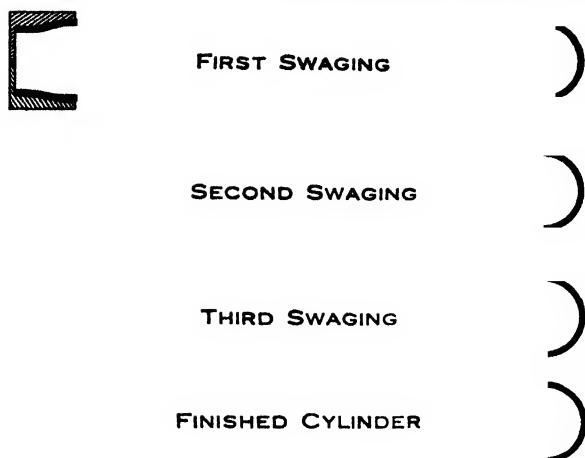


FIG. 41—53. Diagram of the swaging operations.

12 inches in diameter may be cold drawn, or cold drawn and annealed. These operations are carried out as already described.

Forming Cylinders—To form the necks of large cylinders (over 10 inches in diameter), the open end of the tube, after careful inspection, is trimmed evenly, heated to a forging temperature, and swaged under a hammer as shown in Figures 41—52 and 41—53.

Spinning—Another method for contracting, or closing, the ends of seamless tubes is known as spinning. This method is generally applied in the manufacture of small cylinders in which the end of a tube may be closed and perfectly welded in the same operation. Spinning is accomplished by a machine, which in principle is a lathe with a hollow spindle provided with a grip for holding

the tube in correct position for spinning. In carrying out the process, the end of the tube to be spun is heated to a dull-red color, the tube is placed in the machine, and the spindle is rotated at a high speed (600 to 1500 r.p.m.). The tool holder is mounted on a carriage which permits longitudinal and cross feeding of the center of sweep rotation, as well as adjustment of the sweep radius. A blunt tool of high-speed steel is brought to bear against the heated part of the tube and slowly swung around the end of it. The friction of this tool against the rotating tube generates heat at a high rate, and by repeatedly manipulating the tool the temperature may be raised to the point of incipient fusion for the steel of the tube. Figure 41—54 shows the different steps in the operations of closing the end of a tube. This illustration shows that, with each sweep of the tool around the end of the tube, the metal is gathered closer and closer towards the center, until the heat at the center has become so intense as to melt the metal, thus closing the end with a weld. After closure is effected, it is further possible to thicken the head thus formed by continued manipulation of the tool with simultaneous feed against the tube end.

The necks of cylinders can also be formed in this machine by alternately moving the tool carriage incrementally in the axial direction and manipulating the tool so that it does not cross the center of rotation. In this manner, closure is not effected, but a prolongation or neck is formed which may be tapped to receive a valve or connection. In the forming operation, external heat is supplied by oxy-acetylene torches. It can readily be seen that this principle is adapted to the forming of a great variety of ends.

The machine is of special value in the manufacture of small tanks or cylinders by simply cutting pipe into short lengths and spinning the ends to the closure or to the contour desired.



FIG. 41—54. Ends of a tube at different stages of the spinning process, showing the progress of the operation from the tube on the left to the closed cylinder on the right.

SECTION 8

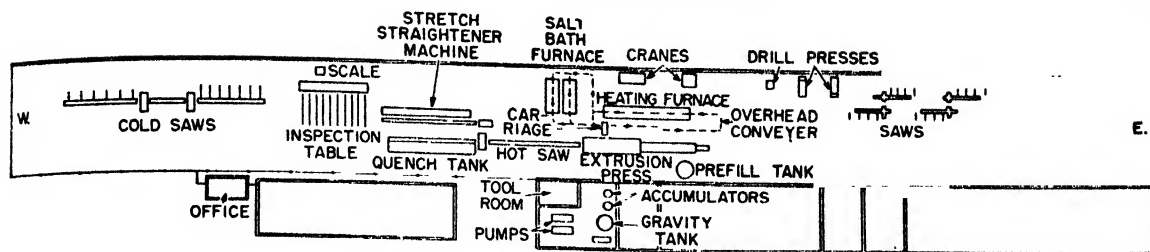
HOT EXTRUSION

Historical—The hot extrusion process has been successfully used to shape non-ferrous metals for many years and has also been used to a limited extent for some years in Europe for the production of carbon-steel tubes and bars. The presses, usually mechanically driven, permitted little flexibility in speed control, and the grease and graphite lubricant used left much to be desired. Tool life was short and only rather low extrusion ratios could be used. This made the operating cost of the process too high for economical use in this country where labor costs are much higher than in Europe.

During the late 1940's, the French inventor Jacques Sejournet developed the use of glass as a lubricant at the Persan plant of Comptoir Industriel D'Etirage et Profilage de Metaux to the point where stainless steel and other ferrous alloys of high strength and high melting point could be extruded much more satisfactorily and economically than was previously possible.

The extrusion process is well suited for the production of tubes or solid shapes of stainless steels and other high-strength alloys for which orders are usually for small quantities. With this process, it is possible to produce shapes which are difficult or impossible to form by other processes, to shape some steels which are difficult to roll or forge and some which previously could only be cast. Included in materials difficult to form are tubes of alloys known as non-pierceable because they cannot be produced on conventional seamless-pipe mills.

Advantages of Hot Extrusion—One of the advantages of the process is the ability to produce orders of small tonnage which cannot be rolled economically. The cost of dies for a special shape is insignificant in comparison with the cost of rolls and a size change can be made on the extrusion press in a few minutes except when it is necessary to change for a different size of billet to produce the required section. It is, therefore, a simple matter



FLOOR PLAN - EXTRUSION DEPARTMENT

Fig. 41-55. General plan of extrusion plant.

to make changes in the design of a section to be extruded, while the cost of such a change to a rolled section might be prohibitive. This ability to make changes is of particular advantage for new developments. Special formed sections can be extruded experimentally in small quantities until the best procedure for extruding and machining can be developed.

The extrusion process, which takes place at a substantially uniform temperature, with the steel under heavy compression in all directions, gives an end product with a fine and uniform structure well suited for further processing by cold drawing or machining. Some producers, particularly the French at Persan, specialize in precision sections which are cold drawn from extruded shapes.

All extrusions tend to be slightly greater in cross-sectional dimensions at the back end than at the front end. Several factors probably contribute to cause this effect. The glass coating is heavier on the front end of the piece, the die wears and its temperature increases as the extrusion is made. To obtain the accuracy of size usually required, it is necessary to cold reduce, cold

draw, or finish machine the extrusion. In the production of alloy tubes, it is possible to extrude small tubes with lighter walls than it is practical to pierce and roll on conventional seamless mills and to produce them with a much better surface. This means that the amount of tube-reducing or cold-drawing work on small tubes can be greatly decreased.

The Extrusion Press—For satisfactory operation, the extrusion press must be ruggedly constructed to withstand the heavy forces involved and remain in accurate alignment. When extruding tubes, it is essential that the mandrel be accurately centered in the die or eccentric tubes will result. The dies, which are thin, must be well backed up to minimize deflection. The die holder must be accurately centered with the bore of the container.

The furnace must be able to heat the billets uniformly to a closely controlled temperature and supply them to the press substantially free of scale.

Before heating, the billets must be properly conditioned. For some non-ferrous metals, the condition of the surface is not of particular importance. For example,



Fig. 41-56. Stud welder which flash welds studs to billet ends to provide support for billets carried on hangers through furnace.

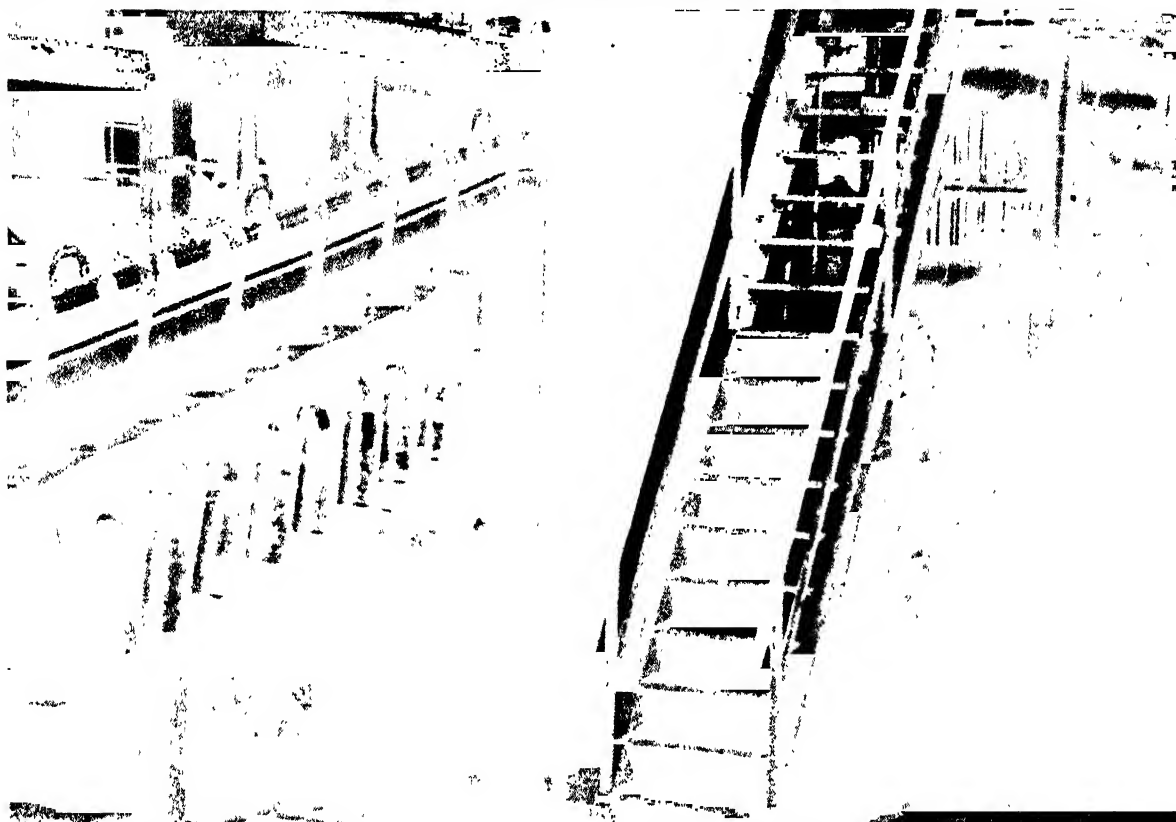


FIG. 41—57. Billets supported on conveyor are shown here entering the gas-fired furnace for preheating to 1600° F.

brass is extruded with a dummy block smaller than the container so that the billet is scalped and the surface layer is not extruded. With stainless steel however, the billets must be machined to a smooth surface.

The extrusion press at Gary Works of National Tube Division embodies a number of new design features which were found to be desirable when the experimental work was being performed in France. It has a main ram of 2000 tons capacity and a separate piercing ram of 500 tons capacity which can be used separately or together. The stroke of the main ram is 100 inches. The container is movable so that the discard can be easily removed as will appear later.

The Extrusion Operations—The general plan of the extrusion plant is shown in Figure 41—55. Turned rounds in lengths up to 20 feet are received and stored in the yard until needed. They are transported into the building by straddle truck and distributed by overhead crane to the four saws for cutting into the desired billet lengths. As mentioned before, these bars already have been turned. The finish must be rather smooth as any undue roughness will affect the finished product. The saws are of the milling-cutter type in order to produce a straight cut with a smooth surface, perpendicular to the axis of the bar. The billets are piled on pallets, which can be handled by a fork truck or by overhead crane. If intended for the production of tubes, they are next delivered to the drill presses where holes about $\frac{1}{4}$ inch larger than the inside diameter of the extruded tube are drilled from end to end on the axis of the billet. It is important that the drilled holes be straight and on center, as otherwise the extruded tube will be eccentric. Three drilling machines using high-speed twist drills about 4 feet long have been provided for this work. Oil holes

through the drills carry an ample supply of coolant to the cutting edges. Two of the machines are of the horizontal type which rotate the work while the drill is fed into it and the other machine is vertical and feeds the rotating work down over a stationary drill. The largest hole which can be drilled by any of these machines is 3 inches in diameter. Most of the stainless steel that is

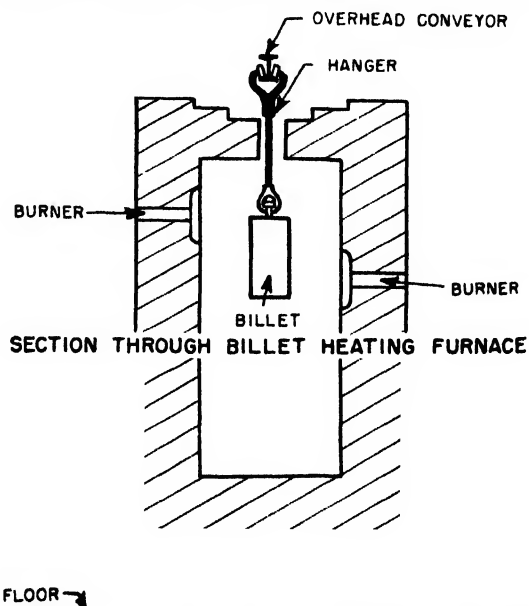


FIG. 41—58. Section through gas-fired billet-heating furnace.

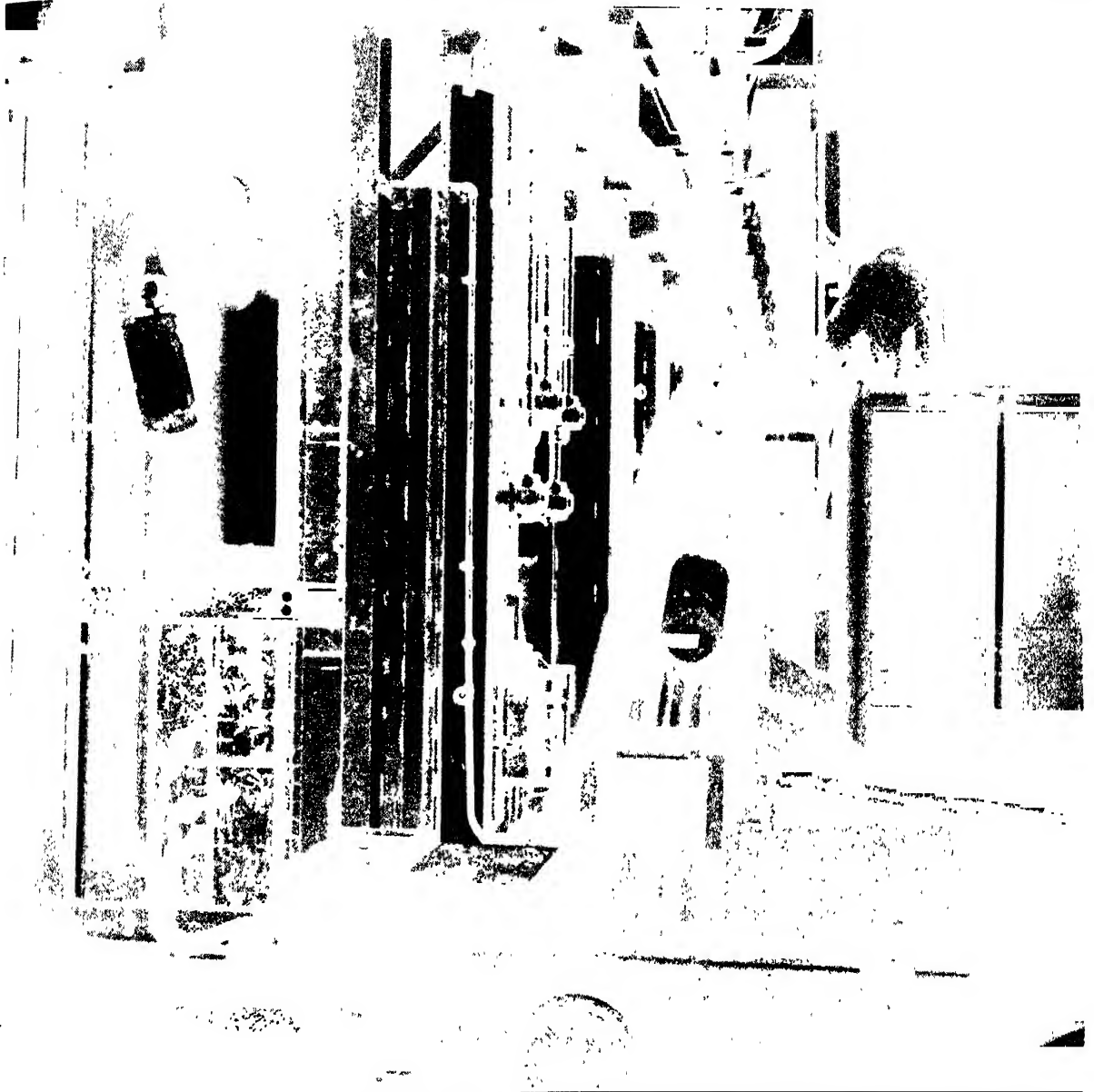
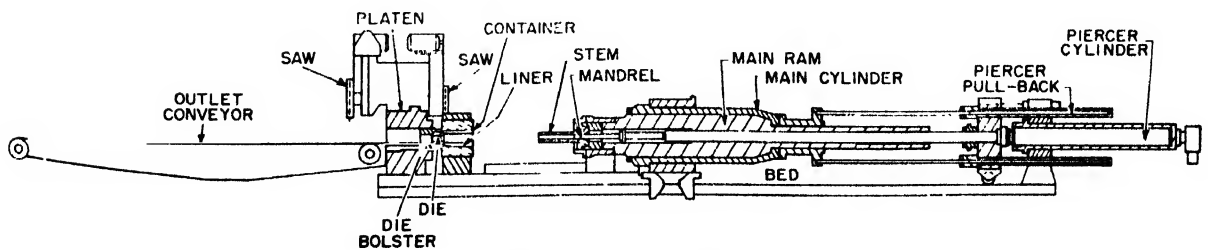


FIG. 41—59. Hot billet sliding down trough to charging carriage.



SECTION THROUGH EXTRUSION PRESS

FIG. 41—60. Schematic longitudinal elevation of an extrusion press.

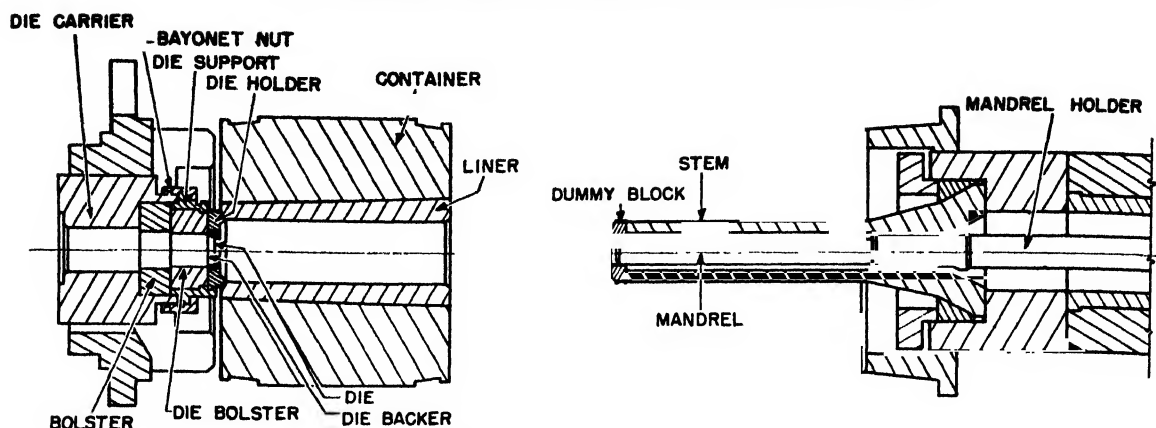


FIG. 41-61. Tool assembly for the extrusion press shown schematically in Fig. 41-60.

drilled and cut tends to work harden, and all sawing and drilling is performed at a slow speed and with a heavy feed.

The billets are then transferred to a lathe where one end is chamfered. This chamfer is provided on the lead end of the billet, which contacts the die, and has been found necessary to eliminate a surface flaw known as "corner defect."

The billets are next taken to the stud welder shown in Figure 41-56. Here, each billet is upended and a $\frac{3}{4}$ -inch carbon-steel stud with a round head is flash-welded to its end. Hangers on the overhead conveyor then pick up the billets and carry them through the gas-fired furnace where they are heated to about 1600° F. The furnace, which is shown in Figures 41-57 and 41-58, has a water-cooled slot in the top to permit the hangers to pass through. The slot is covered by overlapping alloy seal-plates which travel with the hangers and ride on water-cooled rails. The furnace is fired with coke-oven gas and is equipped with 54 radiant burners in the side walls with three zones of control. It has a rated heating capacity of 8 tons per hour at

2300° F. It is operated at only 1600° F, however, in order to minimize scaling of the steel.

The billets leaving the gas-fired furnace are rapidly conveyed to one of the salt pots. These are electrically heated baths of barium chloride which are operated at 2300° F. Each bath is 14 inches wide by 10 feet long inside, and has four pairs of electrodes connected to two transformers of 200-kva capacity each. It is important that the billets be uniformly heated throughout and that they be free of scale as it will cause rapid die wear at the extrusion press. The salt-bath furnace has, to date, proved to be the most satisfactory equipment for achieving the desired results.

After moving through the salt bath, the billets are lifted by conveyor and quickly carried to a point near the extrusion press, where the supporting stud is removed by an air chisel. The layer of salt which adheres to the billet helps to prevent formation of scale. The billet falls into a trough on the charging carriage, as shown in Figure 41-59. As the carriage moves toward the press, the trough drops to a horizontal position and discharges the billet, which rolls across the table to a

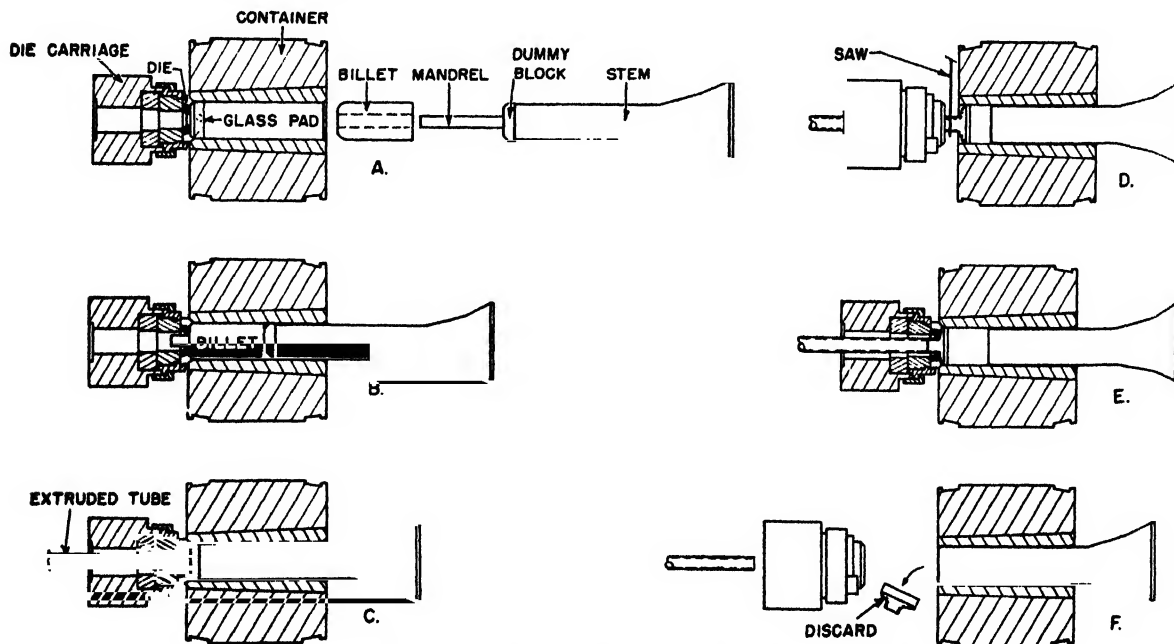


FIG. 41-62. Cycle of operations in the production of an extruded tubular section.



FIG. 41—63. View at the inlet side of the extrusion press.

trough on the opposite end. As it rolls, it wraps itself in a fibre-glass veil which acts as a lubricant in the container of the press. Figure 41—60 shows a longitudinal section through the press. The tooling is shown in Figure 41—61.

Before the glass-wrapped billet is introduced, the stem is fully retracted, the mandrel is extended from the stem the proper distance to permit it to reach through the die when the extrusion is started, a dummy block is placed over the mandrel and against the stem, and the mandrel is covered with a woven-glass sock. The container is placed in its forward or closed position, a die and holder are placed in the die carriage, and the carriage is moved up against the container and locked in position. A glass cartridge of approximately the same diameter as the container is placed in the container and pushed back against the die (Figure 41—62a). This cartridge may be a pad of fibre glass or a cake of powdered glass, or both. This glass, or a substantial portion of it, melts in contact with the hot steel and flows to the die, thus lubricating the bearing surface. The portion of the glass which does not melt serves to insulate the die and die holder from the hot metal. This glass is the important factor in reducing friction, thereby increasing die life and permitting longer pieces to be extruded and greater extrusion ratios to be used.

The stem with the dummy block and the mandrel now move forward rapidly under pre-fill pressure. The man-

drel enters the drilled hole in the billet and the dummy block presses against the back of the billet and compresses it until it fills the container (Figure 41—62b). The stem continues to move under high pressure and forces the hot steel through the die at a high speed (Figure 41—62c). The extrusion is usually performed in 2 or 3 seconds. It is desirable to perform this operation rapidly to avoid loss of heat from the piece and to minimize the temperature rise of the die and mandrel.

The traveling crosshead comes to rest against a splitting stop on the container housing before the dummy block reaches the die face. This leaves a short length of billet unextruded. The dummy block has slots in its outer surface and its face is chamfered so that the hot steel tends to flow over and attach itself to the dummy block. The stem is then pulled back a few inches and the container also moves back a short distance. The unextruded piece of billet, or discard, and the dummy block stick to the container and the extruded piece pulls back through the die, thus leaving a gap between the container and the die. The hot saw then descends through this gap and severs the discard (Figure 41—62d). Another forward movement of the container then pushes the end of the extrusion through the die (Figure 41—62e) and it is removed by the outlet conveyor. The die and its carriage are retracted, the container again moves forward, the crosshead stop moves away, and the stem is moved in to push the discard and dummy block from



FIG. 41-64. View at the outlet side of the extrusion press.

the container (Figure 41-62f). They fall to a conveyor which carries them to the side of the press, where a small hydraulic press squeezes the discard from the dummy block. The dummy block falls to a chute from which it is picked up by a device which raises it into position where the mandrel will pick it up. There are always several dummy blocks in the system to permit cooling before reuse.

After the discard is pushed from the container, the stem is retracted and the mandrel retracts inside the stem where it is water-cooled. A rotating brush moves in and cleans the container of glass residue. The die and its holder are removed from the die carriage and another die and holder previously prepared are inserted. The die which has been used is cleaned and prepared for another extrusion. Normally, from three to six dies are used in rotation. Figures 41-63 and 41-64 show the inlet and outlet sides of the extrusion press.

Piercing Billets—Tubes with inside diameters of $2\frac{1}{2}$ inches or larger and walls of not less than $\frac{5}{16}$ inch can be made by piercing the billets on the extrusion press instead of drilling them as described previously. A piercing head is placed on the end of the mandrel and pushed completely through the billet and through the die. The piercing head and a short slug of metal sheared from the inside of the billet drop off and the stem with the dummy block then advances and extrudes the piece.

Other Billet-Piercing Methods—In other plants it is common to provide a separate vertical press for the piercing operation. A section of the tooling for one such press is shown in Figure 41-65. Here the billet is first

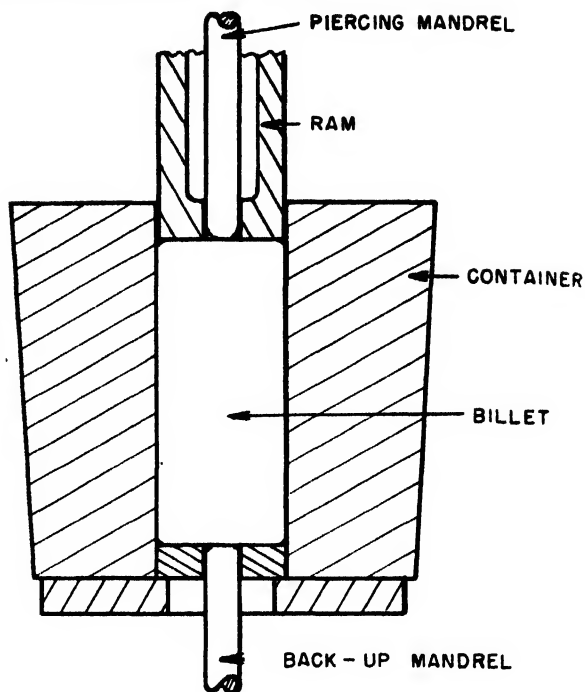


FIG. 41-65. Tooling used in some plants for the piercing of hot billets for extrusion. Piercing is performed on a separate vertical press by the method shown schematically in this sketch. Pierced billets must then be reheated before transfer to the extrusion press.

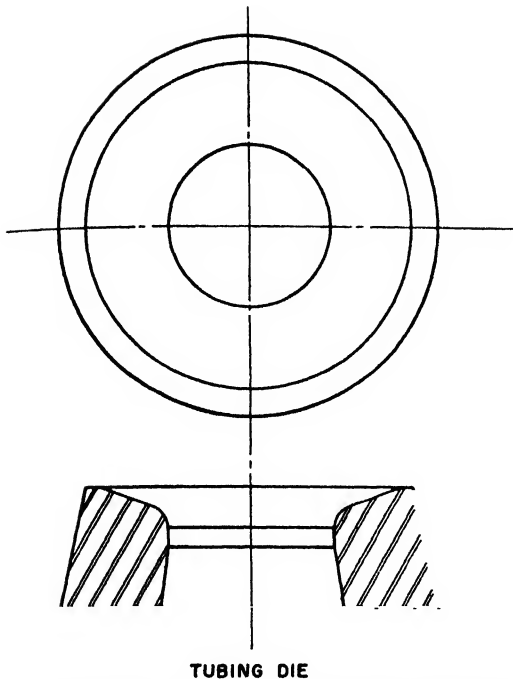
compressed by the ram which then guides the piercing mandrel to the center of the billet. The piercing mandrel advances to a point near the bottom, while the ram retracts and allows the billet to extrude backward. The back-up mandrel is then withdrawn and the piercing mandrel forces a small slug out as it completes its operation. Some use two presses and first pierce to a point near the bottom with a closed die pot on the first press, then complete the piercing on the second machine. In any case, it is necessary to reheat the billet before extruding.

There are other extrusion presses on which a closing plate is inserted between the container and the die while the mandrel pierces the billet to a point near the end, forming a closed cup. The closing plate is then removed and the mandrel completes the piercing operation, forcing a small slug through the die, after which the stem moves ahead to make the extrusion.

When solid shapes are extruded there is, of course, no mandrel, and the stem and dummy block are solid. Otherwise, the operation is exactly the same as for extruding tubes.

Operation of the Press—The press is operated by four men. One man on the inlet side of the press controls the conveyor delivering billets from the salt bath and cuts the stud from the billet. The second man operates all press controls except the dummy-block conveyor and die carriage. The third man removes the dummy block, disposes of the discard, and places the glass pads in the container. The fourth man operates the die carriage and takes care of the dies.

Tooling—All the extrusion tools are highly stressed in operation and must be made of alloy steels which will withstand heat. The containers are 36-inch outside diameter forgings with a $14\frac{1}{2}$ -inch bore at the entry end. The bore has a slight taper to accommodate a liner of the



TUBING DIE

FIG. 41—66. Sketch of die used for the hot extrusion of tubular sections.

desired inside diameter, which is pressed into the container. Liners are not removed until worn out; therefore, a container is provided for each liner. Stems, both solid and hollow, are provided to fit each liner with about $\frac{3}{16}$ -inch clearance on the diameter. These stems are subjected to compressive stresses up to 160,000 lb. per sq. in.

Dies range from $\frac{3}{4}$ to $1\frac{1}{2}$ inches thick and are of the general contour shown in Figure 41—66. They have a short bell-mouth, a straight bearing section which is usually about $\frac{3}{16}$ inch long, and a flared section on the

outlet side. Dies of 10 per cent tungsten steel and 5 per cent chromium hot-work steel have given good service. At best, dies wear rather rapidly, due partly to the fact that their temperature is raised considerably during each extrusion. Long extrusions cause much more rapid die wear than short ones. Dies of harder materials, and steel dies with hard facings such as stellite, have been tried experimentally but have not as yet proved successful.

Dies for solid shapes present more problems than dies for tubes. Unsymmetrical shapes have a tendency to bow or twist when extruded because the metal flows more easily and therefore faster through the wider portions of the aperture and through the portion at the center of the billet. Also, it is difficult to lubricate the re-entrant corners. To minimize these difficulties, the aperture in the die should be located so that the sections more difficult to extrude are as near the center as possible. Bearings sections can be made longer on the sections where the flow is fastest, and the bell-mouth part of the die can be increased to encourage glass flow to certain sections.

Mandrels are made from 5 per cent chromium steel, hardened and ground to size. These mandrels must withstand the heavy compressive stresses transmitted by the billet under pressure and also the tension caused by friction of the extruded piece. Failure is usually due to necking-down near the back end where the stresses are greatest. Due to the heavy stresses, there is a minimum limit for the size of mandrel of any definite material and for any certain extrusion. In general, it has been found best to keep the mandrel diameter over 1 inch, although smaller ones have been used.

Dummy blocks are of the same steel or a similar steel as that used for mandrels, with the addition of about $1\frac{1}{2}$ per cent tungsten.

Dies, mandrels, and dummy blocks are preheated to about 300° F in a small furnace before placing them in use. The container is kept warm with an electric heater when the press is not in use. If these things are not done, the tools may break due to thermal shock.

Power Supply—Power for the press is supplied by two 300-gallon per minute pumps. They supply water at

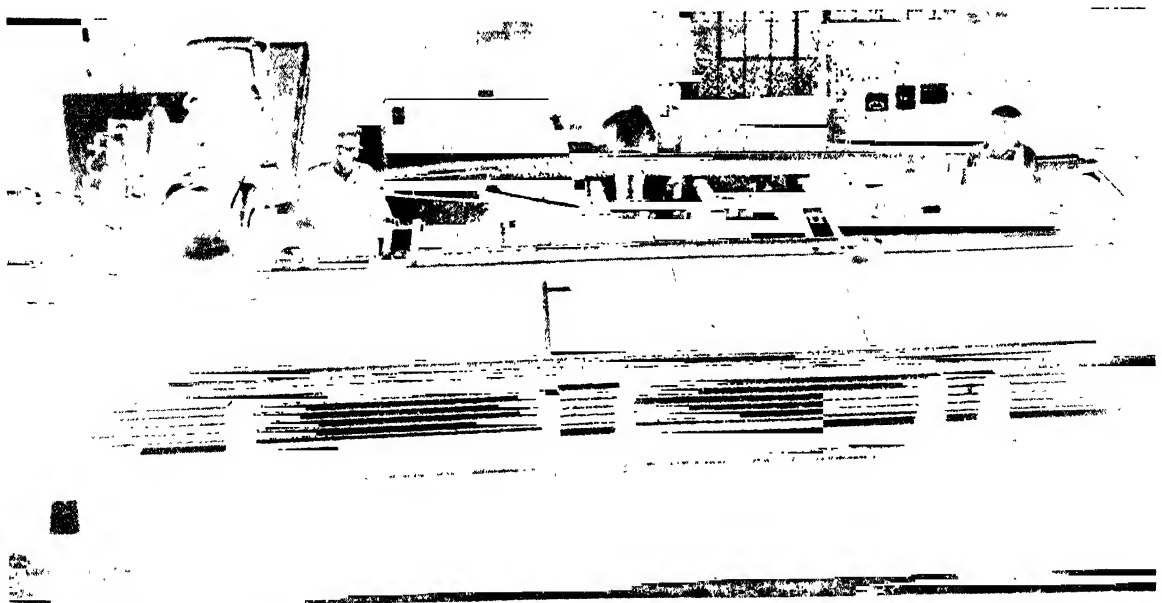


FIG. 41—67. Stretch straightener with rotatable head for the straightening and removal of twist from extruded sections.

3600 lb. per sq. in. pressure through two air-hydraulic accumulators. An air compressor is available to charge the accumulators, when necessary. One pump and one accumulator supply water for the main ram only. The other pump and accumulator supply the piercer ram and auxiliary equipment, including the die carriage, the die locks, the discard separator, and the container shift. An oil-hydraulic system operating at 500 lb. per sq. in. supplies the power to operate the main control valves through servo-mechanisms, the saws, and the hydraulic motor for the charging-carriage travel.

Finishing Operations—The extruded piece is carried on the outlet conveyor past a hot saw which can be used to divide the length if desired. The extrusion is then submerged in a quench tank if it is of austenitic stainless

steel. Steels which cannot be quenched are discharged onto skids to cool.

All extruded pieces are covered with a thin coating of glass which, due to its abrasive nature, must be removed before any cold work is performed. Quenching removes a portion of this glass, but the remainder must be removed by pickling. Before this de-glassing operation, special shapes are straightened on a stretch straightener, shown in Figure 41—67. This machine has a capacity of 100 tons, and one of the heads is rotatable so that any twist in the piece can be removed.

After straightening and de-glassing, solid shapes are cut to length. Tube hollows (a mill term for semifinished tubes produced by hot extrusion), after de-glassing, are further processed by cold drawing or tube reducing.

SECTION 9

FINISHING OPERATIONS

The hot-rolling processes described in the previous sections of this chapter on tubular products do not produce pipe in its final form. Though less spectacular, the finishing operations are a very necessary part of the production process. After leaving the straightening rolls, all pipe passes through a sequence of operations which depend on the size and class being produced. Pipe, rolled to a common specification on the hot mill, may be finished into various classes of pipe in the finishing operations. Inasmuch as these operations differ only slightly and usually only in the design of the equipment, the operations on seamless pipe have been selected as being typical of all classes.

Straightening—From the hot mills the hot-rolled pipe is delivered to the straightening machine which consists of two driven rolls from 2 to 3 feet long and 19 to 28 inches in diameter and three idler rolls 1 to 2 feet long and 12 to 18 inches in diameter, the former and two of the latter being set obliquely and opposite to each other, in position and angle, so that when a length of pipe is inserted between them it is held firmly by two sets of rolls. The third idler roll being between these confining points can be adjusted to bring pressure to the pipe and thereby perform the straightening. As in the reeling operation, the pipe is kept between the rolls by guides properly set between them and at some distance below the center line, depending on the size of pipe. The pipe, which is inserted between the first driven and idler rolls, is grasped, rapidly revolved, and drawn forward. During this action, the axis of the pipe is deflected by the middle idler roll which effects the straightening. The grip of the second driver and third idler roll finally throws the pipe clear of the machine.

Inspection and Cutting—When seamless pipe leaves the straightening rolls, it is delivered to a table where it is thoroughly inspected for straightness, size and external and internal surface defects, such as seams, pits, etc. The pipe is then delivered to the cutting-off machine where the crop ends are removed and the ends are cut smooth and perpendicular to the pipe axis, beveled for welding, or, if the pipe is to be threaded, it is chamfered to aid in starting the threading dies. The end is then reamed to remove any burrs and the wall thickness of the cut end measured to insure proper uniformity. Depending upon the class of pipe, a number of cut-off portions may be subjected to a flattening test to determine the ductility of the steel in the individual pipe.

Pipe Joints—A pipe joint is a means of connecting two or more lengths of pipe, so as to permit transportation of liquids or gases under leak-proof conditions, or

to permit the use of long lengths of pipe for mechanical or structural purposes. Generally speaking, joints may be divided into three classes:

1. Joints with threaded ends for couplings or flanges
2. Special connectors or couplings for use with plain-end or flanged-end pipe; as for example, Dresser, Victaulic, Vanstone, and similar joints.
3. Welded joints, including plain-end pipe beveled for welding, slip-joint casing and line pipe and double-belled-end line pipe with inserted chill ring for welding.

Joints with Threads and Couplings—The threaded and coupled joint is, by far, the most widely used and is the adopted standard of the American Petroleum Institute for casing, tubing, drill pipe, and line pipe. American Petroleum Institute (A.P.I.) casing in sizes 13 $\frac{3}{8}$ inches O.D. and smaller, tubing and drill pipe are manufactured with round threads; A.P.I. line pipe and 16-inch and 20-inch outside-diameter casing with the modified A.P.I. (Briggs') thread. "National" standard pipe to be fitted with couplings or flanges is manufactured to the applicable parts of the American Standards Association Threading Specifications. The great increase in the loads and internal pressures to which many types of threaded joints have become subject in the past few years has resulted in additional efforts to improve physical properties and threading practice. Thread form, thread depth, lead, and taper are maintained within the limits of the rigid tolerances given in the specification of the American Petroleum Institute and other organizations, so that coupling threads will mate properly with the pipe threads when made up to the power-tight position.

Threading Pipe—To secure good threaded joints it is necessary to have clean, smoothly cut threads of the proper taper and pitch, and to secure such threads it is necessary to have threading dies made with full consideration for the following: lip, chip space, clearance, lead, lubricants or cutting oils, and, for power machines, number of chasers.

Lip—Figure 41—68 illustrates clearly what is meant by lip on a chaser. The lip forms a slanted cutting edge which promotes curling of the chips and gives an easy cutting action, similar to that of a properly ground lathe tool, instead of the pushing-off effect caused by chasers which have no lip; it also permits a higher cutting speed. The angle to which the lip should be ground depends upon the kind of material to be threaded and the style and condition of the chasers and chaser holder. For Bessemer-steel pipe, this angle should be from 15 to 20

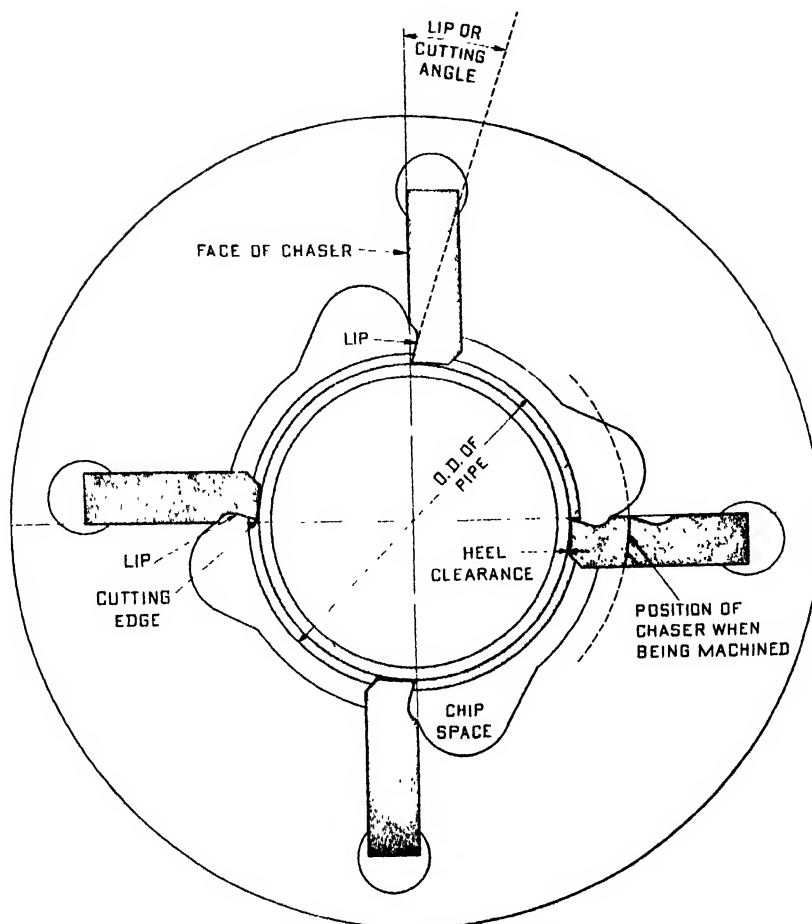


FIG. 41-68. Threading die showing lip, lip angle, chip space, clearance and cutting edge.

degrees; for open-hearth steel, the lip angle should be from 15 to 25 degrees.

Chip Space—Chip space is the space required in the die holder in front of the chasers to prevent the accumulation or packing up of chips. If sufficient chip space is not allowed, the chips will rapidly pack in front of the chaser, causing rough, torn threads, and creating a tendency on the part of the chaser to pick up stickers. The best design for this chip space provides an even curve for the chips to follow, with the back of each chaser well supported.

Clearance—Clearance is the space between the threads of the chasers and the threads on the pipe at a given distance from the cutting edge ("heel clearance" in Figure 41-68). This clearance is secured by die manufacturers in various ways. A simple method for getting clearance in the type of die known as "cutting-edge-on-center" or "center cut," consists of setting the chasers for machining with their cutting edge tangent with a larger circle than they are set for cutting threads. Clearance may be obtained on the "stock-on-center" type of chaser by machining in the same manner as a "cutting-edge-on-center" chaser, with the exception that their cutting edge is tangent to a smaller circle than when they are set for cutting threads. Stock-on-center chasers can also obtain their clearance by setting the chasers ahead of the center-line when they are being machined.

Lead or Throat—Lead is the angle which is machined or ground on the first three threads, more or less, of each

chaser to enable the die to start on the pipe, and also to distribute the work of making the first cut over a number of threads. The lead may be machined or, as is more common, it may be ground after the chasers are tempered. The proper amount of lead is about three threads. As the

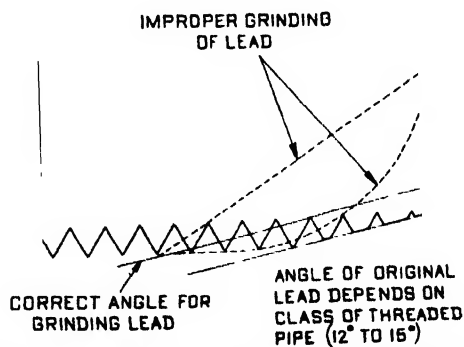


FIG. 41-69. Sketch showing correct angle for re-grinding lead of chasers.

heaviest cutting is done by the lead, this section of the chaser should have a slightly greater clearance angle than the rest of the threads, but care must be used to see that this angle is not excessive. Excess lead clearance will cause the die to feed too fast, and the half threads cut by the lead are consequently damaged by the full teeth of the chasers (see Figure 41—69).

Number of Chasers—To get good results in threading at one cut, the die head should have a suitable number of chasers. The number is determined by the size of the pipe. In some cases as many as eighteen chasers are required. The number necessarily depends upon the design, size, and operative principle of the die; hence, no exact rule can be laid down for universal acceptance. When an insufficient number of chasers is used, the die will chatter and cut a rough thread.

Dies—Dies usually are designed with the chasers evenly spaced and arranged either with the stock on center or with the cutting edge on center, as shown in Figure 41—68, in which case the face of the chaser is in the same plane as the central axis. A more recent development places the chasers at an angle of 24 degrees with a radial or center line and spaces them at varying angles around the pipe. An odd number of chasers is used. This has several advantages among which is the steadying effect the unequally spaced chasers have on the die thereby reducing chatter. Sharpening is simplified since the rake or lip angle is set at 24 degrees by the angle of the chaser with the center line and sharpening consists of merely grinding parallel to the face of the chaser.

Chaser teeth are usually designed to have the even-numbered chasers cut one flank of the thread and the odd-numbered chasers cut the other flank. This is done so that the threads of any individual chaser will not cut both flanks and results in a smoother thread.

Lubricant—Care should be taken to provide the proper quality of threading oil, as the best die made will not produce good results with poor or insufficient oil. With hand tools or where the flow is intermittent, No. 1 lard oil can be used with success. Cottonseed oils have a tendency to gum if not used in a constant flow. Poor lubricants are destructive to dies, and more power is required to cut a thread when they are used. A good quality of sulphurized mineral oil should be used with a constant flow on power machines.

Gaging Pipe Threads—To keep pipe-threading practice at a high degree of accuracy, the mills maintain complete sets of standard gages with which the pipe-thread dimensions may be measured. For each size of pipe threaded, a master plug gage is kept at the mills, and these gages are returned periodically to the National Bureau of Standards for examination of accuracy. Except for the pipe-thread vanish angle, the threaded master plug gage represents a theoretically correct pipe thread as to pitch diameter, lead, taper, and thread form. The ring gage is the transfer medium, also certified for accuracy by the Bureau of Standards, and represents a theoretically correct coupling thread with reference to lead, taper, and thread form. The ring gage is threaded and sized so that it will screw onto the master plug gage a predetermined distance, known as the hand-tight position. On American Petroleum Institute gages, the hand-tight position is expressed as *standoff*, which is the distance from a scribed mark or notch in the master plug gage to the face of the ring gage. The *standoff* represents the advance from the hand-tight to power-tight position which has been determined as necessary to obtain a leak-proof and otherwise efficient joint. The working plug and ring gages used to verify the accuracy of the couplings and threaded pipe, respectively, are in turn

compared with the master plug and ring gages, so that the makeup from hand-tight to power-tight position in the joint will be uniform and within the specification tolerance limits. In addition to plug and ring gages, the mills are provided with various other types of high-precision gaging instruments, each one serving its particular function in indicating the accuracy of thread depth, thread form or angle, taper of threads in a specified length, pitch diameter, and lead or pitch. A more complete description of thread inspection and gaging instruments may be had by reference to American Petroleum Institute Standards 5-B.

Coupling Forgings—All threaded couplings are made of seamless steel forgings of a grade of steel at least equal to that of the threaded pipe with which they are used. Seamless coupling forgings, also called *blanks*, are cut from lengths of seamless pipe, pierced and rolled to the required coupling diameter and wall thickness.

Finishing Steel Couplings—The blanks are stamped with the necessary identifying marks. They are then placed in machines which true up the ends, taper the bore to conform accurately with the taper of the thread and, if required, recess or chamfer the ends internally. This is all performed in the same operation to assure alignment of the two ends of the finished coupling. The preparation of the coupling blanks is of great importance and care is taken to see that all operations associated with their preparation are properly carried out. The prepared blanks are tapped on automatic tapping machines. These machines tap both ends of the coupling in one chucking operation to assure accurate alignment of the finished coupling. After tapping, all couplings are inspected for pitch, taper and thread depth with precision instruments specially designed for this type of work. Size or pitch diameter is checked on hardened and ground, threaded gages.

To prevent galling of the threads, the couplings are electrogalvanized. As a first step in this process, the couplings are first washed with a soda solution to remove the oil adhering from the threading operation. Thoroughly cleaned on the inside, they are placed, several at a time, in a specially constructed plating tank filled with a zinc solution. In this tank the couplings are supported so that they collectively form the cathode, while zinc poles project into the couplings to form the anode. Upon the passage of direct current through the apparatus, the zinc is deposited upon the inside of the coupling as a firmly adhering coat, the thickness of which depends upon current density and time. Since the coat will not adhere firmly if the current density is too great, this factor is limited and maintained to give a tight coat, and thickness is controlled by time. As an additional safeguard against galling, to facilitate the tightening up of the couplings on pipe and also to prevent rusting of the threads in service, a special thread dope composed of zinc dust and non-drying lubricating greases has been prepared for use with couplings.

Testing the Pipe—From the threading machines, or from the cutting-off machines if the pipe is plain end or beveled for welding, the pipe is moved over a final inspection table, where each length is carefully inspected for surface defects, end finish, size, etc., to the hydrostatic-testing machine. If the pipe is threaded, the threads are lubricated, after which the couplings are screwed up to the established hand-tight position, examined for *standoff*, and then brought up to power-tight position by a power screwing-on machine. The pipe is then filled with water and an internal hydrostatic pressure is applied. This pressure may be from 400 to as high as 12,000 pounds per square inch, depending upon the kind of pipe, the size, and the service for which it is to

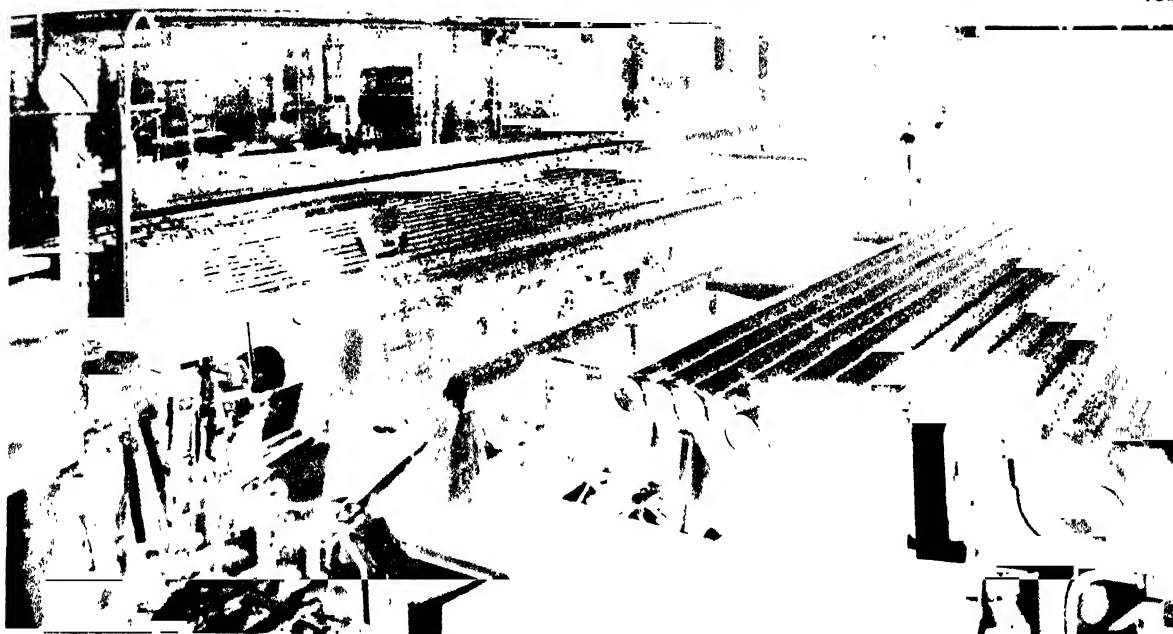


Fig. 41—70. Hydrostatic-testing machine.

be used. Some pipe, such as butt-weld pipe for ammonia purposes is also given an air test while submerged in water. For the hydrostatic test a specially designed machine is provided, which consists of a bench mounted on one end with a water-tight head connected with a hydraulic line and with a similar head made adjustable to suit different lengths of pipe, mounted on the other. A number of clamps for supporting the pipe and an air hammer for tapping the pipe during the test are mounted between these heads. When being tested, welded pipe is placed between the heads, the supporting clamps are applied, and the adjustable head is moved forward to seal the ends of the pipe tightly with packing rings or gaskets. Water is admitted until the pipe is full and no air pockets remain. The pressure line valve is then opened until the gage indicates the specified test pressure, which is maintained for five seconds, during which time the pipe is jarred automatically with the air hammer. The face of the air hammer is brought to bear upon and vibrate against the pipe, which is thereby subjected to impact and vibration while under maximum internal stress. The pipe is then unclamped from the testing machine, and one end is elevated while the water flows out, carrying with it flakes of detached scale.

Because of the enormous head pressures encountered in testing pipe, the higher test pressures are usually applied by what is known as the "field testing" method, so called because it was originally used with portable equipment. The extremely high pressures are employed to test casing and tubing to be used in deep wells where very high working pressures are encountered. A cap is screwed onto the field end pipe threads and a plug, through which the water enters the pipe, is screwed into the coupling on the other end of the pipe. Since there are no restraining forces on the ends of the pipe, the internal pressure acting on the cap and plug hydrostatically test the pipe joint in tension. Because of the very high pressures involved in this test, safety precautions are an important factor when installing and operating this equipment.

A very important factor in setting oil-well casing is the amount of external pressure exerted on the casing. This pressure is usually directly proportional to the depth of

the well and together with the bursting pressure and joint strength determines the grade of steel and wall thickness of casing required in deep wells. The resistance to collapse is proportional to the ratio between the outside diameter and the wall thickness of the casing which is usually expressed as D/t . It also varies with mechanical properties, especially the yield strength, below the range of elastic failure.

In order to assure a satisfactory level of collapse resistance, representative specimens of casing are subjected to external hydraulic pressure under laboratory controlled conditions. The length of the specimen for closed end testing used by National Tube Division, is eight times the diameter. After outside diameter and wall measurements are recorded, the ends are sealed either with portable leakproof heads or by welded plugs and the specimen is inserted in a heavy-wall forged jacket which is sealed and closed with heavy bayonet-type heads. After filling the jacket and bleeding out all entrapped air, hydraulic pressure is applied to the outside of the pipe by a high-pressure electrically driven reciprocating pump until failure of the specimen occurs through collapse of the section. The hydraulic pressure applied during the test is indicated on a calibrated mercury pressure gauge. Open-end tests are made by sealing the specimen using suitable packing rings through openings in the heads of the collapse jacket. In this method, the length of the test specimen is fixed by the length of the collapse jacket and consequently the ratio of the length of specimen to its diameter varies with each size. The method of applying and registering the pressure during the test is the same as with the closed-end test.

The test results are recorded and used as a process control, and as part of the experimental work which is constantly being done to improve casing quality and manufacturing method.

Oiling—Each length of pipe as it leaves the testing pump is measured, and this information together with the necessary identification marks is stenciled on the pipe, which is then given a coating of protective oil as it passes through a spray machine. This oil is a hardening transparent oil that leaves a lacquer finish. Sizes $1\frac{1}{2}$

inches and smaller are identified by stamping the necessary information on metal tags rather than stenciling, and these sizes, after oiling, are bundled to facilitate handling and shipment.

Types and Uses of Joints—The joints shown in Figures 41—71 to 41—80 represent those best known and most commonly used.

Upsetting—For severe service, it is often necessary to provide additional strength in the joint, and for this reason the ends of the pipe are upset before cutting the threads. To accomplish this upsetting, the end of the pipe is heated to forging temperature, then inserted endwise between two semi-circular dies of the upsetting machine. These dies clamp the pipe from the outside, while a mandrel, carrying a collar of the exact size of the outside diameter of the upset end, is inserted into the pipe. As the mandrel advances, the collar comes in contact with the end of the pipe and pushes the hot metal back to fill the ring-like space between the mandrel and the die. By changing the design of the dies, the upsetting may be controlled to displace the extra thickness either to the inside or to the outside of the pipe. Figure 41—71 shows internal upset casing.

Internal Upset Casing with Long Couplings—This design of joint is manufactured with an internal upset to support the weight of a long string of casing when it is suspended freely in a well of great depth. The importance of proper mechanical properties in the steel, as well as highly accurate threading practice, can best be appreciated when it is realized that this joint (as for example in 7-inch outside-diameter casing) may be required to support a 16,000 foot multiple-weight string which is equivalent to a dead load of 540,000 lbs. or 270 tons. Casing joints may also be subject to rock pressures as high as 7,000 pounds per square inch.

Seamless Buttress-Thread Casing—Non-Upset—This joint (Figure 41—72) is a development of National Tube Division to satisfy the petroleum industry's need for a casing joint which will safely and economically support the weight of casing designed for deep wells. The high tensile strength of the buttress-thread joint is due largely to the combined effect of the coupling threads completely engaging the casing thread throughout their entire length, including the vanishing threads, and the three-degree flank angle of the thread which support the weight of the casing in the well. Used on the API casing grades and with full inside clearance, 5½-inch through 7½-inch O.D. casing combination strings of multiple weight can be designed for 20,000-foot depths and corresponding approximately 10,000 lb. per sq. in. bottom hole rock pressure.

A.P.I. Casing with Long Coupling—A.P.I. casing with long couplings is manufactured in sizes 4½ inches to 13¾ inches outside diameter. The general outline of the joint is as shown in Figure 41—73. Casing with long couplings is intended for use in wells of somewhat lesser depths than the internal upset type and may also be used in the

lower sections of casing strings where the internal upset shown in Figure 41—71 is not necessary to obtain the desired joint strength and safety factor against failure. When it is necessary to obtain added clearance inside a larger size of casing or in open holes, couplings of sub-standard outside diameter but otherwise having standard dimensions may be used.

A.P.I. Standard Casing—The A.P.I. Standard Casing Joint (Figure 41—74) was designed for oil-well depths of 5,000 to 7,000 feet with a safety factor of 2.0 against failure in tension based on minimum physical properties. However, this joint may be used in the bottom sections of longer strings of casing, thus resulting in some reduction to the cost of the string. This joint is available with a sub-standard coupling outside diameter, where additional clearance is desired because of the necessity of its being run inside another string of casing or in open hole. The A.P.I. Standard Casing Joint is furnished in sizes 4½ inches to 20 inches outside diameter.

A.P.I. Line Pipe—The A.P.I. line pipe joints are similar in general outline to the A.P.I. standard casing joints (Figure 41—74). A.P.I. line pipe is threaded to American Petroleum Institute Standards 5-L, and the same rigid control of threading tolerances maintained as for A.P.I. casing and all other classes of threaded joints. Until recently, the essential differences between line pipe and standard pipe joints were in the coupling diameters and length and also in the thread length, each of these dimensions being greater in A.P.I. line pipe than in standard pipe. The American Petroleum Institute has since adopted the American Standards Association thread lengths for line pipe, thus eliminating the longer line-pipe threads. Line-pipe and standard-pipe couplings may now be used on pipe threaded to either the A.P.I. or A.S.A. specifications. The threaded line-pipe joint, however, is subject to higher test pressures since, in actual service, line pipe is generally subject to far higher working pressures than is standard pipe; the latter being more suitable for low-pressure work, such as piping for plumbing and sprinkler systems.

A.P.I. External Upset Tubing—A.P.I. external upset tubing (Figure 41—75) is desirable for deeper wells and for wells of any depth that are pumped. Both ends of the tubing are externally upset in order that the metal area at the root of the first exposed thread, when the joint is properly made up, may be at least equal to the metal area in the body of the pipe and thus of equal strength in tension. This tubing can therefore better absorb the dynamic stresses induced by pumping as the possibility of a fatigue failure in the threaded section is reduced to a minimum.

A.P.I. Standard Tubing—A.P.I. standard (or non-upset) tubing is generally used in open-flow wells and in shallow wells where tubing is pulled from the well infrequently and is a medium for pumping or otherwise raising oil to the well surface and thence to storage tanks. The A.P.I. standard tubing joint is similar in gen-

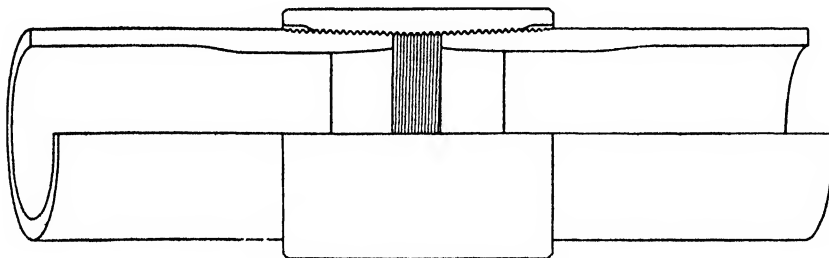


FIG. 41—71. Internal upset casing.

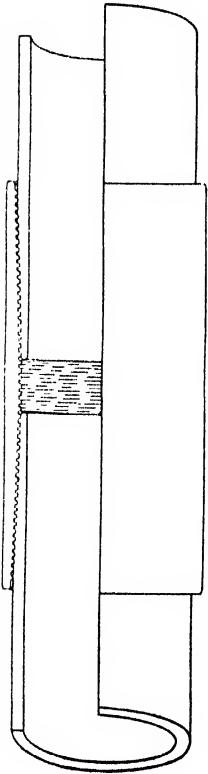


FIG. 41-72. Buttress thread on seamless non-upset casing.

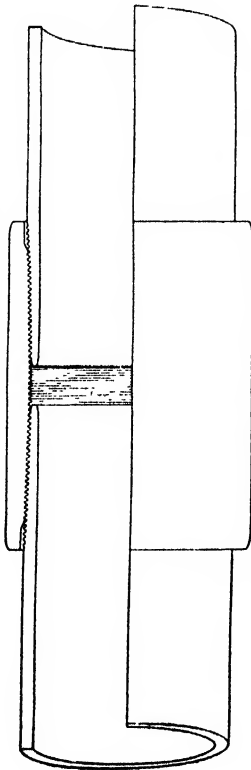


FIG. 41-73. A.P.I. casing (long coupling) A.P.I. tubing

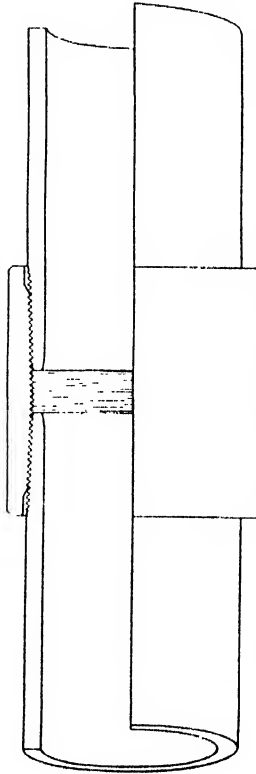


FIG. 41-74. A.P.I. casing (short coupling), A.P.I. line pipe.

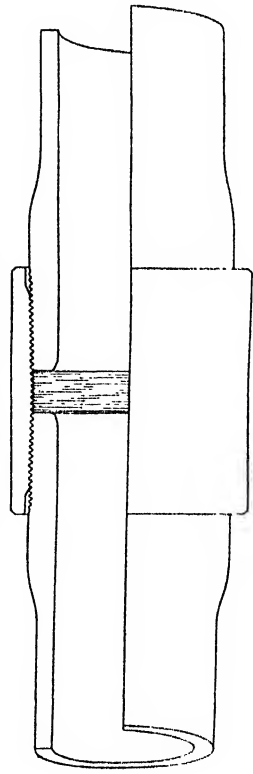


FIG. 41-75. A.P.I. external upset tubing.

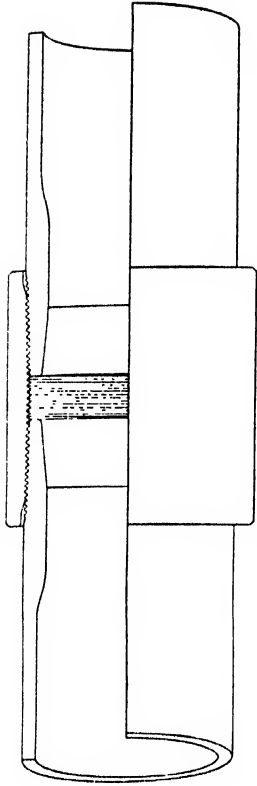


FIG. 41-76. A.P.I. internal upset drill pipe.

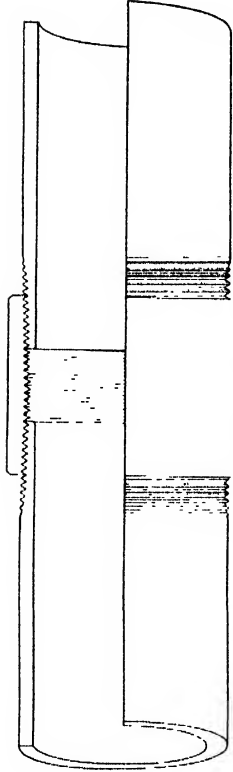


FIG. 41-77. Standard pipe.

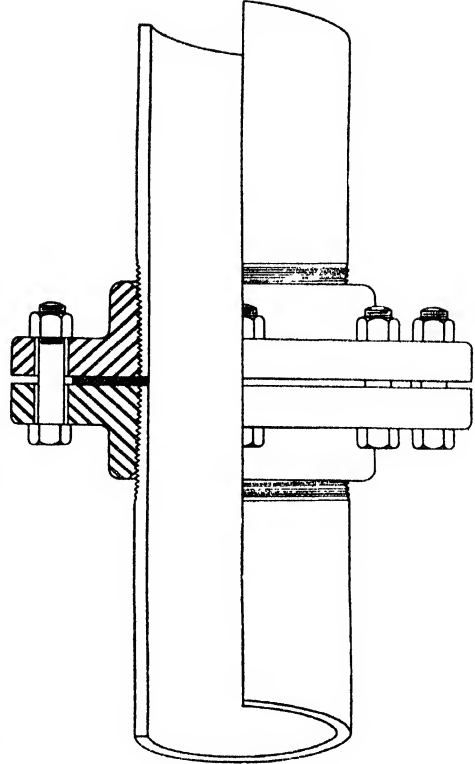


FIG. 41-78. Flanged joints.

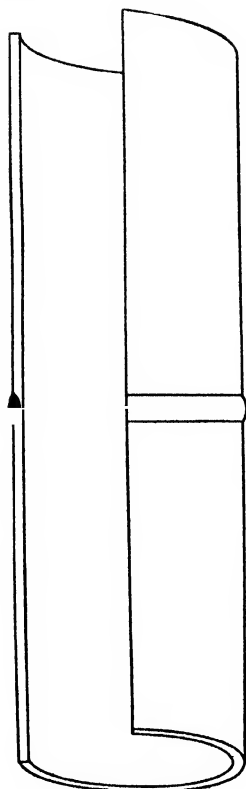


FIG. 41-82. Beveled end for welding.

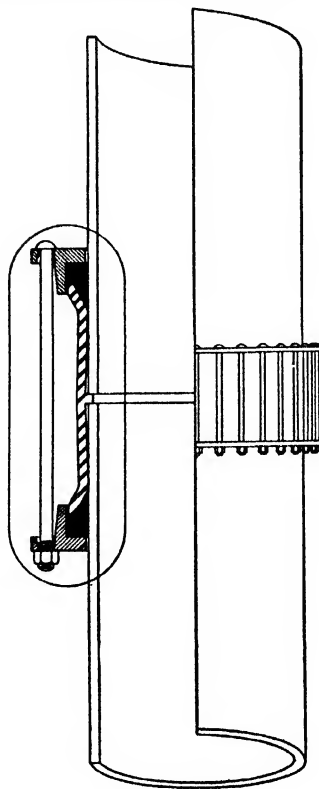


FIG. 41-83. Plain-end coupling.

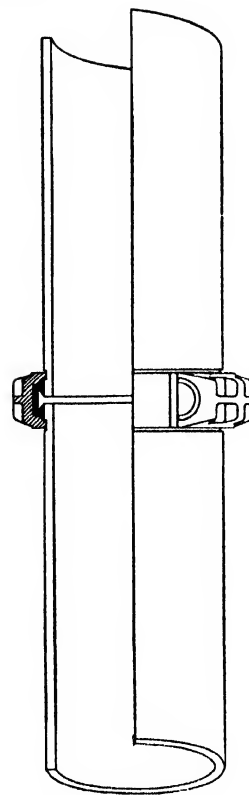


FIG. 41-84. Formed-end coupling.

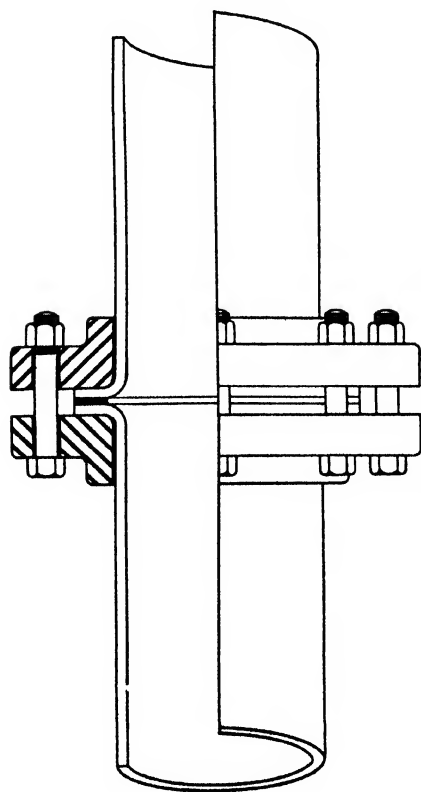


FIG. 41-79. Flanged joint-Vanstone type.

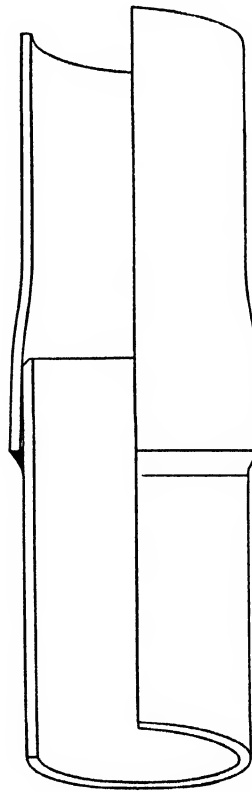


FIG. 41-80. Expanded end for welding.

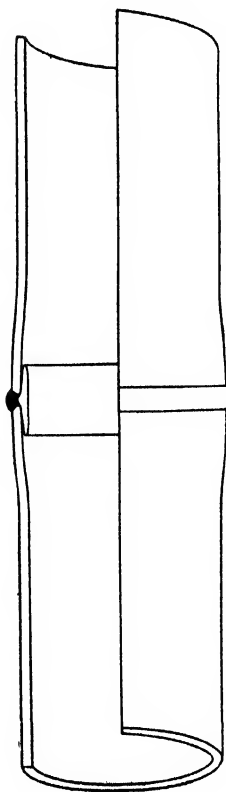


FIG. 41-81. Double-expanded end for welding with chill ring.

eral outline to the A.P.I. casing (long coupling) joint (Figure 41-73).

A.P.I. Internal Upset Drill Pipe—The drill-pipe joint (Figure 41-76), which is manufactured with upset end, is subject to still greater and far more complex loading, since strings of drill pipe as long as 20,000 feet are supported by one joint which also transmits the torque from the rotary table to the body of the pipe and ultimately to the bit. The bit, in turn, revolving at high speed is subject to varying degrees of pressure, depending on the formation being drilled. Where cavings occur, the pressure on the bit, expressed in terms of column loading, is sometimes sufficient to induce very high stresses in the joints. Drill pipe is also subject to high internal mud pressure, which is required to wash cuttings to the surface and to act as a lubricant while drilling. While the joint as shown in Figure 41-76 is still the A.P.I. standard, most strings of drill pipe are now fitted throughout with tool joints; and the design of these tool joints is such that the greater part of the stresses incidental to drilling pass from the body of the pipe through the tool joint and into the next pipe section, thus reducing the possibility of V-notch failures through the threaded section of pipe. This is accomplished by flash-welding the tool joint to the upset, or by welding the tool joint to the body of the pipe back from the threaded end, or by shrink-fitting an extension in the tool joint to a special external upset and machined pipe end. Since it is not the purpose here to describe types and uses of tool joints, the reader is referred to A.P.I. Specifications 7-B and to the catalogs of various tool-joint manufacturers.

Standard Pipe—The standard pipe joint (Figure 41-77), while primarily intended for low-pressure steam, gas, and water lines, as found in buildings and industrial plants, is also used for structural purposes, as in hand railing, scaffolding, etc. This joint has been superseded for long transportation lines by the A.P.I. line-pipe joint and welded line-pipe joint.

Flanged Joints—While only two types of flanged joints are illustrated in Figures 41-78 and 41-79, there are a number of other types of flanged joints that may be more suitably adapted to specific problems, and the fact that they are not described or illustrated does not in any way detract from their recognized merits.

Threaded Flange Joint—Pipe to be fitted with standard screwed flanges is threaded to A.S.A. Standards (Briggs' threads). The flanges for use with steel pipe are generally manufactured from forged or alloy steel, depending on the temperature and pressure to which they are to be subjected. Joints of this type (Figure 41-78) are used for power plants, refineries, etc., where working pressures of 1,500 pounds per square inch and temperatures of 1000° F are quite common.

Flanged Joint—Vanstone Type is illustrated in Figure 41-79. Pipe for use with this and similar types of flanged joints is heated at the ends to forging temperature and upset or thickened by rolling back the metal. This is done to compensate for reduction in the metal thickness that would otherwise occur, when the pipe ends are flanged or rolled back to the position shown in Figure 41-79. Since the rolled-back pipe ends are not faced, leakage is possible, and to avoid this, gaskets of softer material are inserted between the pipe ends when making up the joint.

Expanded End for Welding and Double Expanded Ends for Welding—These types are illustrated by Figures 41-80 and 41-81. The female or expanded ends for both joints are formed by the same machine that prepares the ends for Vanstone type or any other formed-end joint. The joint shown in Figure 41-80 is frequently used for long transportation pipe lines employing light pipe walls

subject to low pressures. Figure 41-80 is also representative of slip-joint casing. The joint is made up in the field with the female or expanded end held uppermost on the casing slips. The male end is lowered into the expanded end, held in as near correct alignment as possible, and quickly tack welded, after which the welding is completed by acetylene torch. The joints shown in Figures 41-81 and 41-82 are also recognized as standards for welded pipe lines. Figure 41-81 shows expanded pipe ends fitted with a backing-up or chill ring which strengthens the joint and prevents the formation of metallic icicles when molten weld metal enters the space between the pipe ends and solidifies. The pipe ends are prepared for welding by thoroughly cleaning and removing any coating or dirt that would affect the quality of the weld, after which the ends are held by clamps in alignment with a specified gap between the ends; the pipe is then tack-welded, and, when in final position, welding is completed. Pipe may also be welded by placing a dolly having rollers mounted on it, under the pipe ends to be welded. The pipe may thus be rotated to facilitate welding. Plain-end pipe for welding is generally furnished to A.P.I. Specification 5-L which requires a bevel angle of 30 degrees and $\frac{1}{16}$ -inch flat at the end of the pipe. Plain-end pipe beveled for welding is used extensively for power-plant work where high temperatures and pressures are encountered. The welding is generally made either by the electric-shielded-arc or oxy-acetylene process, welding rods to suit the chemical and mechanical properties of the steel in the pipe being used.

Dresser-Type Joint—This joint (Figure 41-83) has been frequently used for transportation of high-pressure natural gas, also for low-pressure oil and water lines. It is easily and rapidly assembled and is made leakproof by tightening the bolts, thus exerting pressure on the gaskets and in turn on the pipe perimeter. The two gaskets are of rubber, or other suitable material, depending on the temperature to which the joint is to be subjected, and also upon the corrosive action of the fluid being carried in the pipe. The joint is designed to accommodate contraction or expansion due to temperature changes encountered in the line, and will also take care of a certain amount of misalignment. The ends of the pipe for Dresser-type couplings are gaged to specified diameter and roundness tolerances to insure proper assembly in the field.

Victaulic Joint—Like the Dresser-type joint, the Victaulic joint (Figure 41-84) is quickly assembled and designed to take up expansion or contraction due to normal temperature changes. This joint will also absorb a certain amount of angular misalignment. Figure 41-84 shows the grooved-end type of joint, the pipe ends being grooved or machined out to provide a seat for the coupling. Other formed pipe ends used with Victaulic couplings are:

1. The shouldered end.
2. The expanded end.
3. The folded-back end.

Of these, the expanded end is most generally used. The pipe ends are expanded in an upsetting machine and finished to the specified dimensions. The joint is assembled by first lubricating the pipe ends, and then slipping the ring gasket over one pipe end, after which the second pipe end is brought into position and the ring moved over to cover both ends of pipe. The metal housing or coupling, made in two sections, is mounted over the ring gasket so that it fits into the grooves or over the expanded or shouldered pipe ends, and is made leakproof by tightening the bolts, thus forcing the gasket

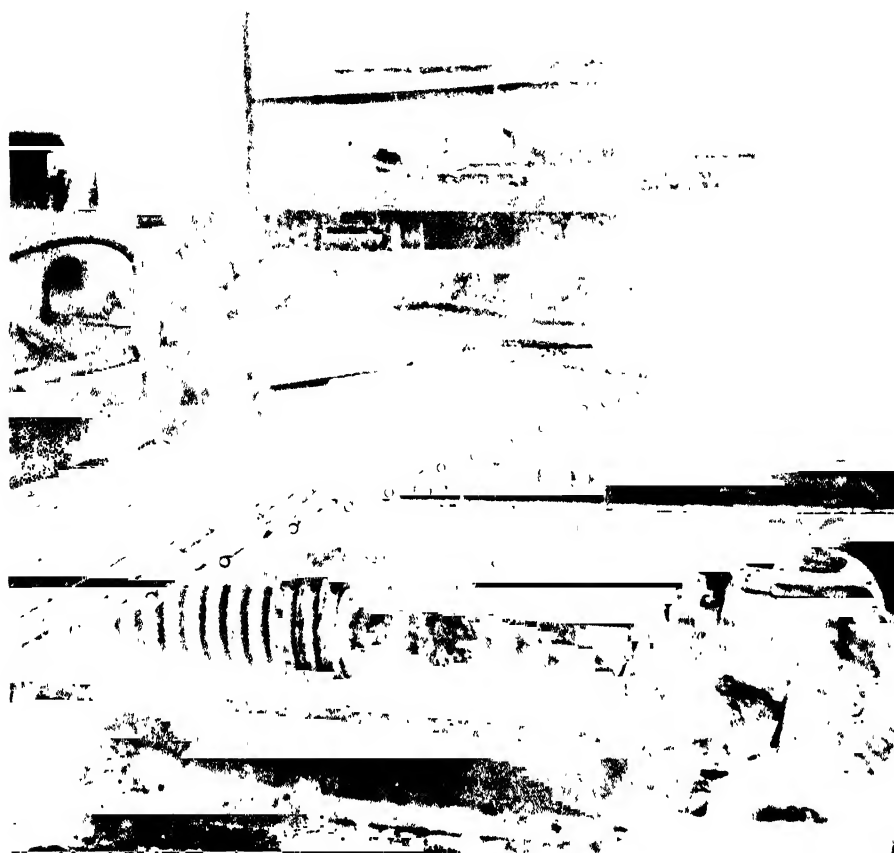


FIG. 41—85. Pipe discharging from a continuous galvanizing kettle.

against the pipe face. As will be seen in the illustration, the gasket is so formed that pressure inside the pipe tends to increase the resistance to leakage.

Galvanizing—The hot-dip process of galvanizing is widely used for applying zinc coating to steel pipe for protection from corrosion. The pipe is first thoroughly cleaned by washing in a water solution of caustic soda to remove all traces of oil, paint and grease. It is then pickled in a hot, dilute sulphuric-acid solution (containing an inhibiting agent) to remove all rust and mill scale, after which it is washed free of adhering acid by immersing in a bath of fresh, clean water. The fluxing operation then follows and is accomplished by immersing the pipe in a hot solution of zinc ammonium chloride until the temperature of the pipe approximates that of the solution. Upon removal from the fluxing tank, the pipe is thoroughly drained, then placed on the charging table of the galvanizing kettle where it is introduced into a bath of molten zinc maintained at a temperature of about 870° F. When the pipe has been immersed for a time sufficient to permit its temperature to reach that of the molten zinc, it is withdrawn slowly in an inclined position to permit the excess zinc to drain from both the inside and outside. It is then conveyed through the cooling tanks, at which time it is given a special chromate treatment which aids in preserving the bright, metallic appearance of the zinc coating.

Several methods are employed for processing pipe through the galvanizing kettle. An outstanding development is the continuous method of processing pipe in the size range $\frac{1}{2}$ -inch to 2 inches inclusive, whereby the pipe are handled on a series of spiral-groove rolls which rotate

the pipe with a screw-feed motion, imparting forward and lateral travel through the galvanizing kettle (Figure 41—85). This is accomplished by automatically charging the pipe on an inclined table equipped with spiral-groove rolls, the pipe moving on a downward slope of approximately 10 degrees and entering the kettle at the corner of one end. The pipe continues downward into the kettle until it is completely immersed, the direction of slope is changed from downward to upward by pivoting the pipe on the center roll. The pipe continues traveling forward and laterally, leaving the kettle at the diagonally opposite corner. A low-pressure air blast is directed on the pipe on the outgoing table, as it emerges from the kettle, to set the coating and maintain a smooth surface and bright luster. The pipe is then discharged into a tank containing a dilute solution of sodium dichromate.

The continuous method of galvanizing pipe has the advantage of a high production rate and uniform and closely controlled operation, $\frac{1}{2}$ -inch standard pipe, for example, being galvanized at a rate of 1250 pieces per hour. With bath temperature and immersion time closely controlled, a decided improvement is obtained in the uniformity of the inside and outside coating of the pipe. A full coating is also obtained, as the pipe is not wiped. This provides a surface coating of almost pure zinc which is believed to have corrosion resistance superior to pipe which has been wiped.

Galvanized pipe in sizes over 2 inches is processed by the batch method wherein the pipe is charged into the galvanizing kettle in batches, the amount of the batch depending on the size of the pipe being processed. In

this process, the pipe arc submerged by a sinking-arm arrangement and after a short interval are withdrawn from the kettle by a magnet, the pipe being pulled out of the kettle on an inclined plane to provide drainage of the excess zinc from the inside and outside surface. No wiping operation is normally employed on pipe processed by the batch method; however, in galvanizing pipe in sizes smaller than 1/2-inch or pipe requiring an exceptionally smooth surface and a minimum weight of coating, the pipe is withdrawn from the kettle on single-draw magnetic rollers, the excess zinc being air-wiped from the outside and blown from the inside surface during withdrawal.

Value of Zinc Coating for Pipe—Though zinc coating is readily soluble in dilute acids, strong alkalis and some mineral-salt solutions, galvanizing affords effective protection against ordinary atmospheric corrosion, because the zinc, when exposed to the air, immediately reacts with oxygen to form zinc oxide. This reaction progresses only to a limited extent, however, for the oxide remains as a thin film on the surface of the zinc coating, and protects the zinc from further oxidation. Since this oxide is insoluble in water and since the film adheres rather tenaciously to the zinc, it protects the underlying coating of zinc against all ordinary types of weathering, and as long as the zinc remains intact, the steel beneath is secure from corrosive action. If the zinc coating is broken, by bending or abrasion, and the underlying steel is exposed to corrosive influences, the zinc will still afford considerable protection. Being electro-positive to the steel, the zinc is dissolved instead of the steel as long as any zinc remains closely adjacent to the exposed steel. In underground piping, where the pipe is in direct contact with the earth, galvanizing is not generally suitable as protection against corrosion, because it may be quickly destroyed either by certain acids or by certain alkali substances in the soil.

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FIG. 42—2. The microstructure of slowly cooled hypoeutectoid steel showing ferrite and pearlite. Magnification: 100X.

ents which are being discussed. It is a homogeneous phase, consisting of a solid solution of carbon in the gamma form of iron. It is formed when steel is heated to a relatively high temperature, say above 1450° F, (788° C). The limiting temperatures for the formation of austenite vary with composition and will be discussed in connection with the iron-carbon diagram. The metallographic appearance of austenite is shown in Figure 42—7, which represents a sample of an alloy steel which has been very rapidly cooled from the temperature range at which the austenite is stable, the steel being high enough in alloy content to make possible the retention of austenite structure at room temperature.

The atomic structure of austenite is that of gamma iron—face-centered cubic—and the atomic spacing varies with the carbon content.

Pearlite—When a plain carbon steel of approximately 0.80 per cent carbon is cooled slowly from the temperature range at which austenite is stable, all of the ferrite

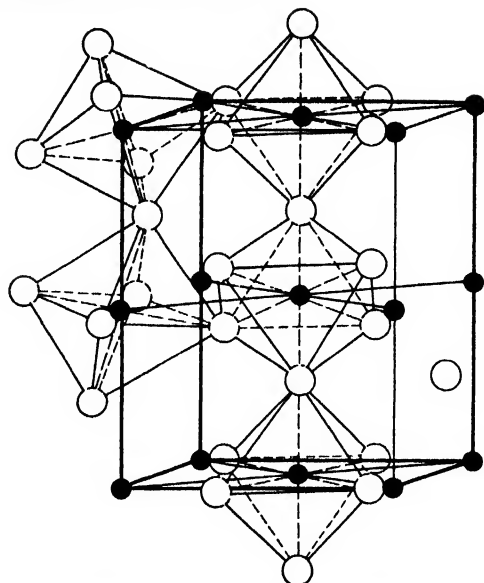


FIG. 42—4. The atomic structure of cementite. Positions of carbon atoms are indicated by solid circles; positions of iron atoms by open circles. (Hendricks, S. B.: *Zeitschrift für Kristallographie*, Vol. 74 (1930), 534-545.)

and cementite precipitate together in a characteristic lamellar structure known as pearlite. This structure is illustrated in Figure 42—8. It is generally similar in its characteristics to an eutectic structure but since it is formed from a solid solution rather than from a liquid phase, it is known as an **eutectoid structure**.

At carbon contents above and below 0.80 per cent, pearlite of about 0.80 per cent carbon is likewise formed on slow cooling, but the excess ferrite or cementite first precipitates usually as a grain boundary network, but occasionally also along cleavage planes of the austenite. This excess ferrite or cementite rejected by the cooling austenite is known as a **proeutectoid constituent**. The carbon content of a slowly-cooled steel can be estimated from the relative amounts of the pearlite and proeutectoid constituents in the microstructure.

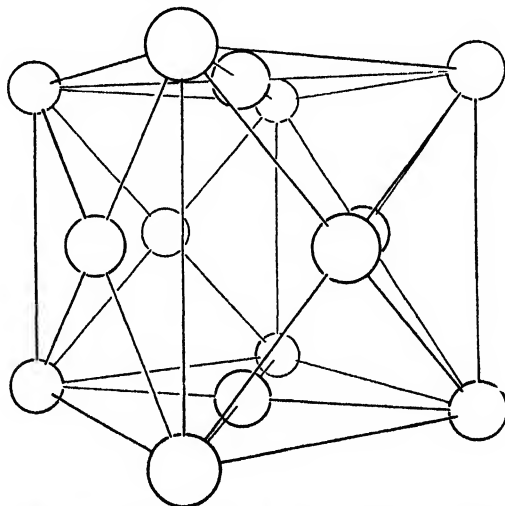
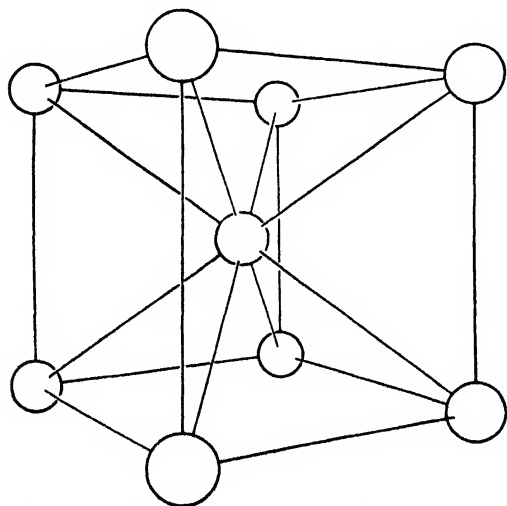


FIG. 42—3. Crystalline structure of the allotropic forms of iron. Each white sphere represents the relative position of an atom in a "unit cube" of: (left) alpha and delta iron, which have the body-centered cubic form, and (right) gamma iron, which possesses the face-centered cubic form.



FIG. 42-5. The microstructure of slowly cooled, high-carbon steel showing pearlite with cementite in the grain boundaries. Magnification: 1000X.

The Iron-Carbon Equilibrium Diagram—The iron-carbon equilibrium diagram furnishes a "map" showing the ranges of compositions and temperatures within which the various phases are stable and the boundaries at which phase changes occur. Although heat treatment is largely concerned with a controlled departure from equilibrium, this diagram represents the limiting conditions and is basic to an understanding of heat-treating principles.

The iron-carbon equilibrium diagram is depicted in Figure 42-9. This diagram, covering, as it does, the temperature range from room temperature to the melting point of iron and carbon contents of from 0 to 5 per cent represents the equilibrium conditions for the entire range of steels and cast irons in both the liquid and solid states. Use of this diagram in ensuing discussions involves two constituents, **ledeburite** and **graphite**, which have not been mentioned up to this point. Although



FIG. 42-7. The microstructure of austenite. Magnification: 500X.

these are not ordinarily constituents of steels, their characteristics will be briefly discussed at this point.

Ledeburite—Ledeburite is the metallographic term for the iron-iron carbide eutectic, containing 4.3 per cent carbon. This eutectic is a constituent of iron-carbon alloys containing more than 2.0 per cent carbon and for this reason the dividing line between steels and cast iron is customarily set at 2.0 per cent carbon. The metallographic appearance of ledeburite in cast iron is shown in Figure 42-10.

Graphite—Cementite is unstable over certain ranges of composition and temperature and decomposes into iron and graphite. Thus, in most slowly-cooled cast irons, graphite is an equilibrium constituent at room temperature. The gray appearance of the fracture of such slowly-cooled cast irons reflects the presence of the graphite and the metallographic appearance of graphite in gray cast iron is shown in Figure 42-11.

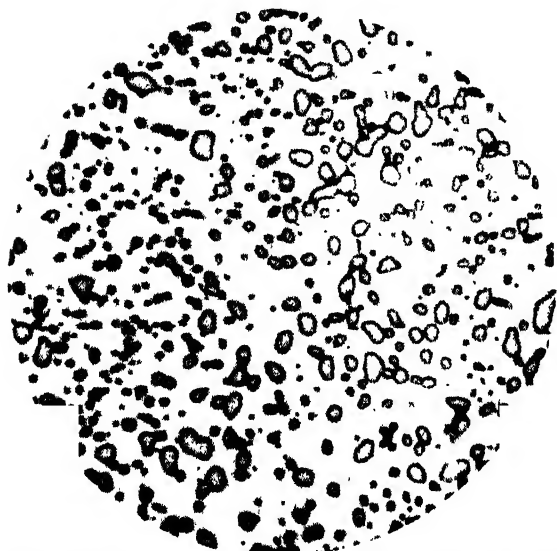


FIG. 42-6. Micrograph showing spheroidized cementite in matrix of ferrite. Magnification: 1000X.



FIG. 42-8. The microstructure of pearlite. Magnification: 1000X.

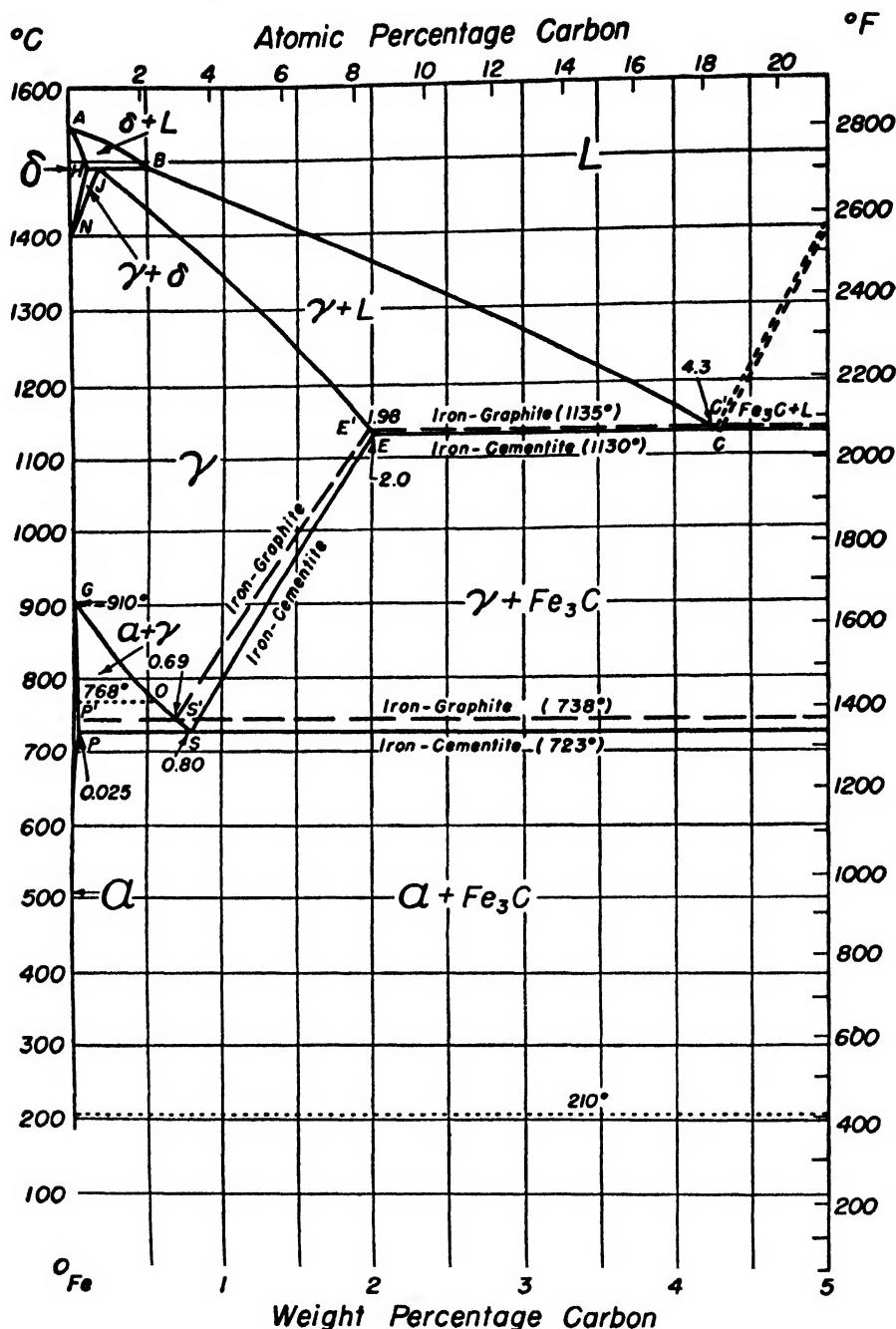


FIG. 42-9. The iron-carbon equilibrium diagram, for carbon contents up to 5 per cent. From "Metals Handbook" (American Society for Metals), 1948 Edition; page 1182.

Graphite may, under certain conditions, be a constituent of steels and the metallographic appearance of graphite in a low-carbon steel which has been subjected to a prolonged heating at a temperature below that at which austenite is formed is illustrated in Figure 42-12.

The Iron-Iron Carbide Equilibrium Diagram for Steels—The portion of the iron-iron carbide diagram of concern in connection with the heat treatment of steel is that part extending from 0 to 2.0 per cent carbon. The general features of this diagram (Figure 42-9) will be discussed and its application to heat treatment will be

illustrated by considering the changes occurring on heating and cooling steels of selected carbon contents as depicted by the diagram.

Critical Temperatures—In Table 42-I, iron is listed as occurring in two allotropic forms, alpha or delta (the latter at very high temperature) and gamma. The temperatures at which these phase changes occur are known as critical temperatures and the boundaries in Figure 42-9 show how these temperatures are affected by composition. For pure iron, these temperatures are 1670° F (910° C) for the alpha-gamma phase change

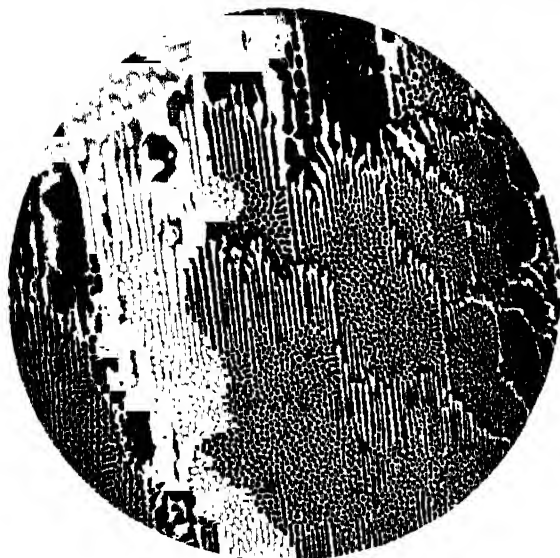


FIG. 42—10. The microstructure of ledeburite. Magnification: 150X.

and 2552° F (1400° C) for the gamma-delta phase change. The critical temperatures on heating are designated as A_c (from the French "chauffage," meaning "heating") temperatures, while those on cooling are designated as A_r (from the French "refroidissement," meaning "cooling") temperatures. Although, in principle, the transformations A_c and A_r result from the same temperature of equilibrium, in practice, the A_r temperatures are lower than the corresponding A_c temperatures, falling as the cooling is more rapid; even with very slow heating or cooling these temperatures do not coincide and, therefore, the subscript "e" (for equilibrium) is used in the designation of the critical temperatures in the equilibrium diagram. The meaning and significance of the critical temperatures will be clarified in the following discussion of the changes occurring on heating and cooling iron-carbon alloys as depicted by the equilibrium diagram.



FIG. 42—11. The microstructure of gray cast iron showing graphite flakes. Magnification: 100X.

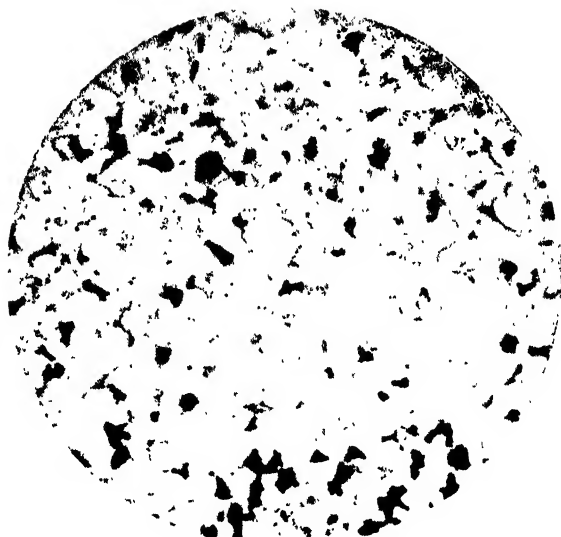


FIG. 42—12. Micrograph showing graphite particles in low-carbon steel. Magnification: 100X.

Changes Occurring on Heating and Cooling Pure Iron—The only changes occurring on heating or cooling pure iron are the reversible changes, (1) at about 1670° F from body-centered alpha iron to face-centered gamma iron and (2) from the face-centered gamma iron to body-centered delta iron at about 2552° F.

Changes Occurring on Heating and Cooling Hypoeutectoid Steels—Hypoeutectoid steels are those which contain less than the eutectoid percentage of carbon (0.80 per cent). The diagram shows the equilibrium constituents are ferrite and pearlite, the relative amounts of each depending upon the carbon content. The diagram also shows that at 1112° F the ferrite may hold in stable solution about 0.007 per cent carbon. At temperatures up to 1333° F, the solubility of carbon in the ferrite increases until at this temperature, the ferrite contains about 0.025 per cent carbon. The first phase change on heating (if the steel contains above 0.025 per cent carbon) occurs at 1333° F and this temperature is therefore designated as the A_1 critical temperature. On heating just above this temperature, the pearlite (ferrite and cementite) all changes to austenite. Some proeutectoid ferrite, however, remains unchanged. As we heat to temperatures farther above the A_1 temperature, the austenite dissolves more and more of the surrounding proeutectoid ferrite, becoming lower and lower in carbon, until at the A_3 temperature, the last of the proeutectoid ferrite has been absorbed into the austenite having the same average carbon content as the steel.

On slow cooling the reverse changes occur. The austenite first rejects ferrite (generally at grain boundaries) on cooling below A_3 and becomes progressively richer in carbon, until, just above the A_1 (eutectoid) temperature, it is substantially of eutectoid composition. On cooling below A_1 , this eutectoid austenite changes to pearlite so that the final product after cooling below A_1 is a mixture of ferrite and pearlite, the relative proportions of each constituent depending upon the carbon content.

Changes Occurring on Heating and Cooling Eutectoid Steels—Since no excess ferrite or cementite is present in eutectoid steel, the only change occurring on slow cooling or heating is the reversible change from pearlite to austenite at the eutectoid temperature. Thus, in the

case of eutectoid steels, the A_2 and A_1 temperatures coincide and this eutectoid composition and temperature is designated as the A_{s-1} point.

Changes Occurring on Heating and Cooling Hyper-eutectoid Steels—The behavior on heating and cooling hyper-eutectoid steels (steels containing more than 0.80 per cent carbon) is similar to that of hypoeutectoid steels except that the excess constituent is cementite rather than ferrite, so that on heating above A_1 , the austenite gradually dissolves the excess cementite until at the A_{cm} temperature all of the proeutectoid cementite has been dissolved and austenite of the same carbon content as the steel is formed. Similarly, on cooling below A_s , cementite precipitates and the carbon content of the austenite approaches the eutectoid composition. On cooling below A_1 , this eutectoid austenite changes to pearlite and the room temperature constitution is, therefore, pearlite and proeutectoid cementite.

The A_s Formerly Designated Critical Temperature—The diagram of Figure 42-9 shows a dotted line at 1414°F . This behavior differs from those at the A_1 and A_2 temperatures in that it does not involve a phase change. In the neighborhood of 1414°F and up to about 1454°F there is a gradual magnetic change, the ferrite being ferromagnetic below this temperature range, and paramagnetic above. The change is also accompanied by a heat effect. This A_s change is of little or no significance in regard to the heat treatment of steel.

The Effect of Alloys on the Equilibrium Diagram—The iron-carbon diagram may, of course, be profoundly altered by alloying elements, and, therefore, its application should be limited to plain carbon and low-alloy steels. The most important general effects of the alloying elements may be listed as follows:

1. The number of phases which may be in equilibrium is no longer limited to two as in the iron-carbon diagram.
2. The temperature and composition range, with respect to carbon, over which austenite is stable may be increased or reduced.
3. The eutectoid temperature and composition may be changed.

The alloying elements may be divided generally into two classes in respect to the second effect: those which enlarge the austenite field and those which reduce it. The elements which enlarge this field include manganese, nickel, cobalt, copper, carbon and nitrogen. Be-

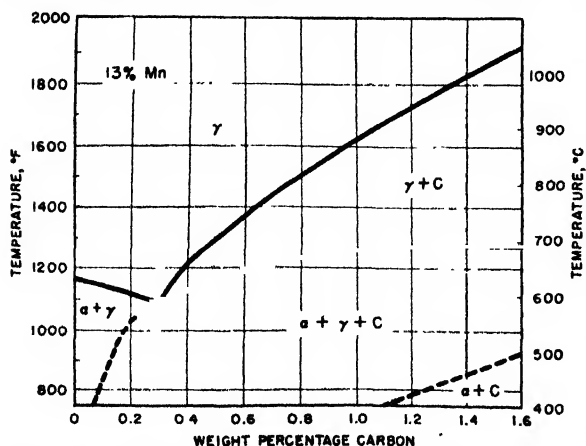


FIG. 42-13. Section of the Fe-C-Mn equilibrium diagram at 13 per cent manganese content. From "Metals Handbook" (American Society for Metals), 1948 Edition; page 1252.

cause of this characteristic, elements of this type are descriptively known as **austenite formers**. Figure 42-13, which shows the 13 per cent manganese section of the iron-carbon-manganese equilibrium diagram, is illustrative of the effect of elements of this type. The large field in which austenite is stable, the lowering of the eutectoid temperature and carbon content, and the three-phase field in which alpha iron, austenite and carbides exist in equilibrium should be noted.

The commoner elements which decrease the "size" of the austenite field include chromium, silicon, molybdenum, tungsten, vanadium, tin, columbium, phosphorus, aluminum and titanium. Such elements are known as **ferrite formers**. Figure 42-14 showing the

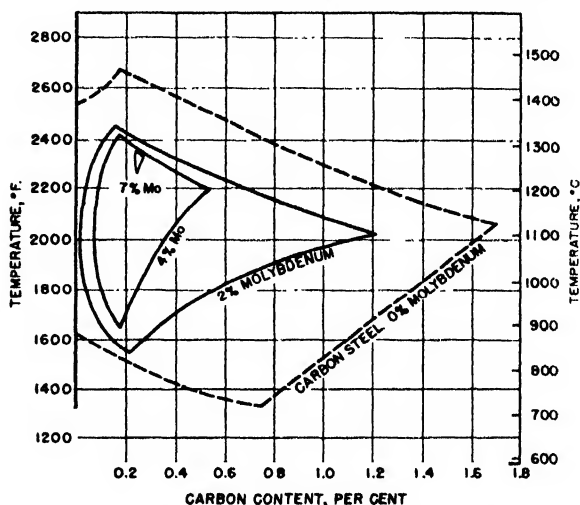


FIG. 42-14. The effect of molybdenum on the composition and temperature range over which austenite is stable (After Bain.)

effect of molybdenum on the composition and temperature range over which austenite is stable will serve to illustrate the effect of alloys of this type. These steels will likewise have the three-phase zone in which austenite, ferrite and carbides will be in equilibrium at temperatures below the austenite field.

The effect of the elements on the eutectoid temperature and composition is summarized in Figure 42-15. It will be noted that manganese and nickel lower the eutectoid temperature while the other elements generally raise it. All of the elements seem to lower the eutectoid carbon content.

Grain Size—As described above, when a piece of steel is heated above the critical temperature, the ferrite and carbide react with one another to form austenite. The austenite is a crystalline phase differing distinctly from either the ferrite or carbide from which it is formed. Like any metal composed of a solid solution, it exists in the form of polyhedral grains. The reaction which forms austenite begins at a number of points in the interface of the carbide and ferrite. Each of the little islands of austenite grows until finally it reaches its similarly growing neighbors. As the temperature above the critical increases, further grain growth occurs, presumably by encroachment of a grain into adjacent grains. The final austenite grain size is, therefore, a function of the temperature above the critical to which it is heated. This grain growth may, however, be inhibited by carbides which dissolve slowly or remain undissolved in the austenite or by a suitable dispersion of non-metallic inclusions. Hot working refines the coarse grain size formed by heating to the relatively high temperatures used in

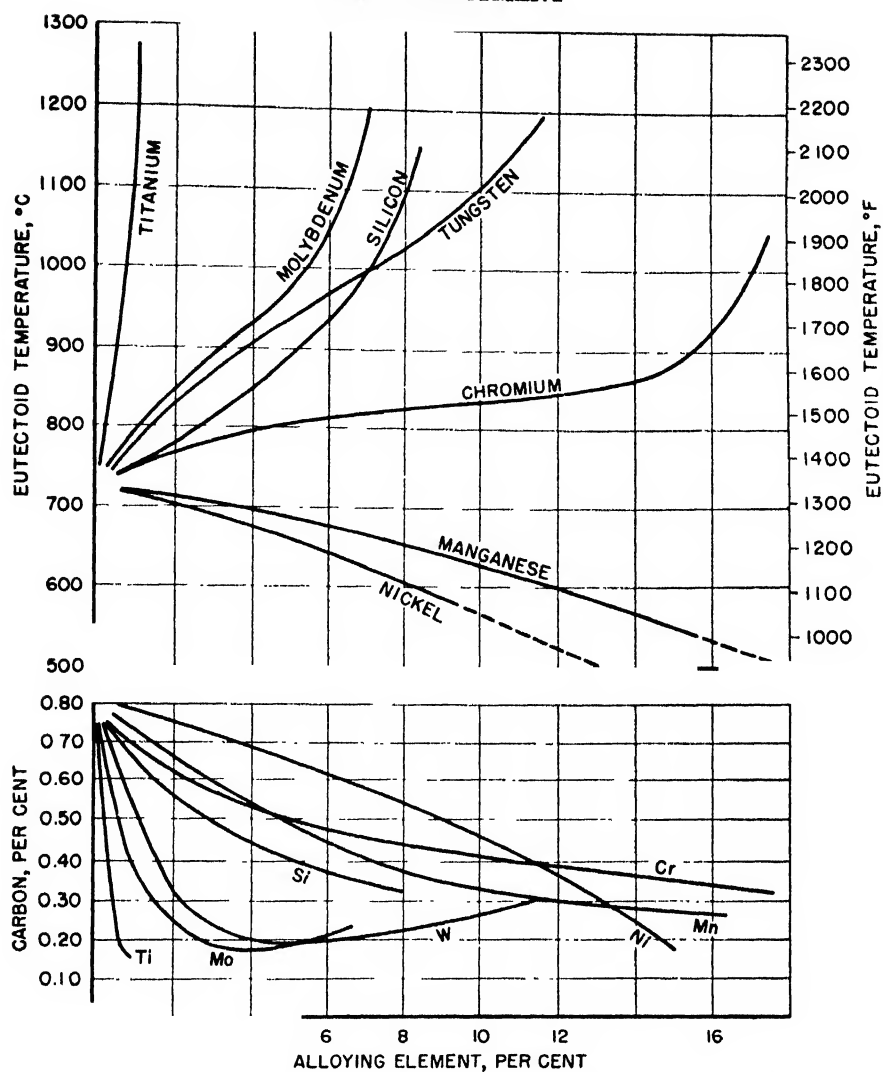


FIG. 42-15. Eutectoid composition and temperature as influenced by several alloying elements. (After Bain.)

forging or rolling and the grain size of hot-worked products is determined largely by the finishing temperature, that is, the temperature at which the final stage of the hot-working process is carried out.

Grain Size and Properties—The coarseness of the ferritic and pearlitic “grains” in the cooled steel reflects the grain size of the austenite prior to its transformation and the properties of the product are profoundly influenced by its grain size. The general effects of the austenite grain size are summarized in Table 42—II.

Determination of Grain Size—Grain size is customarily determined by viewing a polished plane section, prepared in such a way as to delineate the grain boundaries, under a microscope. The grain size is then determined by comparing the size of the outlined grains with a standard chart,* known as the A.S.T.M. standard grain-size chart. Two of the three most common methods of delineating the grain boundaries are to cool

* Metals Handbook (1948), pages 401-403. American Society for Metals, Cleveland, Ohio.

Table 42—II. Trends in Heat-Treated Products

Property	Coarse-Grain Austenite	Fine-Grain Austenite
Hardenability	Deeper Hardening	Shallower Hardening
Toughness	Less Tough	Tougher
Distortion	More Distortion	Less Distortion
Quench Cracking	More Prevalent	Less Prevalent
Internal Stress	Higher	Lower
For Annealed or Normalized Products		
Machinability (Rough)	Better	Inferior
Machinability (Fine Finish)	Inferior	Better

Table 42—III. Comparison of Systems for Reporting Grain Size

A.S.T.M. No.	Grains per sq. in. of Image at 100 X	Grains per sq. mm.	Grains per cu. mm.
—3	0.06	1	0.7
—2	0.12	2	2
—1	0.25	4	5.6
0	0.5	8	16
1	1	16	45
2	2	32	128
3	4	64	360
4	8	128	1020
5	16	256	2900
6	32	512	8200
7	64	1024	23000
8	128	2048	65000
9	256	4096	185000
10	512	8200	520000

the steel from its austenitizing temperature in such a way that the shape of the original austenite grains is outlined by (1) a grain-boundary network of ferrite in the case of hypoeutectoid steels, or (2) a grain-boundary network of cementite in the case of hyper-eutectoid steels. In the third method (3) the steel is

quenched to its full hardness and a polished section is etched with a special reagent which will disclose orientation differences in the martensite grains formed from the austenite on quenching. The index numbers of the standard grain-size chart represent by far the most common method of expressing grain size. These numbers are tabulated in Table 42—III together with the corresponding dimensions.

Fine- and Coarse-Grain Steels—As was mentioned previously, austenitic-grain growth may be inhibited by undissolved carbides or by a suitable distribution of non-metallic inclusions. Steels of this type are commonly referred to as inherently fine-grained or simply as fine-grained steels, while steels which are free from these grain growth inhibitors are known as coarse-grained steels.

The general pattern of grain coarsening in steels of the coarse- and fine-grained types on heating above the critical temperature is illustrated in Figures 42—16 and 42—17. It will be noted that the coarse-grained steel coarsens gradually and consistently as the temperature is increased, while the fine-grained steel coarsens only slightly, if at all, until a certain temperature is reached, above which an abrupt coarsening occurs. This temperature is known as the **coarsening temperature**. It should further be noted that either type of steel may be heat treated so as to be either fine or coarse grained, and as a matter of fact, at temperatures above its coarsening temperature, the fine-grained steel will usually

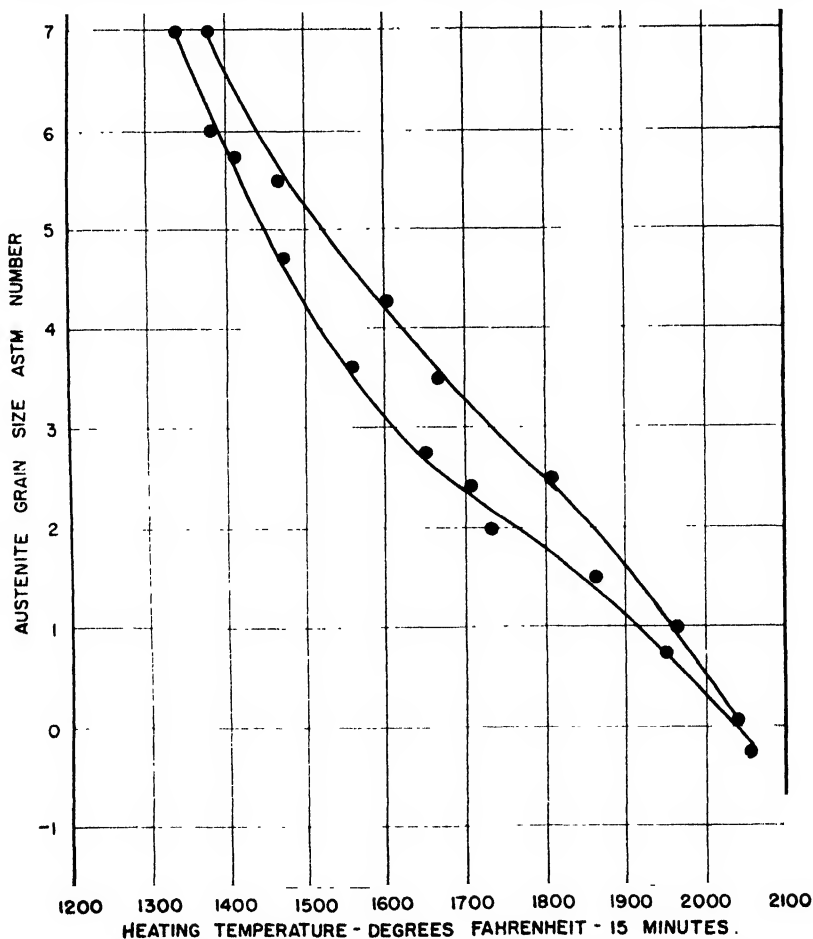


FIG. 42—16. Grain size as a function of austenitizing temperature for inherently coarse-grained steels. (After Bain.)

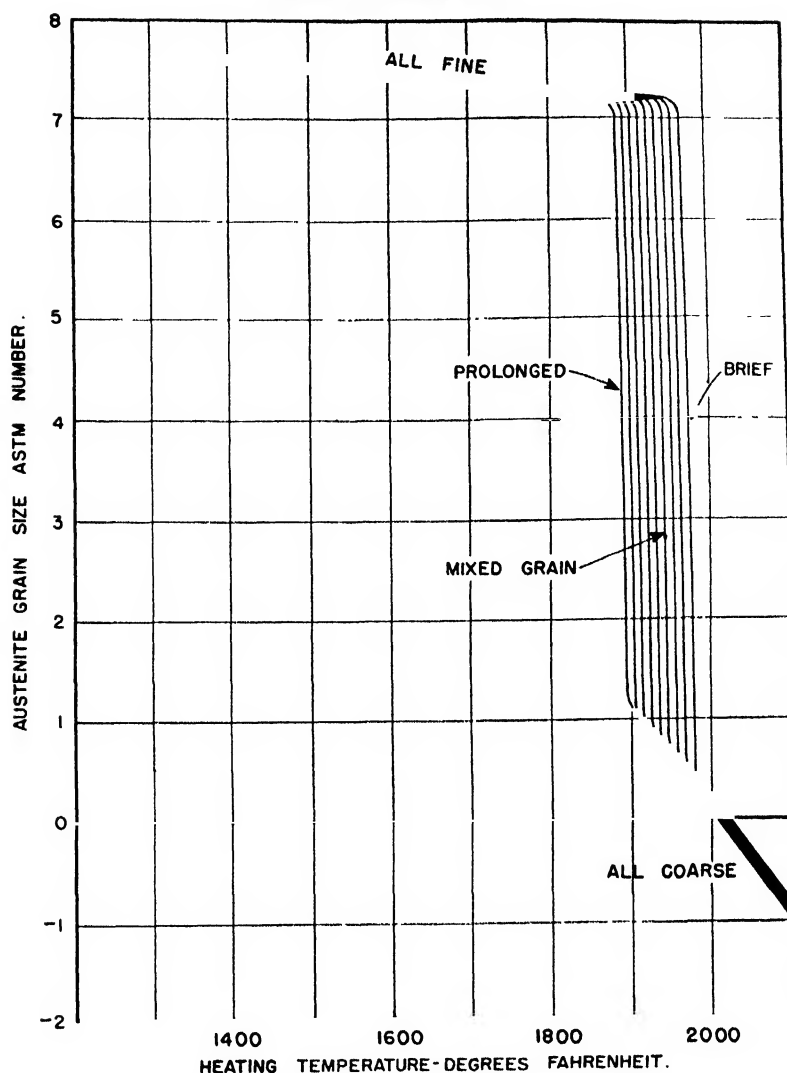


FIG. 42-17. Grain size as a function of austenitizing temperature for an inherently fine-grained steel (schematic). (After Bain.)

exhibit a coarser grain size than the coarse-grained steel at the same temperature.

The usual method of making steels which remain fine grained at 1700° F (A.S.T.M. E19-46) involves the judicious use of aluminum deoxidation. The inhibiting agent in such steels is generally conjectured to be a submicroscopic dispersion of aluminum nitride or, perhaps at times, aluminum oxide.

THE TRANSFORMATION OF AUSTENITE

Thus far, this chapter has largely been concerned with equilibrium conditions. Under equilibrium conditions, that is, with very slow cooling, it has been shown that austenite transforms to pearlite when it is cooled below the A_1 critical temperature, and at a temperature only a little below the A_{e1} temperature. When more rapidly cooled, however, this transformation is depressed and does not occur until a lower temperature is reached. The faster the cooling rate, the lower is the temperature at which transformation occurs. Furthermore, the nature of the ferrite-carbide aggregate formed when the austenite transforms varies markedly with the temperature of transformation and the properties are found to vary correspondingly. Thus, heat treatment is seen to involve

a controlled supercooling of austenite, and in order to take full advantage of the wide range of structures and properties which this permits, a knowledge of the transformation behavior of austenite and the properties of the resulting aggregates is essential.

Isothermal Transformation Diagrams—The transformation behavior of austenite can best be studied by a technique developed at the United States Steel Corporation Fundamental Research Laboratory in 1930. This involves studying the transformation behavior at a series of temperatures below A_1 , by quenching small samples to the desired temperature in a liquid bath, allowing them to transform isothermally and following the progress of the transformation metallographically or by dilatometric measurements. This procedure not only gives a picture of the rates of transformation at the various temperatures, but also furnishes information as to the metallographic structures characteristic of the various temperatures of transformation and permits a determination of the properties of these individual microstructures.

The general pattern of this transformation behavior at a single temperature is shown in Figure 42-18. It will be seen that there is first a period of time before any

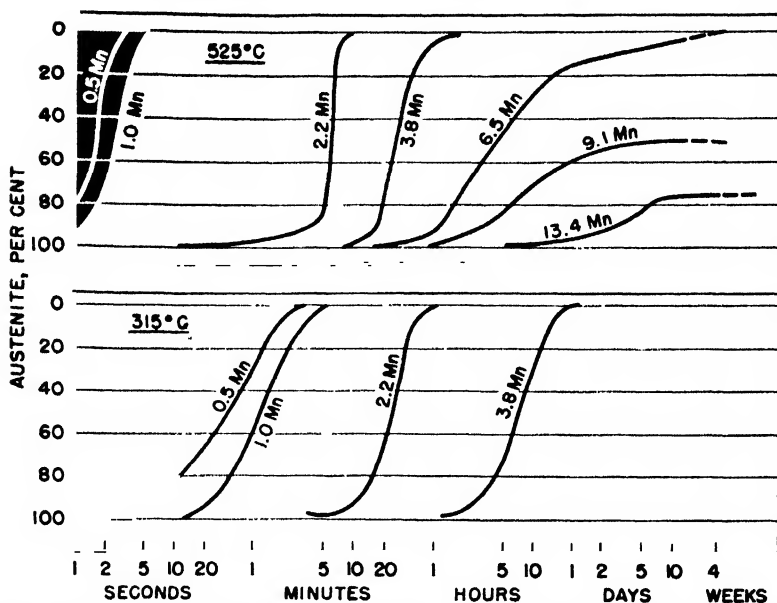


FIG. 42-18. Transformation behavior at a single temperature in a series of manganese steels of about 0.55% carbon content. (After Bain.)

transformation starts. This period is sometimes spoken of as an incubation period. The incubation period presumably represents the period at the start of transformation when the volume of transformed phase around each nucleus is increasing very slowly. The transformation accelerates as it progresses so that the fastest transformation rate corresponds to about 50 per cent transformation. The rate then slows down again and the transformation goes to completion rather slowly. This general pattern is characteristic, but the rates themselves will vary with the temperature of transformation and, as will be shown later, with the composition and grain size of the austenite.

It has become customary to present data obtained in this manner as a plot, with the times required for the beginning and completion (and a few other stages) of the transformation as the abscissa and temperature as the ordinate. Such a diagram is known as an isothermal transformation diagram.

An isothermal transformation diagram for an eutectoid plain carbon steel, together with the hardness values and microstructures characteristic of transformation at the various temperature levels, is shown in Figure 42-19. It will be noted that the diagram has a nose or temperature of most rapid transformation at about 1000° F. The transformation rate at temperatures near the A_1 is very slow and it is likewise relatively slow at a lower temperature range.

Transformation to Pearlite—Transformation over the temperature range of about 1300° to 1000° F (in carbon and low-alloy steels) forms pearlitic microstructures and the characteristic lamellar appearance of these structures is apparent in the photomicrographs. It will also be noted that as the transformation temperature decreases, the lamellae become more closely spaced, so that as transformed at 1000° F they can hardly be resolved by the microscope. The hardness is also seen to increase as the lamellar spacing becomes smaller.

Transformation to Bainite—Transformation to bainite occurs over the temperature range of about 1000° to 450° F. The bainitic microstructures differ markedly from pearlitic in that they are acicular in nature. Here again, the hardness increases as the transformation tem-

perature decreases, though the bainite formed at the highest possible temperature is often softer than pearlite formed at a still higher temperature.

Transformation to Martensite—Transformation to martensite, which in this steel occurs at temperatures below 450° F, differs from transformation to pearlite or bainite in that it is not time dependent, but occurs almost instantly during cooling and the percentage of transformation is dependent only on the temperature to which it is cooled. Thus, in this steel, transformation to martensite will start on cooling to 450° F (designated as the M_s temperature), will be 50 per cent complete on cooling to about 300° F, and will be essentially completed at about 200° F (designated as the M_f temperature). The microstructure of martensite is likewise acicular but it is generally lighter etching than bainite. It is the hardest of the transformation products of austenite. It is possible to form a little martensite at, say, 425° F and then to cause bainite to form thereafter isothermally.

MICROSTRUCTURE AND MECHANICAL PROPERTIES

The dependence of the properties of steel upon its constitution has been emphasized in this chapter and, as would be expected, the properties of steel vary with the temperature at which the austenite transforms in accordance with the corresponding microstructural changes.

The microstructures discussed above fall into three general classes: pearlite, bainite, and martensite. In discussing the relationships between microstructure and properties, a fourth class of microstructure, tempered martensite, must also be considered. This is the structure formed when martensite is reheated to a subcritical temperature after quenching. Its microscopic appearance in a polished and etched specimen is illustrated in Figure 42-20. The general effect of tempering martensite is to precipitate and coagulate the carbide particles, so that tempered martensite microstructures consist of carbide particles dispersed in a ferrite matrix. The steel in this illustration has been tempered at a relatively high temperature and some tendency for the particles to as-

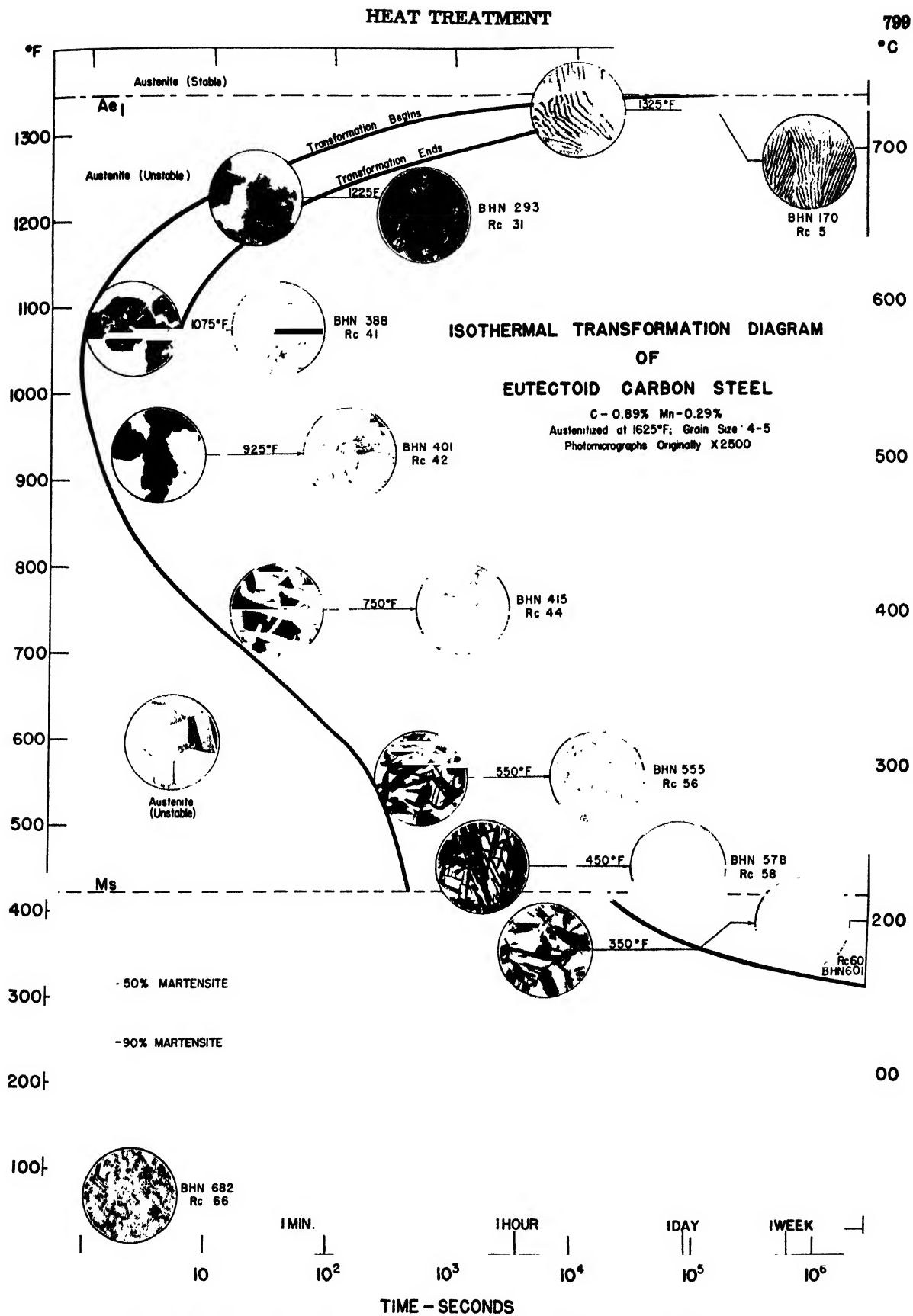


FIG. 42-19. Isothermal transformation diagram for a plain carbon eutectoid

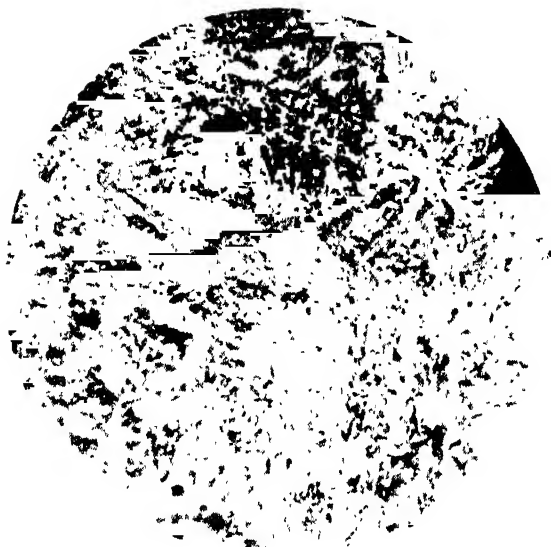


FIG. 42—20. The microstructure of tempered martensite. Tempering temperature, 1022° F (550° C). Magnification: 2000X.

sume a spheroidal form can be noted. This microstructure, consisting of spheroidized or partially spheroidized carbides in a ferrite matrix is, as might be expected, a very favorable one in respect to ductility.

Each of these types of microstructures has characteristic properties typical of the class but the properties of pearlite and bainite also each vary quite widely with transformation temperature.

Properties of Pearlite—In any steel, the pearlites are, as a class, softer than the bainites or martensites. In general, even though softer, they are less ductile than the lower temperature bainites and for a given hardness they will be far less ductile than tempered martensite.

As the transformation temperature decreases within the pearlite range, the inter-lamellar spacing decreases, as was described above, and these "fine" pearlites, formed near the nose of the isothermal diagram, are both harder and more ductile than the "coarse" pearlites formed at higher temperatures. Thus, although, as a class, pearlite tends to be soft and not exceedingly ductile, its hardness and toughness both increase markedly with decreasing transformation temperatures.

Properties of Bainite—In a given steel, bainitic microstructures will generally be found both harder and tougher than pearlite, although the hardness will be lower than that of martensite. Within the class, as with pearlite, the properties generally improve as the transformation temperature decreases and "lower" bainite will compare favorably with, or exceed in toughness, tempered martensite at the same hardness. "Upper" bainite, on the other hand, may be somewhat deficient in toughness as compared with fine pearlite at the same hardness.

The properties of pearlite and bainite in an eutectoid steel are summarized in Figure 42—21.

Properties of Martensite—Martensite is the hardest and likewise the most brittle of the microstructures obtainable in a given steel. The hardness of martensite as a function of carbon content is shown in Figure 42—22. The hardness of martensite, at a given carbon content, varies somewhat with the cooling rate and this figure shows both the maximum hardness obtainable with very rapid cooling and the average hardness values which might be expected in practice.

Although, for some applications, particularly those involving wear resistance, the high hardness of martensite is desirable in spite of the accompanying brittleness, the principal importance of this microstructure is as the starting material for tempered martensite structures, which latter have definitely superior properties.

Properties of Tempered Martensite—Tempered martensitic structures are, as a class, characterized by relatively high toughness at any strength level. Their properties are illustrated in Figure 42—23. This chart designates, within plus or minus 10 per cent, the usual

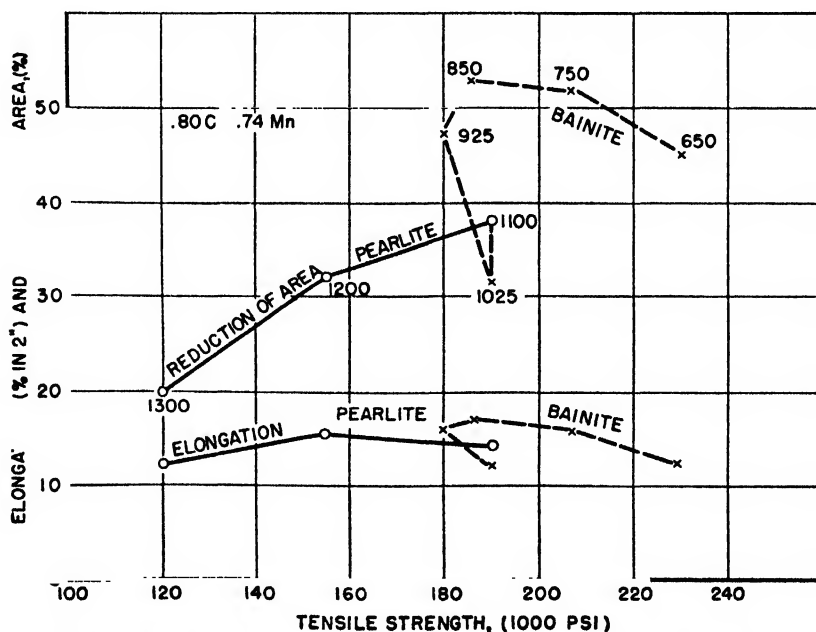


FIG. 42—21. The properties of pearlite and bainite in a eutectoid steel.

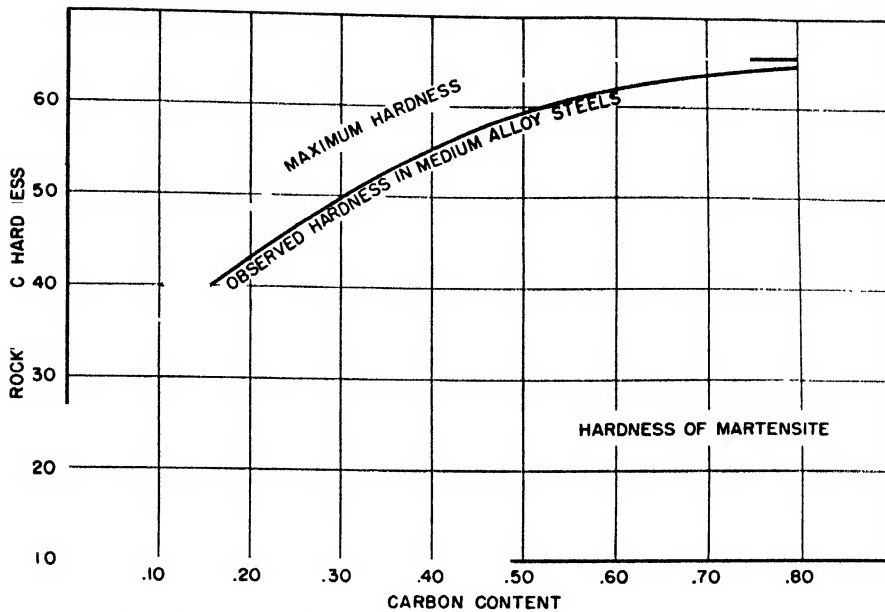


FIG. 42-22. The hardness of martensite as a function of carbon content.

mechanical properties of any steel with this microstructure, regardless of composition. Because of its high ductility at a given hardness, this is the structure that is

aimed for in heat treating for toughness by quenching and tempering.

FACTORS AFFECTING TRANSFORMATION RATES

The major factors affecting the rates of transformation of austenite are its composition, grain size, and homogeneity. In general, increasing carbon and alloy content tend to decrease transformation rates. Increasing the grain size of the austenite likewise tends to decrease transformation rates.

Effect of Carbon Content—The effect on the transformation rates of decreasing the carbon content of a plain carbon steel is illustrated by comparison of Figure 42-24, which shows the isothermal transformation diagram for a 0.35 per cent carbon steel, with that for the eutectoid steel shown in Figure 42-19. It will be noted that the effect of lowering the carbon content has been to shift the lines of the diagram to the left, that is, toward more rapid transformation rates. This diagram differs from that for the eutectoid steel also in that the transformation to pearlite is preceded by a precipitation of ferrite and the diagram, therefore, shows a line designating the time for the initiation of this ferrite precipitation at the temperature levels wherein this separation precedes the formation of pearlite.

Effects of Alloys—Figure 42-25 shows an isothermal transformation diagram for a 0.35 per cent carbon, 1.85 per cent manganese steel. Comparing this with the lower manganese steel (Figure 42-24), it will be noted that the entire curve has been displaced to the right; that is, transformation at all temperature levels starts later and is slower to go to completion. This is characteristic of the effect of alloys in solution in the austenite; in general, increased alloy content delays the start of transformation and increases the time for its completion.

Although alloy additions tend in general to delay the start of transformation and to increase the time for its completion, they differ greatly, nevertheless, in both the magnitude and the nature of their effects. Figure 42-26 represents the isothermal transformation diagram for a 0.33 per cent carbon, 0.45 per cent manganese, 1.97 per cent chromium steel. By comparison with the plain car-

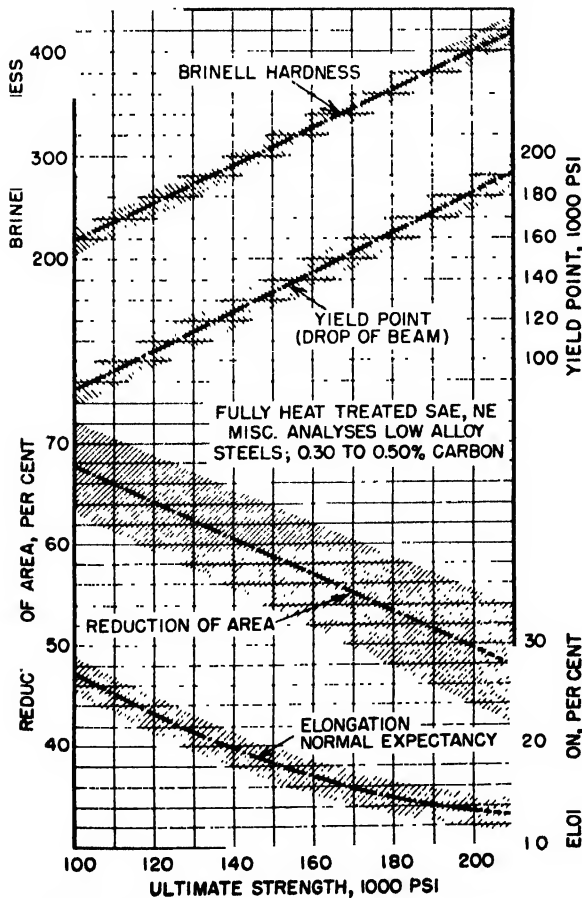


FIG. 42-23. The properties of tempered martensite.

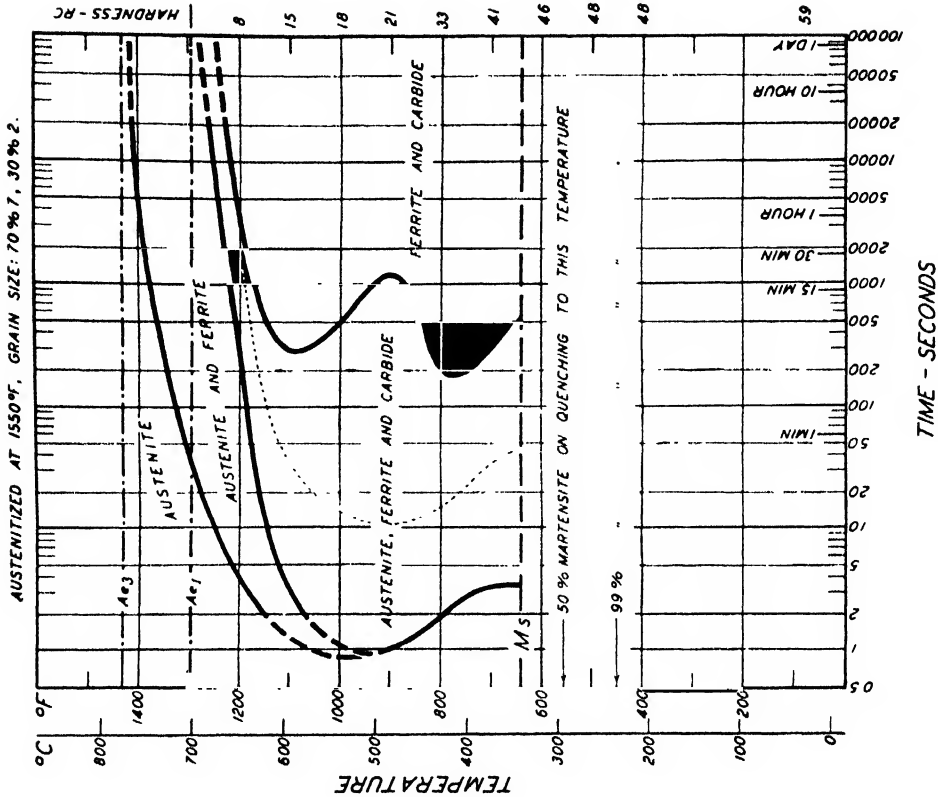


FIG. 42-25. Isothermal transformation diagram for a 0.35% carbon, 1.85% manganese steel.

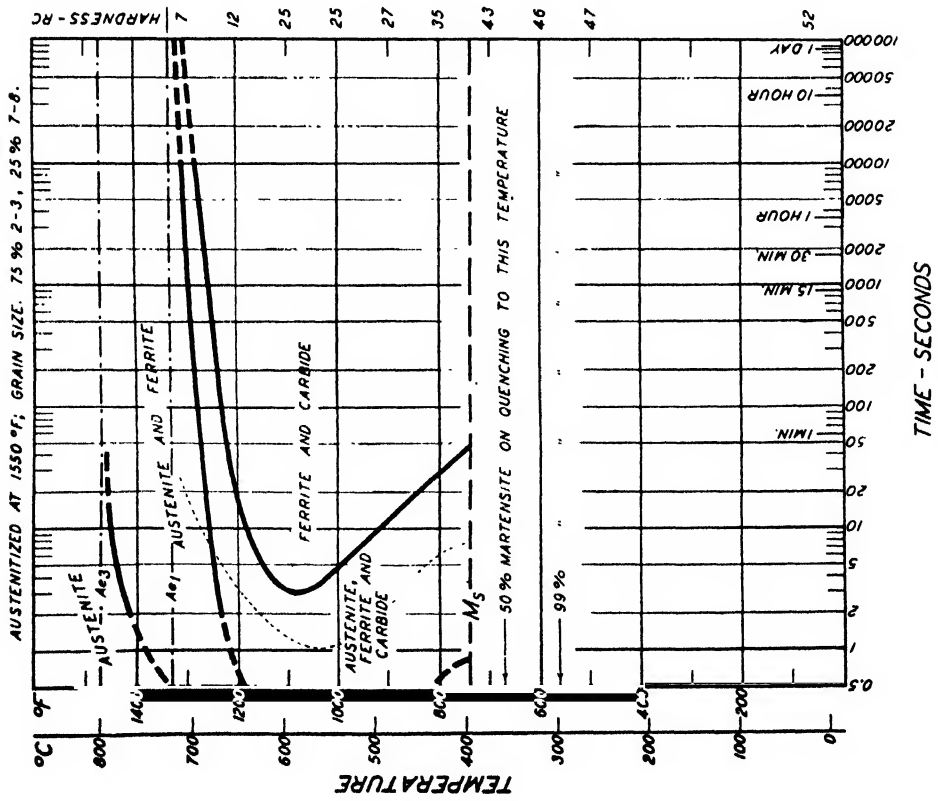


FIG. 42-24. Isothermal transformation diagram for a 0.35% carbon, 0.37% manganese, plain carbon steel.

HEAT TREATMENT

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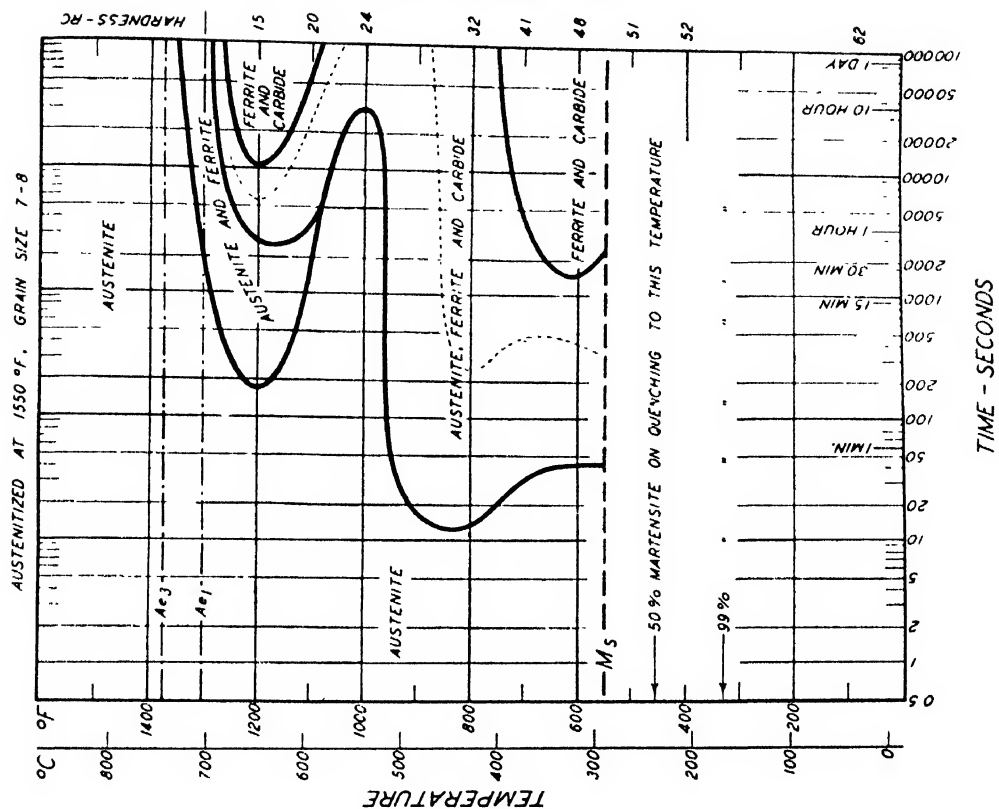


FIG. 42-27. Isothermal transformation diagram for an SAE 4340 steel, containing 0.42% carbon, 0.78% manganese, 1.79% nickel, 0.80% chromium and 0.33% molybdenum.

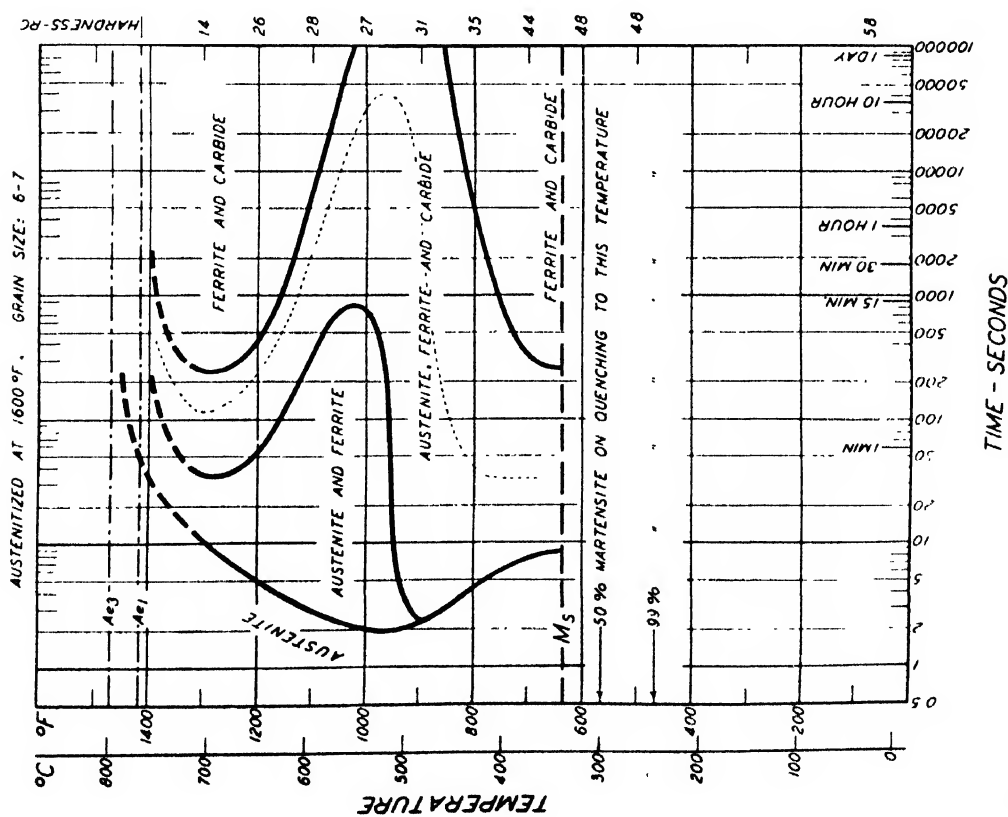


FIG. 42-26. Isothermal transformation diagram for a 0.33% carbon, 0.45% manganese, 1.97% chromium steel.

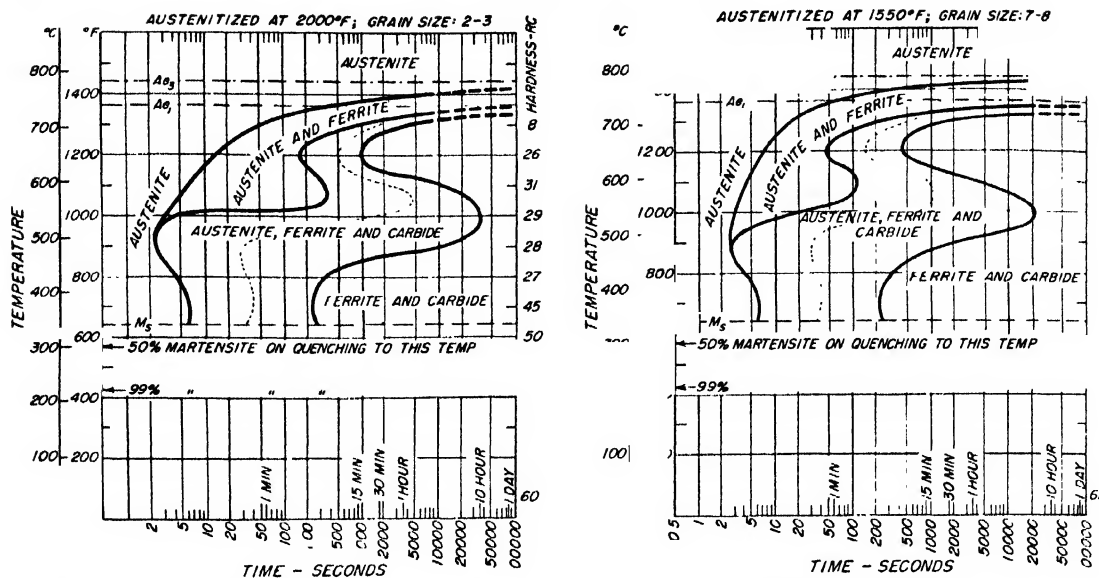


FIG. 42-28. Comparison of the effect of grain size on the isothermal transformation of an SAE 4140 alloy steel, containing 0.37% carbon, 0.77% manganese, 0.98% chromium and 0.21% molybdenum.

bon steel (Figure 42-24), it will be noted that the effect of the chromium has been, not only to move the curve to the right, but also to change the shape of the curve. The time for beginning of transformation in the pearlite region has been greatly increased, while that for the beginning in the bainite region has been only moderately increased. Thus the diagram now has two "noses" (or time minima), one in the temperature region of transformation to pearlite and the other in the bainite region.

Figure 42-27 represents the isothermal transformation diagram for a more-complex alloy steel. This steel (SAE 4340) contains 0.42 per cent carbon, 0.78 per cent manganese, 1.79 per cent nickel, 0.80 per cent chromium and 0.33 per cent molybdenum. It will be noted that the effect of the addition of moderate amounts of these several alloying elements has been to displace the curve even farther to the right than that of the 2 per cent chromium steel (Figure 42-26). This is characteristic of the effect of alloys; relatively small amounts of several alloying elements are more effective in decreasing transformation rates than are larger amounts of a single alloy, i.e., more retarding than if they were merely additive.

Summarizing the effects of alloying elements on transformation behavior, it can be seen that the general effect of increasing the alloy content is to delay both the start and the completion of transformation and that the effect of alloy additions is cumulative. The effects of alloying elements, however, differ greatly both in magnitude and in specific effects on transformation in different temperature regions, so that a precise prediction of the effect of a given alloy combination is not yet quite possible.

Effect of Grain Size—The effect of increasing the grain size of the austenite is similar to that of alloys; it delays both the start and completion of the transformation. This is illustrated by Figure 42-28, which shows the isothermal transformation diagrams for the same alloy steel with both fine- and coarse-grained austenite.

Effect of Homogeneity of Austenite—The general effect of inhomogeneous austenite will be to speed up the start of transformation. This occurs because the initial

transformation will occur in the portions of the austenite which are "leaner" in alloy. In addition, undissolved carbides may act as nuclei for transformation, thereby hastening the start of transformation.

TRANSFORMATION ON CONTINUOUS COOLING

The preceding section has described the manner in which the microstructure and, therefore, the properties of steel vary with the temperature of transformation, and has shown how the isothermal transformation behavior governing these microstructural changes can be studied and depicted as isothermal transformation diagrams. The factors affecting transformation characteristics have been enumerated and the nature of their effects described. The composition of the steel, particularly in respect to the alloying elements, has been shown to be the major factor, and the effects of austenite grain size and homogeneity have also been described. Thus, the basic information about the transformation behavior of a steel is fully described by the isothermal transformation diagram.

This basic information tells what structure is formed at each reaction temperature, if the cooling is interrupted so that the reaction goes to completion at that temperature. The information is equally useful for interpreting behaviors when the cooling proceeds directly without interruption, as is the case in the industrial processes of annealing, normalizing and quenching. In these industrial processes, the time at a single temperature is generally insufficient for the reactions to go to completion at such a single temperature; instead, the end structure consists of an association of microstructures which individually were formed at successively lower temperatures as the piece cooled. But the tendency to form the several structures is still explained by the isothermal diagram.

The final microstructure after continuous cooling will obviously depend upon the times spent at the various transformation temperature ranges through which the piece is cooled. The transformation behavior on continuous cooling thus represents an integration of these times and this integration can be carried out by the

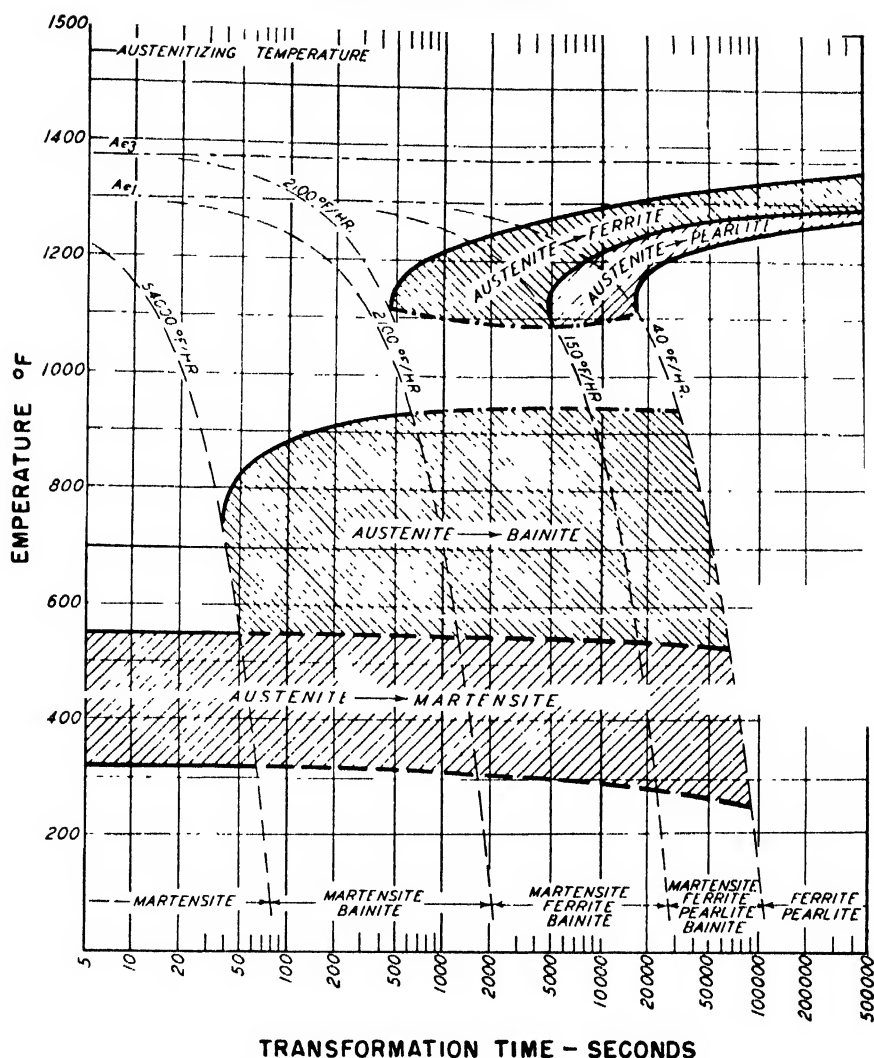


FIG. 42—29. Continuous-cooling transformation diagram for a 4340-type alloy steel, with superimposed cooling curves illustrating the manner in which transformation behavior during continuous cooling governs final microstructures.

method developed by Grange and Kiefer at the U. S. Steel Fundamental Research Laboratory. By this method, a continuous-cooling diagram generally similar to the isothermal transformation diagram, but depicting the transformation behavior on cooling at a series of constant cooling rates, can be constructed.

Such a diagram for an alloy steel is shown in Figure 42—29. This continuous-cooling diagram lies below and to the right of the corresponding isothermal diagram. That is, transformation on continuous cooling will start at a lower temperature and after a longer time than the intersection of the cooling curve and the isothermal diagram would predict, and this displacement is a function of the cooling rate, being larger as the cooling rate increases.

In order to illustrate the manner in which the transformation behavior on continuous cooling will govern the final microstructures, several cooling-rate curves have been superimposed on this diagram. A consideration of the changes occurring during these various cooling cycles will serve to illustrate the manner in which diagrams of this nature can be correlated with heat-

treating practice and used to predict the resulting microstructure.

Considering first the relatively slow cooling rate (less than 40° F per hour), the steel will be cooled through the regions in which transformation to ferrite and pearlite will occur and these constituents, ferrite and pearlite, will, therefore, make up the final microstructure. This cooling rate corresponds to a slow furnace cool, such as might be used in annealing.

At a somewhat faster cooling rate (40° to 150° F per hour), such as might be obtained on normalizing a large forging, the ferrite, pearlite, bainite, and martensite fields will all be traversed and final microstructure will contain all of these constituents.

At cooling rates of 150° to 2100° F per hour, the pearlite field will be missed entirely and the resulting microstructures will consist of ferrite, bainite, and martensite. This, therefore, is the microstructure to be expected on normalizing small or moderate sections of this steel.

Finally, on cooling at rates of from 2100° to 54,000° F per hour, the microstructure will be free of proeutectoid ferrite and will consist largely of bainite with a small

amount of martensite present. A cooling rate of at least 54,000° F per hour is necessary to obtain the fully martensitic structure desired as a starting point for tempered martensite on quenching and tempering.

Thus, the final microstructure, and therefore, the

properties of a steel, are dependent upon the transformation behavior of the austenite and on the cooling conditions, and can be predicted if these factors are known, or can be governed by controlling either or both of these factors.

SECTION 2

HARDENABILITY

The one attribute of a steel which is certainly of the greatest significance to the heat treater is its capacity for hardening, commonly referred to as its hardenability. This attribute has a two-fold significance; it is important, not only in relation to the attainment of a higher hardness or strength level by heat treatment, but also in relation to the attainment of a high degree of toughness through heat treatment to a desirable microstructure, usually tempered martensite or lower bainite. As a matter of fact, the attainment of toughness is the most important, since the attainment of a certain high strength level may often have little significance unless accompanied by a sufficient toughness to meet service requirements.

It should be clearly understood that hardenability refers to the depth of hardening or to the size of piece which can be hardened under given cooling conditions and not to the maximum hardness that can be obtained in a given steel. This maximum hardness, as previously described, is dependent almost entirely upon the carbon content (Figure 42—22), while the hardenability (depth of hardening) is, in general, far more dependent upon the alloy content and grain size of the austenite than upon the carbon content.

Relationship of Hardenability to Transformation Rates—In the preceding discussion, it was shown that, in general, the hardness of steel increases as the transformation temperature decreases. It was also shown that the lower-temperature transformation products, lower bainite and martensite, when tempered, exhibit superior properties in respect to ductility and toughness at a given strength level. It is apparent that, in order to realize the superior properties of these low-temperature transformation products, prior transformation at a higher temperature to softer products must, insofar as possible, be prevented. This means that the steel must be cooled through these high-temperature transformation ranges at a rapid enough rate that transformation does not occur, even at the nose of the transformation diagram. This rate, which will just permit transformation to martensite without any prior transformation at a higher temperature is known as the critical cooling rate for martensite, and furnishes one method for expressing hardenability. It can be readily ascertained from the continuous cooling diagram. For example, in the steel of Figure 42—29, the critical cooling rate for martensite is 54,000° F per hour or 15° F per second.

How Hardenability Is Expressed and Measured—Although the critical cooling rate can be used to express hardenability, it has the disadvantage that, in practice, cooling rates are ordinarily not constant, but vary during the cooling cycle. This is particularly true of liquid quenching, in which case, the cooling rate is always slower as the temperature of the cooling medium is approached, and is also greatly affected by the presence of a vapor phase in the earlier part of the quenching cycle. Furthermore, as already mentioned, hardenability refers to depth of hardening.

In order to facilitate the application of hardenability measurements to practice, it is, therefore, customary to express hardenability in terms of depth of hardening in

a standardized quench. The quenching condition used in this expression is a hypothetical one, in which the surface of the piece is assumed to come instantly to the temperature of the quenching medium. This is known as an ideal quench, and the diameter of a round which will just quench to the desired microstructure, or corresponding hardness value at the center, in an ideal quench is known as the ideal diameter (symbol D_i). Since the cooling rate relationships between the ideal quench and other quenching conditions are known, hardenability values in terms of ideal diameter can be used to predict the size of round which will harden in any quench, the characteristics of which are known, or similarly, if the diameter which will just harden to the center in a standardized quench is known, this can be converted into the ideal diameter value used to express hardenability.

The most direct method of measuring hardenability in terms of ideal diameter is by quenching a cylinder series. In this method, a series of bar sizes are quenched under identical conditions. These bars should have a length at least four times the diameter. They are then sectioned, etched, and cross-section hardness measurements made. The depth of hardening of each of the bars is determined by the point at which the etching characteristics change, which corresponds to a microstructure of 50 per cent martensite, or by the corresponding hardness value. This microstructure of 50 per cent martensite is a very commonly used criterion of hardenability because of the ease with which it may be located. The diameter of the bar in this series which just hardens to the center is noted and this is known as the critical diameter (D) for the series.

As mentioned above, this critical diameter value can be translated into the fundamental terms of ideal diameter (D_i) by the charts of Figures 42—30 and 42—31. In order to make this conversion, however, it is necessary to evaluate the factor expressing the severity of the quench (H factor). This factor can be evaluated quite accurately from the results of the cylinder series by a method to be described, but typical values of the H coefficient are tabulated in Table 42—IV.

Table 42—IV. Typical Values of the H Coefficient Designating Severity of Quench (H Value)

Agitation	Oil	Water	Brine
None	0.25-0.30	0.9-1.0	2
Mild	0.30-0.35	1.0-1.1	2.0-2.2
Moderate	0.35-0.40	1.2-1.3	
Good	0.40-0.50	1.4-1.5	
Strong	0.50-0.80	1.6-2.0	
Violent	0.80-1.1	4.0	5.0

The chart of Figure 42—32 enables the H value to be accurately evaluated from the results of a cylinder series. To use this chart, the values of $\frac{D_i}{D}$ (ratio of unhardened core diameter to diameter of the piece) for

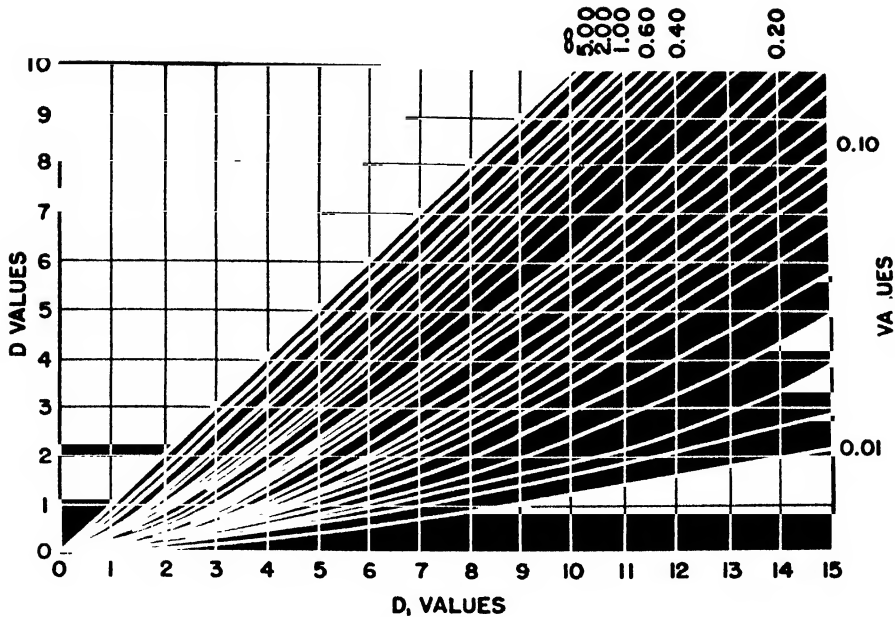


FIG. 42-30. Relationships among ideal diameter, critical diameter and severity of quench. See also Figure 42-31.

the series are plotted against the diameter on a piece of transparent material, using the same coordinates as the chart. The resulting curve is then shifted until it matches one of the curves of the chart. The H value is then determined by dividing the H D value corresponding to any of the points by its diameter.

The following example is illustrative of the use of the charts of Figures 42-31 and 42-32.

Assume that the cylinder section being studied has indicated a critical diameter of 1.2 inches and that the H value of the quench has been found to be 0.4. The horizontal line of Figure 42-31 at 1.2 inches is then followed across until it intersects the curve corresponding to an H value of 0.4. The ideal diameter is then read for the

vertical line at this point and is found to be 2.6 inches. In a similar manner, the size of rounds which could be hardened under different quenching conditions for a steel of this hardenability can be determined from this chart. Thus a water quench with an H value of 1, would permit hardening in a 1.7-inch diameter bar, while air cooling with an H value of 0.02 would permit hardening in only a ¼-inch section.

The cylinder series method, just described, is the most direct method of measuring hardenability, but because of numerous advantages, the end-quench test, developed by Jominy and Boegehold, is the hardenability test which is now by far the most generally accepted and used. In this test, a cylindrical specimen one inch in di-

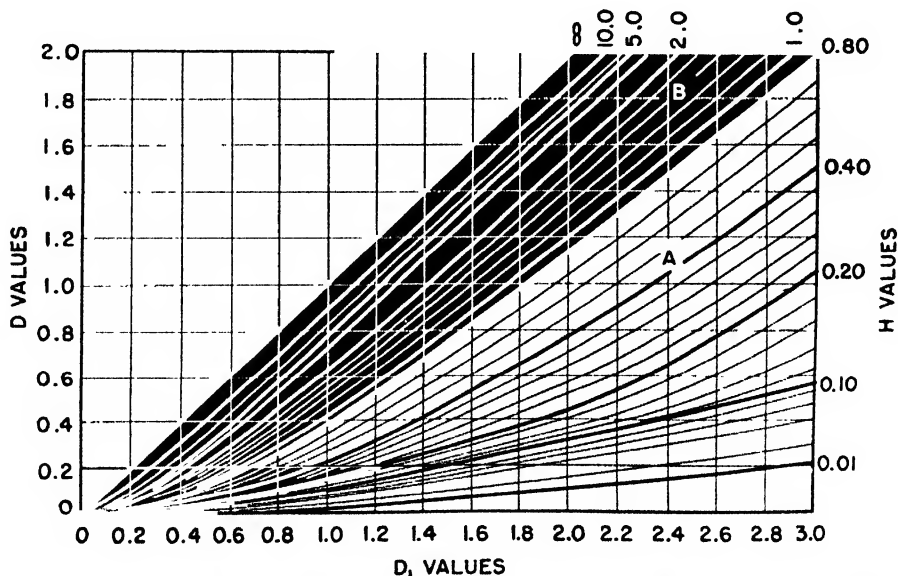


FIG. 42-31. Enlargement of the portion of Figure 42-30 for D values between 0 and 2.1, and D_c values from 0 to 3.0.

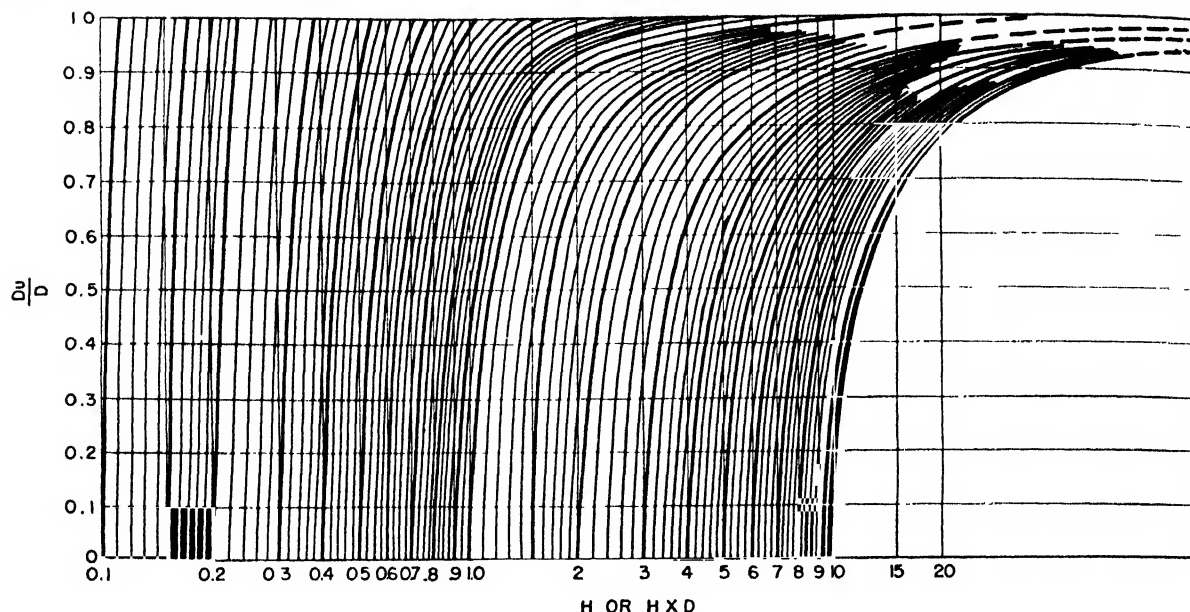


FIG. 42-32. Curves for estimating severity of quench (H) values from a cylinder series.

ameter and four inches long is heated to the desired hardening temperature and quenched in a fixture by a stream of water impinging upon only one end. The bar is then ground on two opposite sides to a depth of 0.015 inch below the surface and hardness measurements made at $\frac{1}{16}$ -inch intervals along the length of the specimen. The hardenability is expressed as a curve of hardness versus distance from the quenched end of the specimen. Figure 42-33 illustrates the type of quenching fixture used for this test and a typical end-quench hardenability curve is shown in Figure 42-34. Standard procedures for this test have been established by the American Society for Testing Materials and the Society of Automotive Engineers and the reader is referred to the publications of these societies for the details of the testing procedures.

This test furnishes a method of applying a continuous series of varying cooling rates to a single specimen, and,

since these rates are known, the results can be converted to hardenability values in terms of ideal diameter. The curve used for this conversion is shown in Figure 42-35. To use this curve, the distance along the end-quench bar to the desired microstructure, or corresponding hardness value, is noted and the ideal diameter corresponding to this distance is read from the curve. This ideal diameter value may then be converted into terms of bar size which can be hardened under any given quenching conditions, by the methods described above.

Hardenability and Heat Treatment—It has been emphasized in the preceding sections of this chapter that the most desirable microstructural constituents, from the standpoint of strength and toughness, are those involving transformation at the lower temperature levels, —lower bainite and tempered martensite. In order to obtain these desirable structures, the transformation rates must be slow enough, or in other words, the

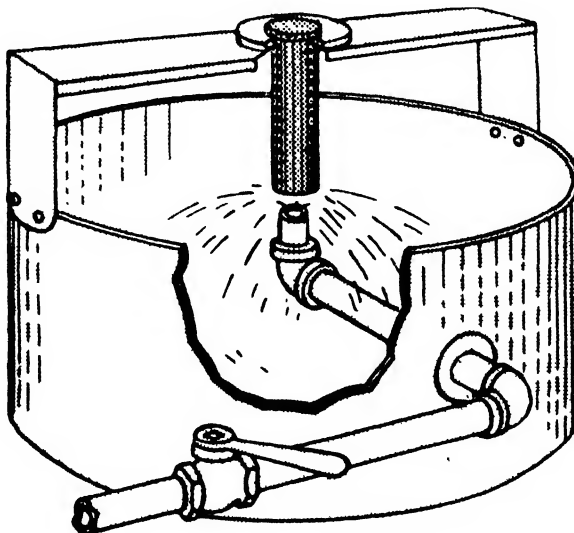


FIG. 42-33. Quenching fixture for end-quench test.

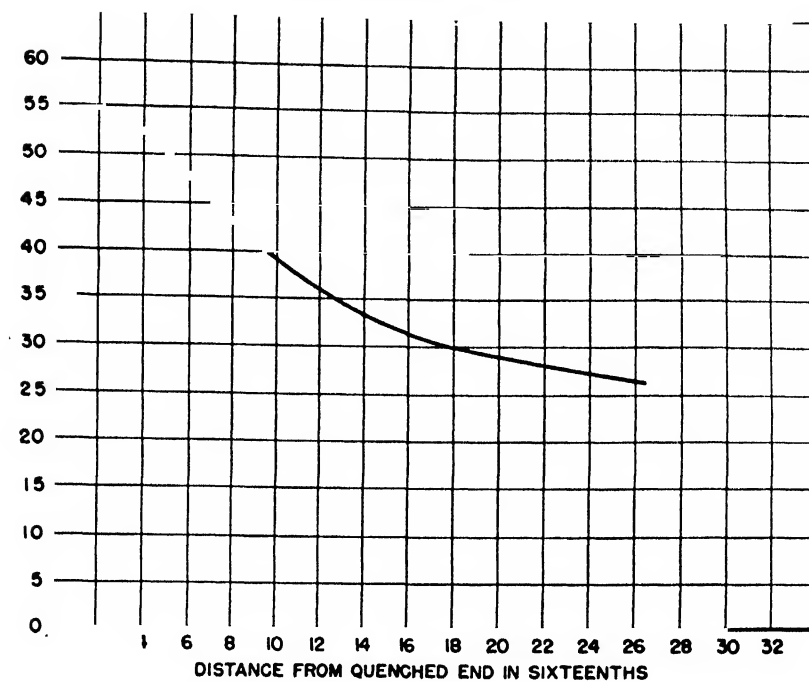


FIG. 42-34. Typical end-quench hardenability curve.

hardenability must be high enough, to prevent prior transformation at a high temperature during the cooling cycle. The results of hardenability measurements serve to establish the limiting conditions in terms of cooling rates or quenching practices necessary to meet this requirement. Similarly, if the heat-treating practice and cooling conditions have been established and evaluated,

the hardenability necessary to obtain the desired microstructure may be determined by the methods described above.

Thus, it is seen that, in general, the suitability of a steel for a given heat treatment practice or the suitability of a heat treatment practice for a given steel is determined largely by its hardenability.

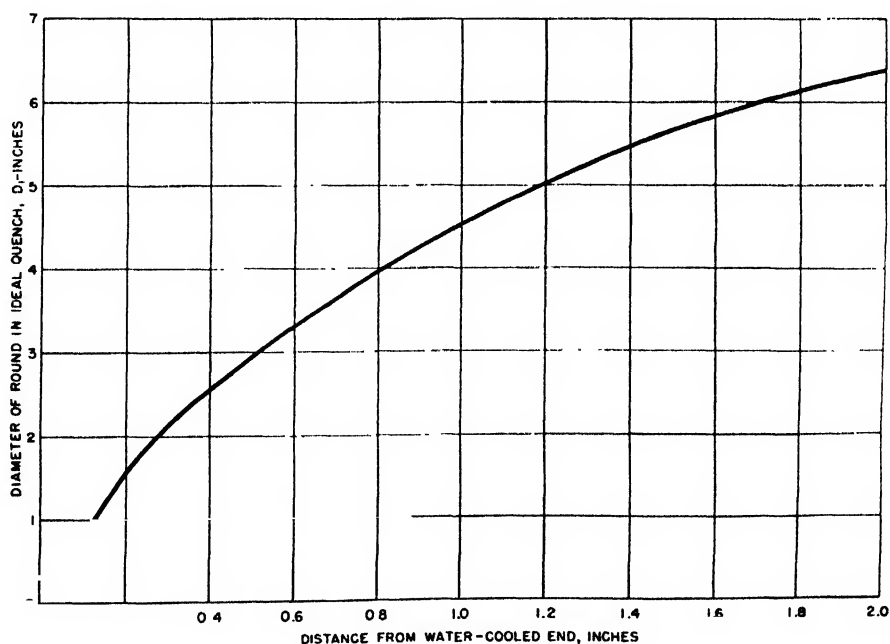


FIG. 42-35. Relationship between the distance from the quenched end of the end-quench test and ideal diameter.

SECTION 3

HEAT-TREATMENT PROCEDURES

Quenching and Tempering—The desirable properties of tempered martensitic microstructures have been emphasized in this chapter. Quenching and tempering is the heat treatment commonly used to obtain such microstructures and, therefore, represents the final heat treatment ordinarily used to obtain optimum properties in heat-treated materials.

This method is depicted diagrammatically in Figure 42—36. It involves a continuous cooling from the austen-

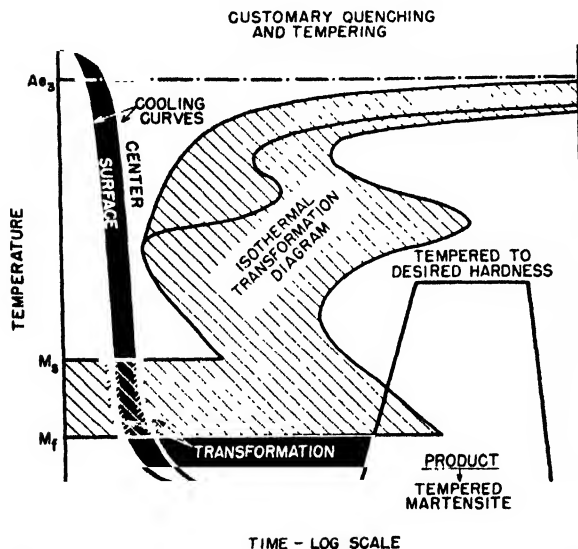


Fig. 42—36. Schematic transformation diagram for quenching and tempering.

itizing temperature through the martensite transformation temperature range at a rate rapid enough to prevent any transformation at temperatures above the M_s temperature, followed by tempering to the desired hardness or strength level.

Heating—The first step in this heat treatment, as in most of the heat treatments to be described, is the heating of the material to a temperature at which austenite is formed. The actual austenitizing temperature should, in general, be such that all carbides are in solution in order that full advantage may be taken of the hardenability effects of the alloying elements, although in some cases, particularly in tool steels, it may be desirable to leave some undissolved carbides. The temperature should not, however, be so high that pronounced grain growth occurs. The piece should be held at the austenitizing temperature long enough to dissolve carbides but, again, not long enough for excessive grain growth to occur.

Too rapid a heating rate may set up high stresses, particularly if irregular sections are involved, and is, therefore, generally undesirable. A heating time of one hour per inch of section is commonly employed, and this is a safe rule. In numerous cases, however, much more rapid heating rates may be employed. In such cases, the safety of the practice must generally be determined by experiment. The available heating rate will, of course, be determined by the mass of the material being heated and the rate at which it can absorb heat, the temperature to

which it is desired to heat, and the temperature and heat-transfer characteristics of the heating medium. In general, heating rates will be faster the higher the temperature, and the times will vary with the square of the thickness or diameter. Salt or liquid baths will have generally higher heat-transfer coefficients and, therefore, will heat more rapidly than furnaces in which the heating is in air. Since the heating rate is a function of the difference in temperature between the piece and the heating medium, rapid heating may be obtained by using a heating medium at a temperature well above the desired austenitizing temperature and removing the piece when this temperature is reached. Advantage is taken of this principle in continuous-furnace practice in which the temperature of the furnace is kept well above the desired temperature and the passage through the furnace regulated so that the piece being treated will reach the desired temperature at the outgoing end of the furnace. Temperature control is, however, uncertain in such treatment. Flame hardening, in which rapid heating is obtained by the actual impingement of a high-temperature flame on the surface of the piece being treated is also based on this principle. These rapid heating practices are the exception, however, and the usual and safe practice is a relatively slow and uniform heating to the austenitizing temperature, followed by a holding period at that temperature long enough to insure that the piece is at a uniform temperature throughout.

Unless special precautions are taken, heating will usually result in a certain amount of oxidation or scaling and may also result in decarburization. Both scaling and decarburization are usually undesirable. Scaling represents a loss of metal, mars the surface finish and may prevent rapid extraction of heat in quenching. Decarburization results in a soft surface and may seriously affect the fatigue life. The two processes do not, however, necessarily proceed together. For this reason, a slightly oxidizing atmosphere is often desirable when freedom from decarburization is important. Since the amount of scaling is largely determined by the time and temperature of the heating operation, austenitizing temperatures and times should be as low as is consistent with the principles described above in order to minimize scaling. Scaling is materially reduced by the presence of 4 per cent or more of carbon monoxide in the furnace atmosphere.

Special measures are necessary if complete freedom from scaling or decarburization is necessary. These measures include heating in a muffle containing reducing gases such as carbon monoxide or methane and hydrogen mixtures, packing in cast-iron chips or in a mixture of charcoal and sodium carbonate or heating in neutral salt or lead baths. All of these methods, however, have limitations and disadvantages and require special precautions to insure their success. The composition of the gases used in controlled-atmosphere heating varies with the temperature and must be carefully balanced so that neither carburization nor decarburization occurs. At the higher temperatures packing mixtures such as charcoal and sodium carbonate may also lead to carburization and their use is frequently very inconvenient. Salt or lead baths may become contaminated with oxides through contact with the atmosphere and these may accelerate decarburization. In general it is very difficult to insure absolute freedom from scaling or decarburization in heating for hardening.

Quenching—The primary purpose of quenching is, as described above, to cool the piece rapidly enough that no transformation occurs at temperatures above the martensite range. The first requisite of a quenching medium is, therefore, a sufficient cooling rate to accomplish this result. The necessary cooling rate is, in turn, determined by the size and hardenability of the piece being quenched, so that the choice of a quenching medium is primarily determined by these factors. The temperature gradient set up by the quenching operation results in relatively high thermal and transformation stresses which are usually, although not always, undesirable since they may lead to cracking or distortion. In order to minimize these stresses, the quenching rate should not be much in excess of that dictated by the size and hardenability of the piece.

The quenching media most commonly used are water, oils, or brine. The relative severity of quench of these media is indicated in Table 42—IV of this chapter. As indicated by this table, brine quenching is the most severe, although when thoroughly agitated, as by a submerged pressure-spray, water approaches it in severity. Oil is considerably less drastic, although its cooling rate may likewise be markedly increased by a proper and sufficient agitation.

Agitation of the quenching medium is important both because of acceleration of the cooling rate and because of the more uniform cooling obtained. Such agitation may be obtained from judiciously placed propellers, from pumps, or from pressure sprays.

The severity of water quenching varies with the temperature of the quenching bath, hot water being quite markedly slower than cold water. This is presumably because of the large amounts of steam which are formed in quenching into hot water and which cling to the work and surround it with "gas pockets." The cooling rate in hot water is, however, not only slower, but less uniform and this lack of uniformity may lead to distortion or even cracking. The increased cooling rate in brine is also presumably due to its increased boiling point which diminishes the chance of gas envelopes forming around the work. The cooling rate of oil quenches tends to increase somewhat with a moderate increase in temperature, presumably because of the decreased viscosity at the higher temperature.

Tempering—The martensite formed by quenching is very hard and very brittle and, as described above, its formation leaves high residual stresses. The purpose of tempering is to relieve these stresses and to improve the ductility, which it does at the expense of strength or hardness. The operation consists of heating at temperatures below the lower critical temperature (A_1). The stress relief and recovery of ductility are brought about through precipitation of carbide from the supersaturated unstable alpha-iron solid solution (martensite) and through diffusion and coalescence of the carbide as the tempering operation proceeds.

The effect of tempering on the residual stresses is illustrated by Figure 42—37. It will be noted that a considerable stress relief has occurred in tempering at 300° F and that tempering at 900° F has lowered the stresses to a quite low value.

A typical illustration of the effect of tempering on ductility as measured by the notch impact test is shown in Figure 42—38. It will be noted that the ductility first increases on tempering at temperatures up to 400° F, then decreases on tempering at temperatures between 400° and 600° F, and finally increases rapidly in tempering at temperatures of 800° F and above. This is a characteristic behavior and, in general, the temperature range 450° to 600° F should be avoided in tempering.

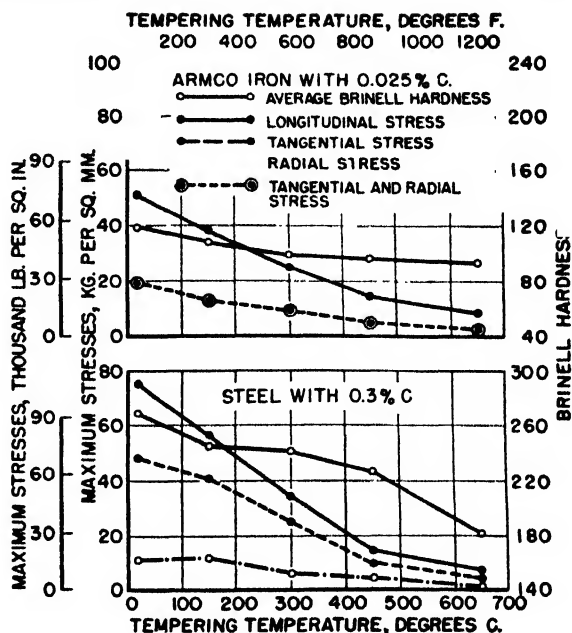


FIG. 42—37. The effect of tempering on residual stresses, in quenched cylinders. (Buhler, Buchholtz and Schulz: Archiv für das Eisenhüttenwesen, Vol. 5, 1932; pages 413-418.)

In order to minimize cracking, the tempering operation should immediately follow the quench. Allowing fully-quenched pieces to stand overnight before tempering is liable to result in a large proportion of cracked work.

The tempering of martensite results in a contraction and if the heating is not uniform, stresses will be set up by this unequal contraction which will cause distortion or even cracking. Similarly, too rapid a heating for tempering may be dangerous because of the sharp temperature gradient set up between the surface and interior of the piece. Recirculating-air furnaces are ideal

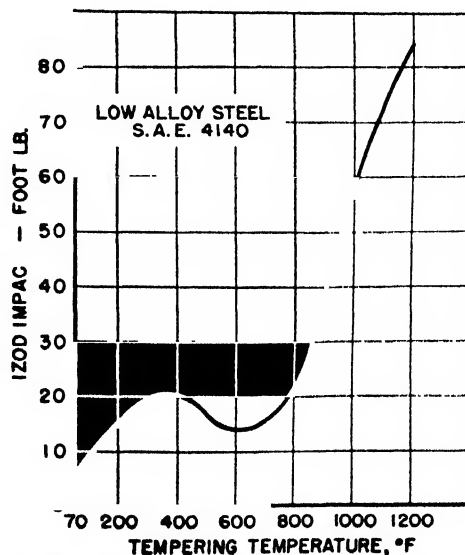


FIG. 42—38. The effect of tempering temperature on ductility (Charpy impact) properties of low-alloy steel.

for obtaining the uniform heating desired for tempering and are very commonly employed for this purpose. Oil or salt baths are very commonly used for low-temperature tempering and are generally safe, in spite of their rapid heating rate, since the temperature differential is low. Lead or salt baths may be used for higher tempering temperatures if the pieces to be tempered are not too large or irregular so that the heating stresses may be kept at a safe level.

Some steels exhibit a loss of toughness on slow cooling from temperatures of about 1000° F and above (the phenomenon known as "temper brittleness" which will be discussed further in another chapter) and therefore, a rapid cooling after tempering is generally desirable in these cases.

Martempering—As discussed above, the transformation to martensite, occurring during the rapid cooling through the martensite temperature range with the accompanying sharp temperature gradient, results in high stresses. A modified quenching procedure, known as martempering, which was developed by B. F. Shepherd, is helpful in lowering these stresses after quenching. This method is illustrated diagrammatically in Figure 42—39. In practice, it is ordinarily carried out by

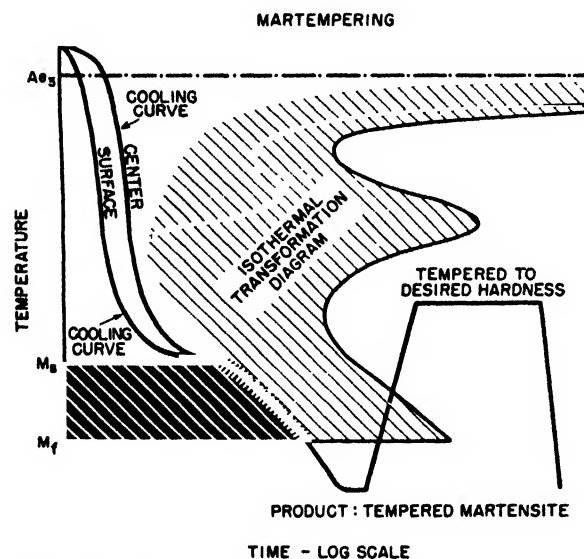


FIG. 42—39. Schematic transformation diagram for martempering.

quenching the piece into a molten-salt bath at a temperature just above the M_s temperature, holding in this bath long enough to permit the piece to acquire the temperature throughout, and then air cooling to room temperature. Transformation to martensite then occurs during the relatively slow air cooling and, since the temperature gradient characteristic of the conventional quench is absent, the stresses set up by the transformation are much lower than in conventional quenching and tempering. Along with these lower stresses goes, of course, a much greater freedom from distortion and cracking. After martempering, the piece may be tempered to the desired strength level. Martempering has been applied to the heat treatment of tools, bearings, dies, etc. in which difficulty was encountered with quench cracking or distortion when heat treated by conventional quenching and tempering.

Austempering—As discussed above, the properties of lower bainite are generally similar in respect to strength

and somewhat superior in ductility to those of tempered martensite. Austempering, which is an isothermal heat treatment to lower bainite, therefore, offers an alternative method of heat treatment for obtaining optimum strength and ductility.

The austempering treatment is illustrated diagrammatically in Figure 42—40. It involves quenching to the

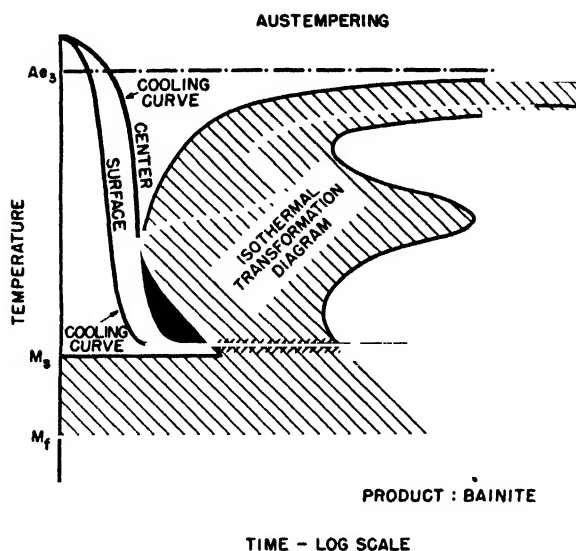


FIG. 42—40. Schematic transformation diagram for austempering.

desired temperature in the lower bainite region, usually in molten salt, and holding at this temperature until transformation is complete. It is the usual practice to hold for a time twice as long as that indicated by the isothermal transformation diagram to insure complete transformation of segregated areas. The piece may be quenched or air cooled to room temperature after transformation is complete and may be tempered to a lower hardness level if desired.

Austempering has the tremendous advantage over conventional quenching and tempering that the bainite transformation takes place isothermally at a relatively high temperature so that the transformation stresses are very low, with a resultant absolute minimum of distortion and a practically complete assurance that quench cracking will not occur.

Austempering, on the other hand, has the disadvantage, which it shares with martempering, that, because of the slower cooling rates of the molten salt baths as compared with the usual water or oil quenches, a higher hardenability steel is required to prevent high temperature transformation during the cooling to the bainite temperature. Along with these higher hardenabilities also go longer times for complete transformation to bainite so that austempering may be considerably more time consuming than martempering or conventional quenching and tempering.

This hardenability limitation may be overcome to a certain extent by the introduction of a prequench in water or oil to a temperature just below the M_s temperature, so that some martensite transformation occurs prior to the final holding at the bainite transformation. The final product is then a mixture of tempered martensite and bainite and steel with this microstructure has good properties.

Largely because of this hardenability limitation, austempering has found its widest application in the heat treatment of plain high-carbon steels in small section sizes, such as sheet, strip and wire products. It is, however, also being used for the heat treatment of alloy steels and cast irons for applications in which it is essential that distortion be held to a minimum.

Normalizing—Normalizing involves reheating the steel above its critical temperature (A_{c3}) and air cooling. It has two primary purposes: to refine the grain, and to obtain a carbide size and distribution which will be more favorable for carbide solution on subsequent heat treatment than the as-rolled structure.

The as-rolled grain size depends principally upon the finishing temperature in the rolling operation. This is subject to wide variations and there is, therefore, a corresponding wide variation in the grain size of the as-rolled products. The normalizing operation, as the name implies, serves to refine a coarse grain size resulting from a high finishing temperature and to establish a uniform, relatively fine-grained microstructure.

In alloy steels, particularly if they have been slow cooled after rolling, the carbides in the as-rolled condition tend to be rather large and massive. These large carbides are difficult to dissolve on subsequent austenitizing treatments. This carbide size, likewise, will be subject to wide variations, depending on the rolling and slow-cooling practice. Here again, normalizing tends to establish a more uniform and finer carbide particle size which will facilitate subsequent heat treatment to a more uniform final product.

The usual practice is to normalize from 100° to 150° F above the critical temperature, but for some alloy steels with carbides that are soluble only with difficulty, considerably higher temperatures may be used to obtain carbide solution. Heating, in general, should be slow enough to insure uniform temperatures and low thermal stresses. It is now a very common practice to carry out this operation in continuous furnaces. Continuous normalizing is particularly well adapted to sheet and strip because it may be heated quickly, but it is also used for plates and bars. The heating operation may, however, be carried out in any type of furnace which will permit uniform heating and accurate temperature control.

Annealing—The principal purposes of annealing are to relieve cooling stresses or stresses induced by cold or hot working, and to soften the steel so as to improve its machinability or formability. It may involve only a subcritical heating to relieve stresses, to recrystallize cold-worked material, or to spheroidize the carbides or it may involve heating above the critical temperature with subsequent transformation to pearlite or directly to a spheroidized structure on cooling.

Full Anneal—As discussed above, the most favorable microstructure for machinability in the low- or medium-carbon steels is coarse pearlite. The customary heat treatment to develop this microstructure is a full anneal, illustrated diagrammatically in Figure 42-41. It consists of austenitizing at a relatively high temperature so that full carbide solution is obtained, followed by a slow cooling so that transformation occurs only and completely in the high-temperature end of the pearlite range. This is a simple heat treatment and is reliable for most steels. It is, however, rather time consuming since it involves a slow cooling over the entire temperature range from the austenitizing temperature to a temperature well below that at which transformation is complete.

Isothermal Annealing—This annealing to coarse pearlite can, of course, be carried out isothermally by cooling to the proper temperature for transformation to coarse

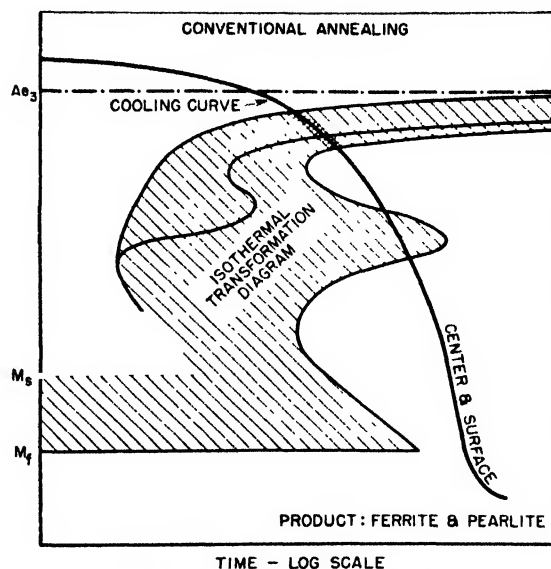


FIG. 42-41. Schematic transformation diagram for full annealing.

pearlite, and holding at this temperature until transformation is complete in a manner similar to the austempering procedure. This method is illustrated diagrammatically in Figure 42-42. Such an isothermal-annealing cycle may make possible a very considerable time saving over the conventional full-annealing treatment described above. Neither the time from the austenitizing temperature to the transformation temperature, or from the transformation temperature to room temperature is critical and these may be speeded up as much as is desired or is practical. Furthermore, if the extreme softness of the coarsest pearlite is not necessary, the transformation may be carried out at the "nose" of the curve where the transformation goes to completion most rapidly and the operation thereby further expedited; the pearlite is much finer and the hardness is higher.

Isothermal annealing is most practical for applications in which full advantage may be taken of the rapid cool-

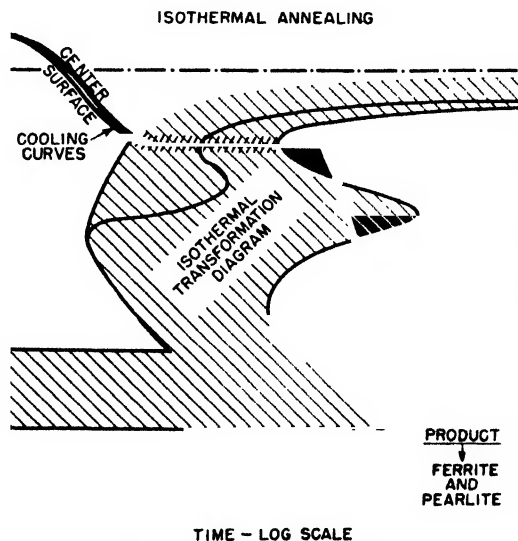


FIG. 42-42. Schematic transformation diagram for isothermal annealing.

ing to the transformation temperature and from this temperature down to room temperature. Thus for small parts which can be conveniently handled in salt or lead baths, this isothermal annealing makes possible large time savings as compared with the conventional slow furnace cooling. It is also very conveniently adapted to continuous heat treatment, and continuous annealing by this method is commonly referred to as "cycle annealing." This is usually carried out in an especially designed furnace, incorporating an air-blast chamber in order to cool rapidly from the high-heat stages for austenitizing down to the lower-temperature stages in which the transformation to pearlite occurs. This permits an accelerated cooling down to the transformation temperature. On the other hand, the method offers no particular advantage for applications such as the batch annealing of large furnace loads in which the rate of cooling to the center of the load may be so slow as to preclude any rapid cooling to the transformation temperature. For such applications, the conventional full-annealing method usually offers a better assurance of obtaining the desired microstructure and properties.

Spheroidize Annealing—Coarse pearlite microstructures are too hard for optimum machinability in the higher carbon steels, and such steels are, therefore, customarily annealed to develop spheroidized microstructures. This may be accomplished by tempering the as-rolled, slow-cooled or normalized materials at a temperature just below the lower critical temperature. Such an operation is known as "sub-critical annealing." Full spheroidization of the carbides by this method may require long holding times at the sub-critical temperature and the method may, therefore, be slow, but it is a simple heat treatment and may frequently be more convenient than annealing above the critical temperature.

It has been found, however, that the procedures described above for annealing to produce pearlite, can, with some modifications, be applied to annealing methods that will result in spheroidized microstructures. If free carbide remains after the austenitizing treatment, transformation (in the temperature range at which coarse pearlite would ordinarily form) would proceed to spheroidized rather than to pearlitic microstructures. Thus, heat treatment to form spheroidized microstructures can be carried out in a manner completely analogous to heat treatment to form pearlite, except for the use of lower austenitizing temperatures. Spheroidize annealing may thus involve a slow cooling similar to the full-annealing treatment to produce pearlite or it may be an isothermal heat treatment similar to the isothermal annealing to form pearlite. An austenitizing temperature not more than 100° F above the lower critical temperature is customarily used for this super-critical annealing to produce spheroidized microstructures.

Process Annealing—Process annealing is the term used to describe the sub-critical annealing of cold-worked materials. It customarily involves heating at a temperature high enough to cause recrystallization of the cold-worked structure and to soften the steel.

The most important example of process annealing is the box annealing of cold-rolled low-carbon sheet steel; the sheets are packed in a large box which is sealed to protect them from oxidation. This annealing is usually carried out at temperatures of from about 1100° to 1300° F. The heating and holding at temperature usually takes about 24 hours after which the charge is slowly cooled in the box, the entire process taking about 40 hours.

The process and equipment are described in detail in Chapter 34.

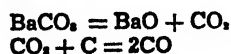
SECTION 4

CARBURIZING

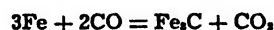
In carburizing, a high-carbon surface layer is imparted to low-carbon steel by heating it in contact with carbonaceous materials. On quenching after carburizing, the high-carbon "case" becomes very hard, while the low-carbon core remains comparatively soft. The result is a very wear-resistant exterior combined with an interior possessing great toughness, particularly suitable for gears, camshafts, etc.

Carburizing is most commonly carried out by packing the steel in boxes with carbonaceous solids, sealing to exclude the atmosphere and heating to about 1700° F for a period of time depending upon the case depth desired. This process is known as **pack carburizing**. Carburizing may also be carried out by heating the steel in direct contact with carburizing gases, in which case the process is known as **gas carburizing**; or, least commonly, in liquid baths of carburizing salts, in which case it is known as **liquid carburizing**.

Pack Carburizing—Although in pack carburizing, as described above, the parts are packed in a solid compound, the actual carburizing medium is carbon-monoxide gas. The carburizing compound usually consists of charcoal, coke and an energizer such as barium or sodium carbonate. The energizer is supposed to break down in the presence of carbon, forming carbon monoxide as follows:



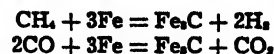
but the precise behavior of the energizer is not exactly known. The CO then carburizes the steel:



The usual carburizing temperature is 1700° F, and a case depth of about $\frac{1}{16}$ inch is ordinarily obtained on carburizing 8 hours at this temperature. Where speed is a primary consideration, temperatures up to 1750° F may be used. The higher temperatures, however, require more careful control to assure uniform results. For some steels, especially the higher-nickel alloy steels, lower temperatures of 1625° to 1650° F are used, and these lower temperatures have the advantage of decreasing warpage.

The carbon content at the surface should ordinarily not be over 1.15 per cent and the gradation toward the core should be uniform. This can be controlled to a considerable extent by the composition of the carburizing compound; for example, reducing the amount of energizer and increasing the charcoal or coke content will lower the surface carbon. In general, the maximum surface carbon will increase as the carburizing temperature decreases because of the low diffusion rate of the carbon at the lower temperature.

Gas Carburizing—The principal carburizing agents in gas carburizing are methane and carbon monoxide; the reactions may be represented as follows:



The most common practice is to lead the gas into a heated retort in which the work is continuously tumbled by rotating the retort. Such carburizing is faster than

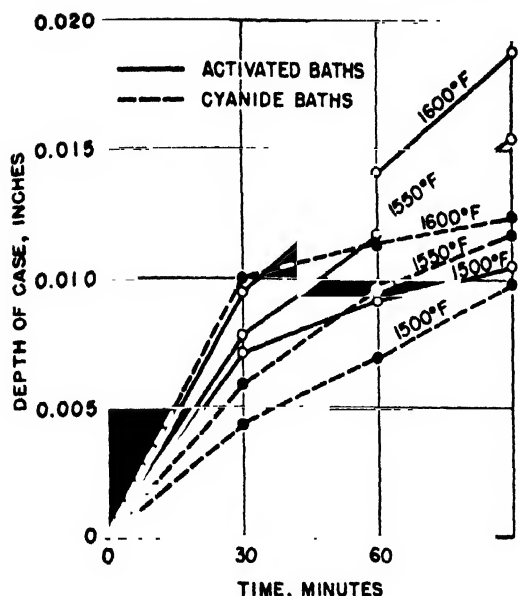


FIG. 42-43. Comparative case depths as a function of time and temperature in activated and cyanide baths. From "Metals Handbook," 1939 Edition (page 1061). Published by American Society for Metals.

pack carburizing since the time of heating the carburizer is saved, and the case depth can be held to close limits. Gas carburization has the further advantage that the carburizing cycle may be followed up by a diffusion cycle during which no carburizing gas is admitted, and thereby a lower surface carbon and a better gradation of the case obtained.

Natural gas, which consists largely of methane, with ethane varying from 5 to 10 per cent, is very satisfactory for gas carburizing. The carburizing gas may, however, be produced in a separate gas generator, or bottled propane may be used.

Liquid Carburizing—Immersion of steel parts in a molten-salt bath containing about 30 per cent sodium cyanide at 1600° F for ½ to 1 hour periods to obtain a light (0.010 inch) hard case for wear resistance has been practiced for many years. The case is a mixture of carbides and nitrides and its relatively high hardness reflects to a considerable extent the presence of iron nitride.

The above cyaniding process is now, however, largely being replaced by liquid carburizing in activated baths which employ a floating slag of calcium cyanide as the active agent and which produce deeper cases which are lower in nitrogen and higher in carbon than those obtained with the simple sodium-cyanide bath. A typical composition of an activated bath is as follows:

Calcium Cyanamide	CaCN ₂	2-5%
Calcium Cyanide	Ca(CN) ₂	43-48%
Sodium Chloride	NaCl	30-35%
Calcium Oxide	CaO	14-16%
Carbon	C	4-5%

The comparative case depths as a function of time and temperature in activated and cyanide baths are shown

in Figure 42-43. The cases obtained with activated baths range from 0.70 to 1.00 per cent carbon with about 0.2 per cent nitrogen concentrated mainly at the surface of the steel parts.

Heat Treatment of Carburized Parts—Since carburized articles have a high-carbon case and a low-carbon core, the proper heat-treating temperature for the case will be too low for the core, and vice versa. Thus a double treatment is desirable to obtain optimum properties of both case and core; the piece is first heated to above the critical temperature corresponding to the low-carbon core and suitably cooled to refine its structure. It is then reheated to just above the critical temperature of the case and quenched to harden the case. However, when the carburizing temperature is not too high, refining of the core may not be essential, and after cooling from the carburizing temperature, a single reheating and quench from above the critical temperature of the case is sufficient. Quenching directly from the carburizing temperature is sometimes employed, although this may lead to undue warpage.

Nitriding—The nitrogen case-hardening process which is termed "nitriding" consists in subjecting machined and preferably heat-treated parts to the action of a nitrogenous medium, commonly ammonia gas, under certain conditions whereby surface hardness is imparted to the material without necessitating any further treatment. Wear resistance, retention of hardness at elevated temperatures, and resistance to certain types of corrosion are other properties imparted to the steel by nitriding.

It has been found that chromium and aluminum are desirable in steels for nitriding and compositions especially adapted to nitriding have been developed. A typical composition is as follows:

Carbon	0.20-0.30%
Manganese	0.40-0.60%
Aluminum	0.90-1.40%
Chromium	0.90-1.40%
Molybdenum	0.15-0.25%

Usual conditions for the nitriding process consist of subjecting the articles to the action of ammonia gas at temperatures ranging from 930° to 1220° F. The range most commonly used is 950° to 1000° F. Nitrided cases are ordinarily light, case depth of 0.010 to 0.015 inch being obtained in 48 hours at 975° F. The surface hardness is, however, very high (900 to 1200 Vickers), and this hardness is retained even after reheating to temperatures up to 900° F.

Chapter 43

CARBON STEELS

SECTION 1

CLASSIFICATION AND APPLICATION

The plain carbon steels undoubtedly represent the most important group of engineering materials known. They represent by far the major percentage of steel production and the widest diversity of application of any of the engineering materials. These applications are so diversified that anything like a complete listing, or even a classification on the basis of application, is impossible. Many of the important classes of application have, however, been discussed in this book. These include castings, forgings, tubular products, plates, sheet and strip, wire and wire products, structural shapes, bars, tools, and such railway materials as rails, wheels and axles.

Although a classification by application is impossible, plain carbon steels may be generally classified in accordance with their method of manufacture as basic open hearth, acid open hearth or acid Bessemer steels. The basic open hearth steels, of course, represent the preponderance of the tonnage. This classification may be

extended to include the method of deoxidation used. For example, the basic open hearth steels may be rimmed, semi-killed or fully killed.

The plain carbon steels may also be classified on the basis of carbon content as hypo-eutectoid or hyper-eutectoid steels; the hypo-eutectoid steels are those in which the carbon content is below the eutectoid value of about 0.80 per cent, and the hyper-eutectoid steels those with carbon contents above this value.

The composition ranges for the plain carbon steels have been standardized by the American Iron and Steel Institute and these composition ranges are listed in Tables 43—I, 43—II, 43—III and 43—IV. Concerning these four tables, it should be noted that the prefix letters "B" and "C" are used to designate the two principle steel-making processes for carbon steels as follows: "B" denotes acid Bessemer carbon steel, "C" denotes basic open hearth steel.

SECTION 2

FACTORS AFFECTING CARBON-STEEL PROPERTIES

The principal factors affecting the properties of the plain carbon steels are the carbon content and the microstructure. The general relationships between microstructure and properties, and the factors governing microstructure, have been discussed in the preceding chapter on heat treatment, and need not be repeated here. Most of the plain carbon steels are, however, used without a final heat treatment and the factors affecting the microstructure and thereby the properties in such as-rolled or as-forged products will be emphasized in this chapter.

In addition to the predominant effects of carbon content and microstructure, the properties of plain carbon steels may be modified by the effects of residual elements other than the carbon, manganese, silicon, phosphorus and sulphur which are always present, or the properties of carbon steel may also be affected by the presence of gases, especially oxygen, nitrogen and hydrogen and their reaction products. These incidental elements are usually picked up from the scrap, from the deoxidizers, or from the furnace atmosphere. The gas content is largely dependent upon the melting, deoxidizing and pouring practice so that the final properties of the plain carbon steels are, to a very considerable extent, dependent upon the steelmaking practice used in their production.

Thus, the factors governing the properties of a plain carbon steel are primarily its carbon content and microstructure, with the microstructure being determined largely by the composition and the final rolling, forging

or heat-treating operation, and secondarily by the residual alloy, non-metallic and gas content of the steel which, in turn, depend upon the steelmaking practice.

Carbon Content and Properties—The average mechanical properties of as-rolled one-inch bars of carbon steels, as a function of carbon content, are shown in Figure 43—1. These values are based on statistical analyses made by several investigators and plotted by Sisco. This figure is illustrative of the general effect of carbon content when the microstructure and grain size are held reasonably constant. It will be seen that the hardness, tensile strength and yield strength increase with increasing carbon content, while the elongation, reduction of area, and Charpy impact values decrease sharply.

Effect of Microstructure and Grain Size—The general relationships between microstructure and properties have been discussed in Chapter 42 on heat treatment. The carbon steels, being of relatively low hardenability, are predominantly pearlitic in the cast, rolled or forged conditions. The constituents of the hypo-eutectoid steels are, therefore, ferrite and pearlite, and of the hyper-eutectoid steels, cementite and pearlite. As described in the previous chapter, the properties of such pearlitic steels are dependent primarily upon the interlamellar spacing of the pearlite and the grain size. Both the hardness and the ductility increase as the interlamellar spacing or the pearlite-transformation temperature decreases, and the ductility increases with decreasing grain size. The effect of grain size was discussed in some detail in the preceding chapter. The effect of the

Table 43—L. Standard A.I.S.I. Composition Ranges of Carbon Steels⁽¹⁾
(Basic Open Hearth and Acid Bessemer Carbon Steels)⁽²⁾

A.I.S.I. Number	Chemical Composition Limits, Per Cent				Corresp. SAE No.
	C	Mn	P (Max.)	S (Max.)	
C1006	0.08 Max.	0.25-0.40	0.040	0.050	1006
C1008	0.10 Max.	0.25-0.50	0.040	0.050	1008
C1010	0.08-0.13	0.30-0.60	0.040	0.050	1010
C1011	0.08-0.13	0.60-0.90	0.040	0.050	—
C1012	0.10-0.15	0.30-0.60	0.040	0.050	—
C1015	0.13-0.18	0.30-0.60	0.040	0.050	1015
C1016	0.13-0.18	0.60-0.90	0.040	0.050	1016
C1017	0.15-0.20	0.30-0.60	0.040	0.050	1017
C1018	0.15-0.20	0.60-0.90	0.040	0.050	1018
C1019	0.15-0.20	0.70-1.00	0.040	0.050	1019
C1020	0.18-0.23	0.30-0.60	0.040	0.050	1020
C1021	0.18-0.23	0.60-0.90	0.040	0.050	1021
C1022	0.18-0.23	0.70-1.00	0.040	0.050	1022
C1023	0.20-0.25	0.30-0.60	0.040	0.050	—
C1024	0.19-0.25	1.35-1.65	0.040	0.050	1024
C1025	0.22-0.28	0.30-0.60	0.040	0.050	1025
C1026	0.22-0.28	0.60-0.90	0.040	0.050	1026
C1027	0.22-0.29	1.20-1.50	0.040	0.050	1027
C1029	0.25-0.31	0.60-0.90	0.040	0.050	—
C1030	0.28-0.34	0.60-0.90	0.040	0.050	1030
C1031	0.28-0.34	0.30-0.60	0.040	0.050	—
C1032	0.30-0.36	0.60-0.90	0.040	0.050	—
C1033	0.30-0.36	0.70-1.00	0.040	0.050	1033
C1035	0.32-0.38	0.60-0.90	0.040	0.050	1035
C1036	0.30-0.37	1.20-1.50	0.040	0.050	1036
C1037	0.32-0.38	0.70-1.00	0.040	0.050	—
C1038	0.35-0.42	0.60-0.90	0.040	0.050	1038
C1039	0.37-0.44	0.70-1.00	0.040	0.050	1039
C1040	0.37-0.44	0.60-0.90	0.040	0.050	1040
C1041	0.36-0.44	1.35-1.65	0.040	0.050	1041
C1042	0.40-0.47	0.60-0.90	0.040	0.050	1042
C1043	0.40-0.47	0.70-1.00	0.040	0.050	1043
C1045	0.43-0.50	0.60-0.90	0.040	0.050	1045
C1046	0.43-0.50	0.70-1.00	0.040	0.050	1046
C1049	0.46-0.53	0.60-0.90	0.040	0.050	1049
C1050	0.48-0.55	0.60-0.90	0.040	0.050	1050
C1052	0.47-0.55	1.20-1.50	0.040	0.050	1052
C1053	0.48-0.55	0.70-1.00	0.040	0.050	—
C1055	0.50-0.60	0.60-0.90	0.040	0.050	1055
C1060	0.55-0.65	0.60-0.90	0.040	0.050	1060
C1065	0.60-0.70	0.60-0.90	0.040	0.050	1065
C1069	0.65-0.75	0.40-0.70	0.040	0.050	—
C1070	0.65-0.75	0.60-0.90	0.040	0.050	1070
C1072	0.65-0.76	1.00-1.30	0.040	0.050	—
C1075	0.70-0.80	0.40-0.70	0.040	0.050	—
C1078	0.72-0.85	0.30-0.60	0.040	0.050	1078
C1080	0.75-0.88	0.60-0.90	0.040	0.050	1080
C1084	0.80-0.93	0.60-0.90	0.040	0.050	—
C1085	0.80-0.93	0.70-1.00	0.040	0.050	1085
C1086	0.82-0.95	0.30-0.50	0.040	0.050	1086
C1090	0.85-0.98	0.60-0.90	0.040	0.050	1090
C1095	0.90-1.03	0.30-0.50	0.040	0.050	1095
B1010	0.13 Max.	0.30-0.60	0.07-0.12	0.060	—

SILICON.—When silicon is required, the following ranges and limits are common for basic open-hearth steel grades:

Standard Steel Designations

Up to C 1015 Excl.

C 1015 to C 1025 Incl.

Over C 1025

Silicon Ranges or Limits

0.10 Max.

0.10 Max., 0.10/0.20, or 0.15/0.30

0.10/0.20, or 0.15/0.30

COPPER.—When required, copper is specified as an added element to a standard steel.

^{a)} Hot-rolled carbon-steel bars and semifinished products not exceeding 200 square inches cross-sectional area.

⁶⁰ From: Steel Products Manual, American Iron and Steel Institute; Section 2, Semifinished Carbon-Steel Products, and Section 3, Hot-Rolled Carbon-Steel Bars.

Table 43—II. Standard A.I.S.I. Composition Ranges of Basic Open Hearth Resulphurized Carbon Steels ^ω

A.I.S.I. No.	Chemical Composition Limits, Per Cent				Corre- sponding SAE No.
	C	Mn	P (Max.)	S (Max.)	
C1108	0.08/0.13	0.50/0.80	0.040	0.08/0.13
C1109	0.08/0.13	0.60/0.90	0.040	0.08/0.13	1109
C1110	0.08/0.13	0.30/0.60	0.040	0.08/0.13
C1113	0.10/0.16	1.00/1.30	0.040	0.24/0.33
C1115	0.13/0.18	0.60/0.90	0.040	0.08/0.13	1115
C1116	0.14/0.20	1.10/1.40	0.040	0.16/0.23	1116
C1117	0.14/0.20	1.00/1.30	0.040	0.08/0.13	1117
C1118	0.14/0.20	1.30/1.60	0.040	0.08/0.13	1118
C1119	0.14/0.20	1.00/1.30	0.040	0.24/0.33	1119
C1120	0.18/0.23	0.70/1.00	0.040	0.08/0.13	1120
C1125	0.22/0.28	0.60/0.90	0.040	0.08/0.13
C1126	0.23/0.29	0.70/1.00	0.040	0.08/0.13	1126
C1132	0.27/0.34	1.35/1.65	0.040	0.08/0.13	1132
C1137	0.32/0.39	1.35/1.65	0.040	0.08/0.13	1137
C1138	0.34/0.40	0.70/1.00	0.040	0.08/0.13	1138
C1140	0.37/0.44	0.70/1.00	0.040	0.08/0.13	1140
C1141	0.37/0.45	1.35/1.65	0.040	0.08/0.13	1141
C1144	0.40/0.48	1.35/1.65	0.040	0.24/0.33	1144
C1145	0.42/0.49	0.70/1.00	0.040	0.04/0.07	1145
C1146	0.42/0.49	0.70/1.00	0.040	0.08/0.13	1146
C1148	0.45/0.52	0.70/1.00	0.040	0.04/0.07
C1151	0.48/0.55	0.70/1.00	0.040	0.08/0.13	1151

SILICON.—When silicon is required, the following ranges and limits are common for basic open-hearth steel grades:

Standard Steel Designations

Up to C 1113 Excl.
C 1113 and Over

Silicon Ranges or Limits

0.10 Max.
0.10 Max., 0.10/0.20, or 0.15/0.30

^ω See Footnote (1), Table 43—I.

Table 43—III. Standard A.I.S.I. Composition Ranges of Basic Open Hearth Rephosphorized and Resulphurized Carbon Steels ^ω.

A.I.S.I. Number	Chemical Composition Limits, Per Cent				Corresp. SAE No.
	C	Mn	P	S	
C1211	0.13 Max.	0.60/0.90	0.07/0.12	0.08/0.15	—
C1212	0.13 Max.	0.70/1.00	0.07/0.12	0.16/0.23	—
C1213	0.13 Max.	0.70/1.00	0.07/0.12	0.24/0.33	—

SILICON.—Because of the technological nature of the process basic open hearth rephosphorized and resulphurized steels are not furnished to specified limits for silicon.

^ω See Footnote (1), Table 43—I.

Table 43—IV. Standard A.I.S.I. Composition Ranges of Acid Bessemer Resulphurized Carbon Steels ^ω

A.I.S.I. No.	Chemical Composition Limits, Per Cent				Corre- sponding SAE No.
	C	Mn	P	S	
B1111	0.13 Max.	0.60/0.90	0.07/0.12	0.08/0.15	1111
B1112	0.13 Max.	0.70/1.00	0.07/0.12	0.16/0.23	1112
B1113	0.13 Max.	0.70/1.00	0.07/0.12	0.24/0.33	1113

SILICON.—Because of the technological nature of the process, acid Bessemer steels are not furnished with specified silicon content.

^ω See Footnote (1), Table 43—I.

Table 43—V. Standard Mechanical Property Specifications ^(a)

Tensile Ranges (Lb. per Sq. In.)	American Society for Testing Materials Specifications		
	Title	Designation	Grade
45,000/55,000	Structural—General	A-306	45
45,000/55,000	Boiler Rivet Steel	A-31	A
48,000/58,000	Structural for Locomotives & Cars	A-113	C
50,000/60,000	Structural—General	A-306	50
50,000/62,000	Structural—Locomotives & Cars	A-113	B
52,000/62,000	Structural Rivet Steel	A-141	—
55,000/65,000	Structural—General	A-306	55
55,000/65,000	Structural for Ships	A-131	Rivet
58,000/68,000	Boiler Rivet Steel	A-31	B
58,000/71,000	Structural for Ships	A-131	Struct.
60,000/72,000	Structural—General	A-306	60
60,000/72,000 *	Structural—Bridges and Buildings	A-7	—
60,000/72,000 *	Structural—Locomotives and Cars	A-113	A
65,000/77,000	Structural—General	A-306	65
68,000/82,000	High Strength Rivet Steel	A-195	—
70,000/85,000	Structural—General	A-306	70
75,000/90,000	Structural—General	A-306	75
80,000 min.	Structural—General	A-306	80

* 60,000/75,000 psi permitted for thickness or diameter over 1½ in.

(a) From: Steel Products Manual, Section 8, American Iron and Steel Institute.

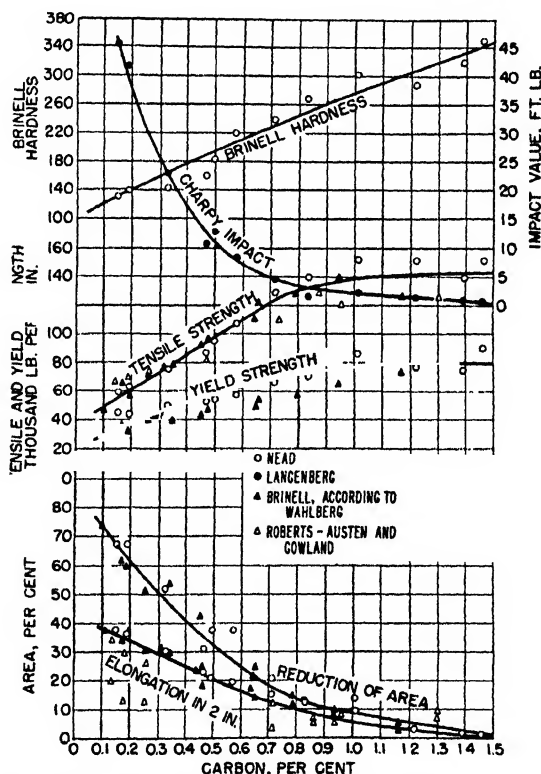


FIG. 43—1. Variations in average mechanical properties of as-rolled, one-inch diameter bars of plain carbon steels, as a function of carbon content. (After Sisco.)

interlamellar spacing of the pearlite was also discussed and illustrated for a eutectoid steel. The effect of this variable on tensile strength is further illustrated in Figure 43—2, which shows the approximate relationship between tensile strength and carbon content for a series of plain carbon steels isothermally transformed to fine

and coarse pearlitic microstructures. The line for the 1200° F transformation product in this illustration is generally similar to, although slightly above, the tensile strength line for as-rolled bars in Figure 43—1, indicating that these as-rolled bars have transformed during cooling at temperatures in the vicinity of 1200° F.

THE MAKING, SHAPING AND TREATING OF STEEL

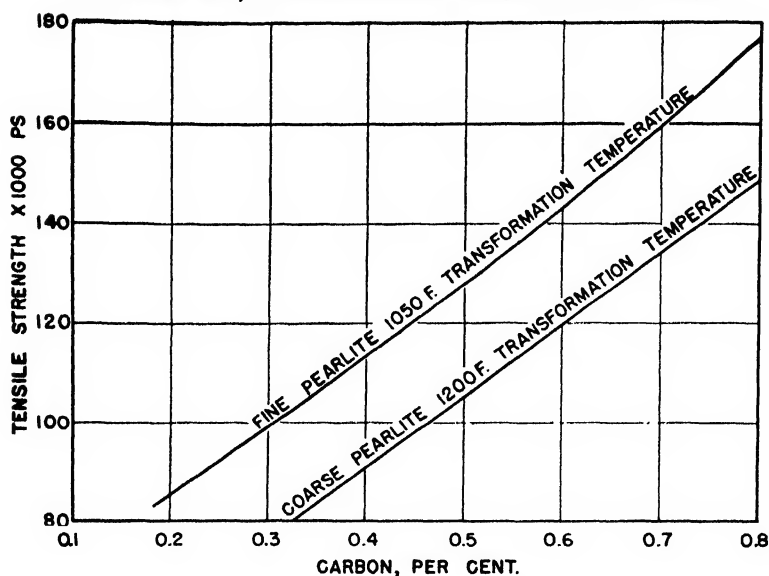


FIG. 43-2. Relationship between tensile strength and carbon content of a series of plain carbon steels isothermally transformed to fine and coarse pearlitic microstructures.

SECTION 3

FACTORS AFFECTING MICROSTRUCTURE AND GRAIN SIZE

Composition—As explained in Chapter 42 on heat treatment, the microstructure of steel is determined by the temperature range in which transformation of the austenite takes place on cooling. This, in turn, is determined by the cooling rate employed and the transformation rate of the steel. This latter factor is dependent largely upon the composition; thus, for a given cooling rate after rolling, for instance, the resulting microstructure is largely dependent upon the composition. The composition will, of course, similarly control the microstructure for given cooling conditions in cast, as-rolled, or heat-treated carbon steels.

The austenite-transformation behavior in carbon steel is determined almost entirely by the carbon and manganese content; the effects of phosphorus and sulphur are almost negligible, and the silicon contents are normally so low that they are likewise ineffective. The carbon content is ordinarily chosen in accordance with the strength level desired and the manganese content then selected in order to produce suitable microstructure and properties at this carbon level under the given cooling conditions.

Microstructure of Cast Steels—The microstructure of as-cast steels is, of course, determined by the composition and cooling conditions in the same manner as in wrought steels. Cast steels are usually very coarse grained since the austenite forms at a high temperature, and the pearlite is usually coarse since the cooling through the critical range, particularly if the casting is cooled in the mold, is usually quite slow. In hypoeutectoid steels, ferrite is precipitated ordinarily at the original austenite boundaries during the cooling. In hyper-eutectoid steels, cementite is similarly precipitated. Such mixtures of ferrite or cementite and coarse-grained coarse pearlite have, as would be expected, poor properties both in respect to strength and ductility, and heat treatment is usually necessary to obtain suitable microstructures and properties in cast steels.

The dendritic segregation occurring during the solidification of steel castings also results in an irregular microstructure and correspondingly poor properties, and the homogenization of this segregated structure is another function of the heat treatment of cast steels.

A typical microstructure of an as-cast carbon steel is shown in Figure 43-3.

Effects of Hot Working—Most carbon steels are used in the form of as-rolled finished sections and the microstructure and properties of these sections are determined largely by the composition, rolling practice and cooling

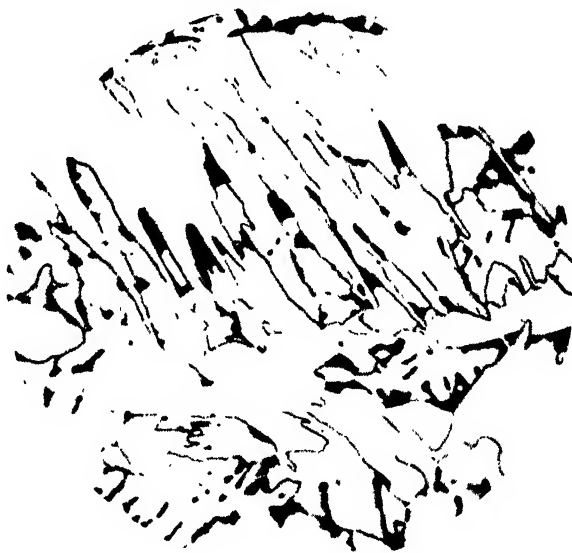


FIG. 43-3. As-cast microstructure of 0.20 per cent carbon steel. Nital etch; magnification: 200X.

conditions after rolling. The rolling or hot working of these sections is ordinarily carried out in the temperature range at which the steel is austenitic and has four major effects, as follows:

1. Considerable homogenization that tends to eliminate dendritic segregation occurs during the heating for rolling.
2. The dendritic structure is broken up during rolling.
3. Recrystallization occurs during rolling so that the final austenitic grain size is determined by the temperature at which the last passes are made (the finishing temperature).
4. Dendrites and inclusions are reoriented in the rolling direction so that the final ductility in the rolling direction is markedly improved.

Thus, the homogeneity and grain size of the austenite is largely determined by the rolling practice. It should be pointed out, however, that, as discussed in earlier chapters, the recrystallization characteristics of the austenite and, therefore, the austenite-grain size characteristic of a given finishing temperature may be markedly affected by the steelmaking practice, particularly in respect to the deoxidation practice used.

The distribution of the ferrite or cementite and the nature of the pearlite is, however, as has been explained earlier, determined by the cooling rate after rolling. Since the usual practice is air cooling, the final microstructure and, therefore, the properties of these as-rolled sections will be principally dependent on the composition and section size.

Bars, rails or structural shapes are rolled in one direction only and this rolling tends to orient the grains along the direction of rolling. Inclusions which are plastic at the rolling temperature are also elongated. This elongation tends to improve the mechanical properties of the rolled steel as compared to the as-cast steel, when the material is subjected to stresses acting longitudinally, as in the case of tensile specimens cut in the direction of rolling. However, specimens cut transversely to the direction of rolling have lower values for elongation, reduction of area, and impact resistance than do the longitudinal specimens. This effect is minimized by cross

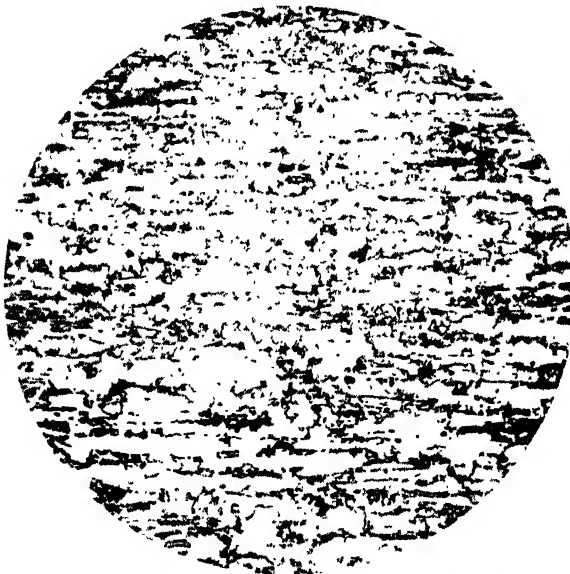


FIG. 43-4. Microstructure of full-hard cold-reduced black plate (85 per cent reduction). Nitral etch; magnification: 200X.

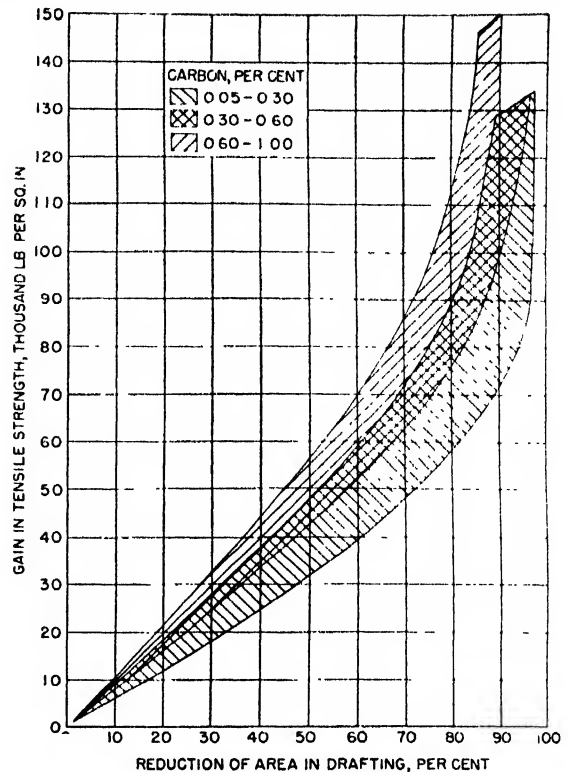


FIG. 43-5. Increase of tensile strength of plain carbon steel with increasing amounts of cold working.

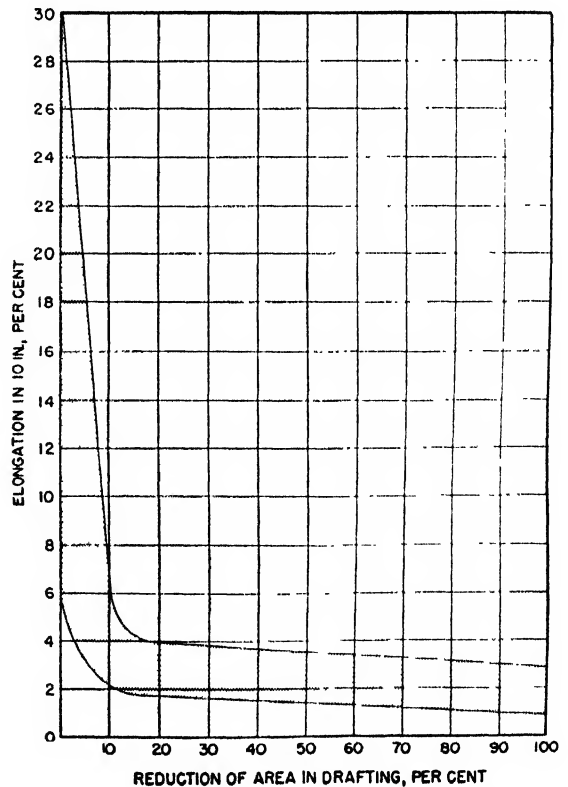


FIG. 43-6. Effect of cold working on the ductility of plain carbon steel.

rolling, i.e., by hot working in both directions, and this is frequently practiced in sheet and plate products.

Effects of Cold Working—The manufacture of wire, sheet and strip, and tubular products often involves a cold-working operation and the general effects of cold working will, therefore, be discussed in this chapter. The effects of this cold working may often be destroyed by a suitable annealing operation but some products, particularly wire, are used in the cold-worked condition.

A typical microstructure of a heavily cold-worked steel is shown in Figure 43-4. The elongation of the ferrite and pearlite grains and the generally distorted

microstructure are characteristic. The most pronounced effect of this cold work is an increase in strength and hardness and a decrease in ductility as represented by elongation and reduction of area. The effects of cold working on tensile strength and elongation are shown in Figures 43-5 and 43-6. Upon reheating cold-worked steel to the recrystallization temperature (750° F) or above, depending upon composition, amount of cold work and other variables, the original microstructure and properties may be restored. The annealing of cold-worked steels (process annealing) has been discussed in the chapter on heat treatment.

SECTION 4

HEAT TREATMENT OF CARBON STEELS

Although the majority of carbon steels are used without a final heat treatment, heat treatment may be employed to improve the microstructure and properties for specific applications. The principles of these heat treatments have been discussed in Chapter 42 and many of the heat-treating practices have been described in detail in the chapters of the various carbon-steel products. As mentioned earlier, the heat treatment of cast carbon steels improves the properties of the material by breaking up the dendritic structure and refining the grain size and microstructure. These treatments usually involve normalizing the castings at a high temperature to homogenize the dendritic structure, followed by annealing at a lower temperature for grain refinement. The use of a final quenching and tempering treatment to obtain optimum microstructure and properties in castings is, however, becoming an increasingly common practice. The heat treatment employed in processing wrought steel products are described in the following paragraphs.

Annealing—Annealing is practiced for applications requiring better machinability or formability than would be obtained with the as-rolled microstructure. This is usually a full anneal to form coarse pearlite, although a sub-critical anneal or spheroidizing treatment is occasionally practiced. Process annealing to obtain optimum formability in cold-rolled strip, sheet and tubing is, of course, a universal practice.

Normalizing—The grain size of as-rolled products is, as described above, largely dependent upon the finishing temperature in rolling and this is difficult to control.

Therefore, a final normalizing treatment from a relatively low temperature may be used to establish a fine uniform grain size for critical applications in respect to ductility or toughness.

Quenching and Tempering—The quenching and tempering of plain carbon steels to obtain optimum microstructures and properties is being increasingly practiced. Because of the relatively low hardenability of these steels, this type of treatment falls generally into two classifications, as follows:

1. Heat treatment to produce essentially tempered martensite for optimum properties. Hardenability restrictions limit the application of this type of treatment to section sizes of not more than $\frac{3}{8}$ to $\frac{1}{2}$ inch. It is, however, commonly practiced for small tools, sheet and strip, etc., which fall within this size limitation.

2. Heat treatment to form fine pearlite. Quite large sections of plain carbon steels may be quenched and tempered to produce fine pearlite microstructures, thereby making available the greatly improved strength and ductility associated with this microstructure as compared with the properties of the coarse pearlite of the usual as-rolled or normalized products.

Austempering—Thin sections (0.2-inch and below) of carbon steels are particularly suitable for austempering, since the times for transformation to bainite are relatively short and this heat treatment is likewise being increasingly practiced for applications requiring toughness at high hardness in such section sizes.

SECTION 5

AGING IN CARBON STEELS

Aging in steel is manifested as a spontaneous increase in hardness at room temperature, the process being accelerated by raising the temperature slightly. It is generally assumed to be caused by the disintegration of a supersaturated solid solution. In a system in which the solid solubility of the solute decreases sharply with temperature, the solute may be retained in supersaturated solid solution on rapid cooling, but it will tend to precipitate out on standing. Such incipient or complete precipitation is considered to be the cause of aging in steels. This precipitation is accelerated by straining, and straining frequently plays an important part in the aging of steel. To separate the effects of straining from the effects of precipitation in the absence of strain, aging in the absence of strain is referred to as quench aging, and aging after or during straining is known as strain aging.

The elements in carbon steel which seem most likely

to cause aging are carbon, oxygen and nitrogen. Curves for the change of solubility of these three elements in ferrite as a function of temperature, are shown in Figure 43-7. These curves are all of the type that can lead to aging. Probably all three of these elements play a part in the aging of steel, but it is very difficult to isolate their individual effects.

The general nature of the hardness increase from quench aging is shown in Figure 43-8. It will be noted that aging above room temperature results in a more rapid hardness increase, but that the maximum hardness attained is lower than in steel aged at room temperature. Aging for times beyond that corresponding to the maximum hardness results in a decrease in hardness. This is sometimes called overaging.

It has been found that steels which are drastically deoxidized with aluminum or aluminum and titanium are

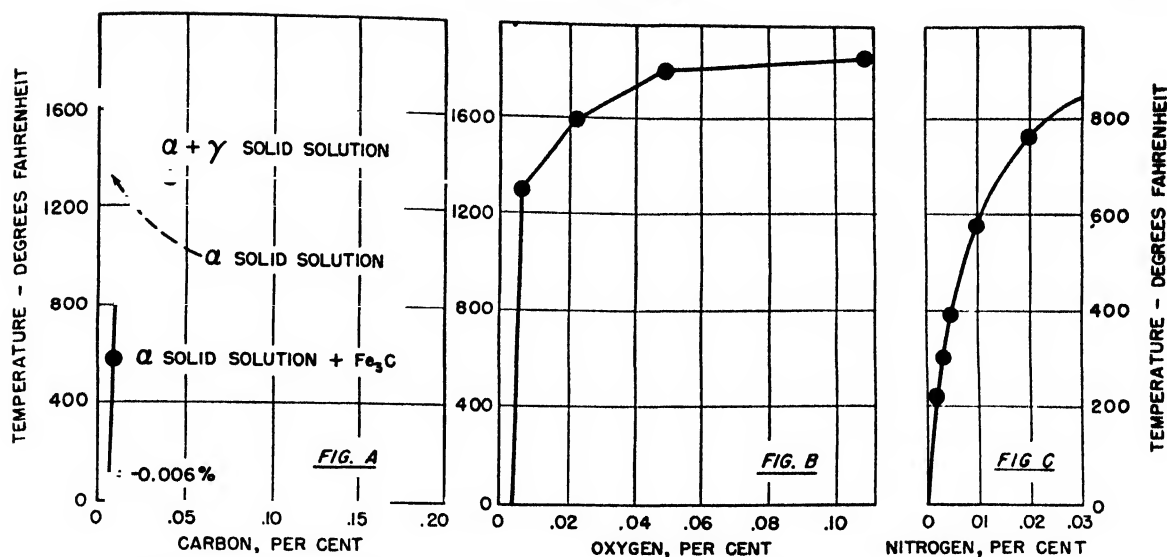


FIG. 43-7. Effect of temperature upon the solubilities of carbon, oxygen, and nitrogen in ferrite. (From "Metals Handbook," 1948 Edition; American Society for Metals.)

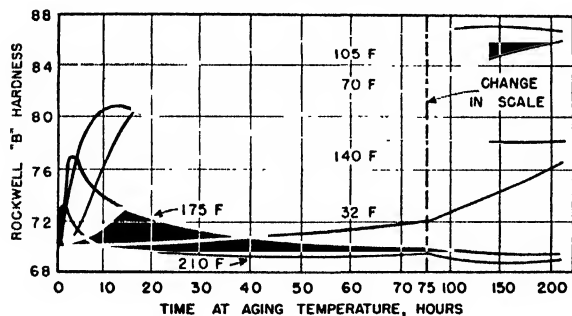


FIG. 43-8. Changes in hardness of 0.06 per cent carbon steel quenched from 1325° F after aging at indicated temperatures. (From "Metals Handbook," 1948 Edition; American Society for Metals.)

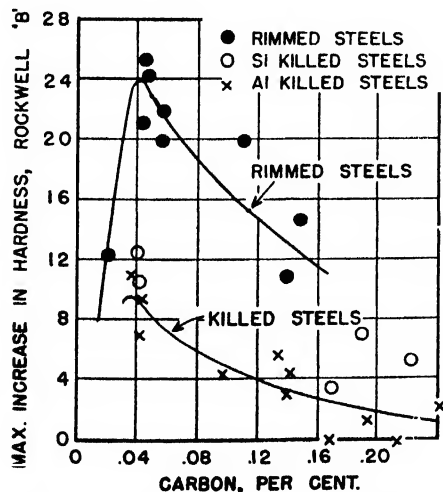


FIG. 43-9. Effect of deoxidation practice on quench-aging characteristics of carbon steels. (From "Metals Handbook," 1948 Edition; American Society for Metals.)

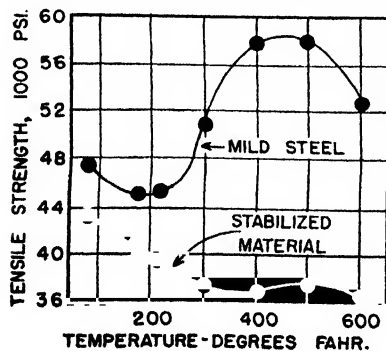
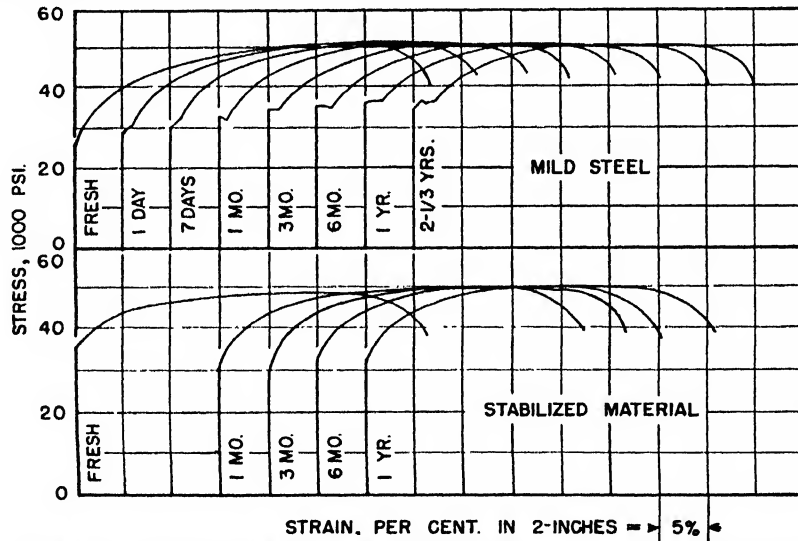
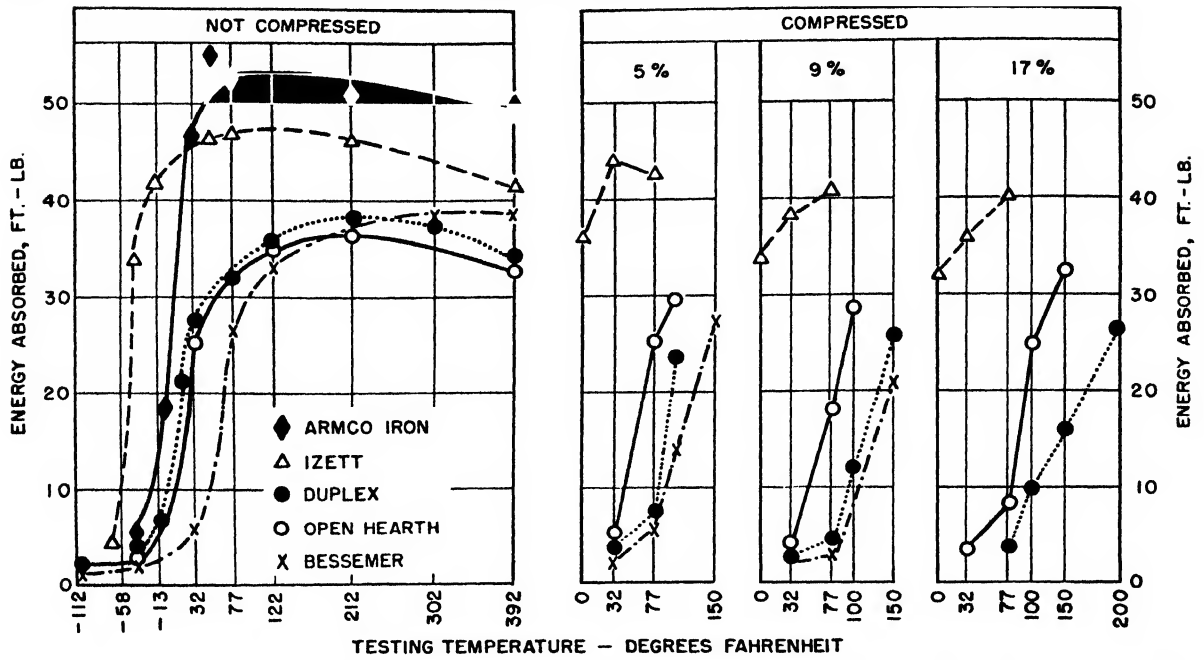
essentially non-aging and that the rimmed and Bessemer steels are the most susceptible to aging. This effect is illustrated by Figure 43-9, which shows the increase in hardness resulting from quench aging in steels with three different deoxidation practices as a function of carbon content. It is perhaps significant that the maximum aging effect was found at 0.04 per cent carbon, which is approximately the maximum solid solubility of carbon in ferrite. In general, the changes in hardness on strain aging are somewhat less than those from quench aging, but there is some indication that strain aging may be more embrittling than quench aging.

The effects of strain aging on the impact properties of steels with different manufacturing practices are shown in Figure 43-10. The Izett steel is a strongly deoxidized "non-aging" steel. Since these tests were made immediately after straining, the embrittlement represents a combination of the effect of cold working and of precipitation occurring during the straining. Further embrittlement would result from aging, particularly accelerated aging, and a maximum embrittlement has been found in heating about ½ hour at 500° F.

The effect of aging is also reflected in the characteristics of the stress-strain curve. Most as-rolled carbon steels show a jog in the stress-strain curve or a drop of the beam at the yield point in tensile testing. This jog or drop of the beam disappears if the specimen has been subjected to a previous strain, but again reappears upon aging after this straining. This phenomenon does not occur, however, in a strongly deoxidized "non-aging" steel. This effect on the stress-strain curves is illustrated in Figure 43-11.

Still another manifestation of the strain aging phenomenon is the increased hardness and decreased ductility of steels aged at 400° to 500° F, as shown by tensile tests at this temperature. This phenomenon is illustrated in Figure 43-12 and the distinction between the stabilized and aging type of steel is apparent.

These aging phenomena are of importance for applications of carbon steels in which formability or toughness are of importance, and possible embrittlement from strain aging must be given careful consideration whenever a material is subjected to strain during fabrication or use. The yield point phenomenon discussed above is of importance in the stamping or drawing of sheet, since



the jog in the stress-strain curve will be reflected in a roughening of the surface, known as stretcher strains. Galvanizing embrittlement is also a reflection of strain

aging with the short time exposure at 850° F of the galvanizing treatment constituting an accelerated aging treatment.

SECTION 6

EFFECT OF RESIDUAL ELEMENTS

In addition to the carbon, manganese, phosphorus, sulphur and silicon which are always present, carbon steels may contain small amounts of other elements. These include gases, such as hydrogen, oxygen or nitrogen which are introduced during the steelmaking process, nickel, copper, molybdenum, chromium and tin which may be present in the scrap, and aluminum, titanium, vanadium, zirconium or boron which may be introduced during the deoxidation process.

The effects of oxygen and nitrogen have been discussed in the section on aging, and their effect on aging is their principal effect in carbon steel.

In steel, hydrogen has a definite embrittling effect, the mechanism of which is not well understood. Although hydrogen will diffuse out of steel at room temperature if sufficient time is allowed, tension tests on carbon steels will show low ductility if made soon after rolling and the ductility will increase on aging at room temperature or after shorter times at slightly elevated temperatures. This effect is, of course, more pronounced in larger section sizes because of the longer time required for the diffusion of hydrogen to the surface. Hydrogen contents of about 0.0005 per cent will give rise to this effect and such contents are common in as-rolled steels.

Hydrogen also plays an important role in the phenomenon known as **flaking** which is manifested as internal cracks or bursts, usually occurring during the cooling from rolling or forging. The phenomenon is more pronounced in heavy sections and in the higher carbon steels. In carbon steels, flaking may be prevented by slow cooling after rolling or forging. This slow-cooling operation presumably permits the hydrogen to diffuse out of the steel and thereby minimizes the susceptibility to flaking. Such a controlled slow-cooling operation after rolling is now standard practice in the manufacture of rails, and this practice has practically eliminated the

occurrence of flaking and resultant occasional "transverse fissure" failures.

The alloying elements, such as nickel, chromium, molybdenum and copper, which may be introduced in the scrap will, of course, increase the hardenability of carbon steels, although, since the percentages are ordinarily low, this effect will usually not be large. It may, however, change the heat-treating characteristics and for applications in which ductility is important, such as steels for deep drawing, the increased hardness from these residual elements may be serious.

Tin in relatively low amounts is harmful in steels for deep drawing, but for most applications the effect of tin in the amounts ordinarily present is negligible.

Aluminum, as described above, is generally desirable since it acts as a grain refiner and tends to decrease the susceptibility to strain aging. It has the disadvantage, however, that it tends to promote graphitization and is, therefore, undesirable in steels to be used for high-temperature applications. The other elements which may be introduced as deoxidizers, titanium, vanadium or zirconium, are, unless intentionally added, ordinarily present in such small amounts as to be generally ineffective.

Boron in amounts as low as 0.001 per cent may markedly increase hardenability, but the boron contents are ordinarily well below this value and are ineffective.

Chapter 44

ALLOY STEELS

Introductory—Alloy steels may be defined as those steels which owe their enhanced properties to the presence of one or more special elements or to the presence of larger proportions of elements such as manganese and silicon than are ordinarily present in carbon steel. The major classifications of steels containing alloying elements are as follows:

1. High-strength low-alloy steels
2. AISI alloy steels
3. Alloy tool steels

4. Stainless steels
5. Heat-resisting steels
6. Electrical steels (silicon steels)

The high-strength low-alloy steels, stainless steels, heat-resisting steels, and electrical steels are discussed in other chapters. This chapter covers the AISI alloy steels, often referred to commonly as "constructional alloy steels," and the alloy tool steels, with an introductory discussion of the functions of the alloying elements.

SECTION 1

FUNCTIONS OF THE ALLOYING ELEMENTS

As stated above, alloying elements are added to steel to enhance its properties. In the broadest sense, alloy steels may contain up to approximately 50 per cent of alloying elements, and the enhancement of properties may be a specific and direct function of the alloying elements, as in the instances of the increased corrosion resistance of the high-chromium steels and the enhanced electrical properties of the silicon steels. In the narrower and more technical sense, however, the term "alloy steels" refers to the heat-treatable alloy constructional and automotive steels which contain from about one to three or four per cent of alloying elements. The American Iron and Steel Institute definition of alloy steel is as follows: "By common custom steel is considered to be alloy steel when the maximum of the range given for the content of alloying elements exceeds one or more of the following limits: manganese, 1.65 per cent; silicon, 0.60 per cent; copper, 0.60 per cent; or in which a definite range or a definite minimum quantity of any of the following elements is specified or required within the limits of the recognized field of constructional alloy steels: aluminum, boron, chromium up to 3.99 per cent, cobalt, columbium, molybdenum, nickel, titanium, tungsten, vanadium, zirconium, or any other alloying element added to obtain a desired alloying effect."

Such steels have been standardized and classified jointly by the American Iron and Steel Institute and the Society of Automotive Engineers, and represent by far the largest tonnage of alloy steels. Alloy steels of this type are generally known as "AISI alloy steels" and will be so designated in this chapter. As previously stated, they are also commonly referred to as "constructional alloy steels." The composition of these steels is shown in Table 44—I.

As was emphasized in the chapter on heat treatment, the mechanical properties of steel are dependent upon its microstructure. In the AISI alloy steels the effect of the alloying is indirect; i.e., through their influence on the microstructure of the material. The AISI alloy steels make it possible to attain desirable microstructures and corresponding desirable properties over a very much

wider range of sizes and sections than is possible with the carbon steels.

Hardenability—The mechanism by which the alloying elements affect the microstructure obtained with a given heat treatment is discussed in the chapter on heat treatment. It is shown that the alloying elements in general decrease the rates of transformation of austenite at sub-critical temperatures, thereby facilitating the attainment of low-temperature transformation to martensite or lower bainite when these are the end products desired, without prior transformation to unwanted higher temperature products. It was pointed out that this function of the alloying elements could be evaluated and expressed in terms of the property known as hardenability. Alloying elements thus control microstructure through their effect on hardenability, and this hardenability effect is by far their most important function.

It was also shown in the chapter on heat treatment that the properties of tempered martensite, which represents the most desirable microstructure in respect to strength and toughness, were characteristic of the microstructure rather than of the composition. Thus, alloy steels of equal hardenabilities, but utilizing different combinations of alloying elements, are generally interchangeable for heat treatment to produce this microstructure. This principle permits an intelligent choice of alloy combinations which, for reasons of economy or availability, are best suited for particular applications. This principle was widely used during World War II to develop substitute compositions utilizing the alloying elements most available at the time, thereby conserving the scarcer alloying elements.

Effects of the Alloying Elements on Hardenability—**The Multiplying Factor Principle**—The effects of the alloys on grain size or hardenability may be quantitatively evaluated by hardenability measurements on a series of steels in which a single alloying element is the only variable. This method is illustrated by Figure 44—1 which shows the hardenabilities of two series of steels in terms of ideal diameter for a microstructure of 50 per cent martensite. These series were made by additions of phosphorus to successive ingots so that in each series

**Table 44—I. Chemical Compositions of Open Hearth and Electric Furnace Alloy Steels
(Bars, Billets, Blooms and Slabs).**

(The ranges and limits in this table apply to steel not exceeding 200 sq. in. in cross-sectional area.)

Chemical Composition Ranges and Limits, per cent									Corresponding SAE Number
AISI Number		Mn	P (Max.)	S (Max.)	Ni	Cr	Mo	V (Min.)	
1330	0.28/0.33	1.60/1.90	0.040	0.040	0.20/0.35	—	—	—	1330
1335	0.33/0.38	1.60/1.90	0.040	0.010	0.20/0.35	—	—	—	1335
1340	0.38/0.43	1.60/1.90	0.040	0.010	0.20/0.35	—	—	—	1340
1345	0.43/0.48	1.60/1.90	0.040	0.010	0.20/0.35	—	—	—	1345
2317	0.15/0.20	0.40/0.60	0.040	0.010	0.20/0.35	3.25/3.75	—	—	2317
2515	0.12/0.17	0.40/0.60	0.040	0.040	0.20/0.35	4.75/5.25	—	—	2515
E2517	0.15/0.20	0.45/0.60	0.025	0.025	0.20/0.35	4.75/5.25	—	—	2517
3120	0.17/0.22	0.60/0.80	0.010	0.010	0.20/0.35	1.10/1.40	0.55/0.75	—	3120
3130	0.28/0.33	0.60/0.80	0.010	0.010	0.20/0.35	1.10/1.40	0.55/0.75	—	3130
3135	0.33/0.38	0.60/0.80	0.010	0.010	0.20/0.35	1.10/1.40	0.55/0.75	—	3135
3140	0.38/0.43	0.70/0.90	0.010	0.040	0.20/0.35	1.10/1.40	0.55/0.75	—	3140
E3310	0.08/0.13	0.45/0.60	0.025	0.025	0.20/0.35	3.25/3.75	1.40/1.75	—	3310
E3316	0.14/0.19	0.45/0.60	0.025	0.025	0.20/0.35	3.25/3.75	1.40/1.75	—	3316
4023	0.20/0.25	0.70/0.90	0.010	0.010	0.20/0.35	—	—	0.20/0.30	4023
4024	0.20/0.25	0.70/0.90	0.040	0.035/ 0.050	0.20/0.35	—	—	0.20/0.30	4024
4027	0.25/0.30	0.70/0.90	0.040	0.040	0.20/0.35	—	—	0.20/0.30	4027
4028	0.25/0.30	0.70/0.90	0.040	0.035/ 0.050	0.20/0.35	—	—	0.20/0.30	4028
4032	0.30/0.35	0.70/0.90	0.040	0.040	0.20/0.35	—	—	0.20/0.30	4032
4037	0.35/0.40	0.70/0.90	0.010	0.010	0.20/0.35	—	—	0.20/0.30	4037
4042	0.40/0.45	0.70/0.90	0.010	0.040	0.20/0.35	—	—	0.20/0.30	4042
4047	0.45/0.50	0.70/0.90	0.010	0.010	0.20/0.35	—	—	0.20/0.30	4047
4053	0.50/0.56	0.75/1.00	0.010	0.040	0.20/0.35	—	—	0.20/0.30	4053
4063	0.60/0.67	0.75/1.00	0.040	0.010	0.20/0.35	—	—	0.20/0.30	4063
4068	0.63/0.70	0.75/1.00	0.010	0.010	0.20/0.35	—	—	0.20/0.30	4068
4118	0.18/0.23	0.70/0.90	0.010	0.010	0.20/0.35	—	0.40/0.60	0.08/0.15	4118
4130	0.28/0.33	0.40/0.60	0.010	0.010	0.20/0.35	—	0.80/1.10	0.15/0.25	4130
TS 4130	0.28/0.33	0.45/0.65	0.010	0.040	0.20/0.35	—	0.90/1.20	0.08/0.15	—
TS 4132	0.30/0.35	0.45/0.65	0.010	0.040	0.20/0.35	—	0.90/1.20	0.08/0.15	—
4135	0.33/0.38	0.70/0.90	0.010	0.010	0.20/0.35	—	0.80/1.10	0.15/0.25	4135
4137	0.35/0.40	0.70/0.90	0.010	0.010	0.20/0.35	—	0.80/1.10	0.15/0.25	4137
TS 4137	0.35/0.40	0.75/1.00	0.010	0.010	0.20/0.35	—	0.90/1.20	0.08/0.15	—
4140	0.38/0.43	0.75/1.00	0.040	0.010	0.20/0.35	—	0.80/1.10	0.15/0.25	4140
TS 4140	0.38/0.43	0.80/1.05	0.010	0.040	0.20/0.35	—	0.90/1.20	0.08/0.15	—
4142	0.40/0.45	0.75/1.00	0.010	0.040	0.20/0.35	—	0.80/1.10	0.15/0.25	4142
TS 4142	0.40/0.45	0.80/1.05	0.040	0.040	0.20/0.35	—	0.90/1.20	0.08/0.15	—
4145	0.43/0.48	0.75/1.00	0.010	0.010	0.20/0.35	—	0.80/1.10	0.15/0.25	4145
TS 4145	0.43/0.48	0.80/1.05	0.010	0.040	0.20/0.35	—	0.90/1.20	0.08/0.15	—
4147	0.45/0.50	0.75/1.00	0.040	0.010	0.20/0.35	—	0.80/1.10	0.15/0.25	4147
4150	0.48/0.53	0.75/1.00	0.010	0.010	0.20/0.35	—	0.80/1.10	0.15/0.25	4150
TS 4150	0.48/0.53	0.80/1.05	0.010	0.010	0.20/0.35	—	0.90/1.20	0.08/0.15	—
4320	0.17/0.22	0.45/0.65	0.010	0.010	0.20/0.35	1.65/2.00	0.40/0.60	0.20/0.30	4320
4337	0.35/0.40	0.60/0.80	0.010	0.010	0.20/0.35	1.65/2.00	0.70/0.90	0.20/0.30	4337
E4337	0.35/0.40	0.65/0.85	0.025	0.025	0.20/0.35	1.65/2.00	0.70/0.90	0.20/0.30	—
4340	0.38/0.43	0.60/0.80	0.040	0.040	0.20/0.35	1.65/2.00	0.70/0.90	0.20/0.30	4340
E4340	0.38/0.43	0.65/0.85	0.025	0.025	0.20/0.35	1.65/2.00	0.70/0.90	0.20/0.30	E4340
4608	0.06/0.11	0.25/0.45	0.040	0.010	0.25 Max.	1.40/1.75	—	0.15/0.25	4608
4615	0.13/0.18	0.45/0.65	0.040	0.040	0.20/0.35	1.65/2.00	—	0.20/0.30	4615
4617	0.15/0.20	0.45/0.65	0.040	0.040	0.20/0.35	1.65/2.00	—	0.20/0.30	4617
4620	0.17/0.22	0.45/0.65	0.040	0.040	0.20/0.35	1.65/2.00	—	0.20/0.30	4620
X4620	0.18/0.23	0.50/0.70	0.010	0.040	0.20/0.35	1.65/2.00	—	0.20/0.30	X4620
4621	0.18/0.23	0.70/0.90	0.010	0.010	0.20/0.35	1.65/2.00	—	0.20/0.30	4621
4640	0.38/0.43	0.60/0.80	0.040	0.010	0.20/0.35	1.65/2.00	—	0.20/0.30	4640
4720	0.17/0.22	0.50/0.70	0.010	0.010	0.20/0.35	0.90/1.20	0.35/0.55	0.15/0.25	4720
4812	0.10/0.15	0.40/0.60	0.040	0.010	0.20/0.35	3.25/3.75	—	0.20/0.30	4812
4815	0.13/0.18	0.40/0.60	0.040	0.040	0.20/0.35	3.25/3.75	—	0.20/0.30	4815
4817	0.15/0.20	0.40/0.60	0.010	0.040	0.20/0.35	3.25/3.75	—	0.20/0.30	4817
4820	0.18/0.23	0.50/0.70	0.040	0.040	0.20/0.35	3.25/3.75	—	0.20/0.30	4820
5015	0.12/0.17	0.30/0.50	0.040	0.040	0.20/0.35	—	0.30/0.50	—	5015

(Continued on next page)

Table 44—I (Continued). Chemical Compositions of Open Hearth and Electric Furnace Alloy Steels
(Bars, Billets, Blooms and Slabs).

(The ranges and limits in this table apply to steel not exceeding 200 sq. in. in cross-sectional area.)

AISI Number	Chemical Composition Ranges and Limits, per cent.									Corre- spond- ing SAE Number
	C	Mn	P (Max.)	S (Max.)	Si	Ni	Cr	Mo	V (Min.)	
5046	0.43/0.50	0.75/1.00	0.040	0.040	0.20/0.35	—	0.20/0.35	—	—	5046
5117	0.15/0.20	0.70/0.90	0.040	0.040	0.20/0.35	—	0.70/0.90	—	—	5117
5120	0.17/0.22	0.70/0.90	0.040	0.040	0.20/0.35	—	0.70/0.90	—	—	5120
5130	0.28/0.33	0.70/0.90	0.040	0.040	0.20/0.35	—	0.80/1.10	—	—	5130
5132	0.30/0.35	0.60/0.80	0.040	0.040	0.20/0.35	—	0.75/1.00	—	—	5132
5135	0.33/0.38	0.60/0.80	0.040	0.040	0.20/0.35	—	0.80/1.05	—	—	5135
5140	0.38/0.43	0.70/0.90	0.040	0.040	0.20/0.35	—	0.70/0.90	—	—	5140
5145	0.43/0.48	0.70/0.90	0.040	0.040	0.20/0.35	—	0.70/0.90	—	—	5145
5147	0.45/0.52	0.70/0.95	0.040	0.040	0.20/0.35	—	0.85/1.15	—	—	5147
5150	0.48/0.53	0.70/0.90	0.040	0.040	0.20/0.35	—	0.70/0.90	—	—	5150
5152	0.48/0.55	0.70/0.90	0.040	0.040	0.20/0.35	—	0.90/1.20	—	—	5152
5155	0.50/0.60	0.70/0.90	0.040	0.040	0.20/0.35	—	0.70/0.90	—	—	5155
5160	0.55/0.65	0.75/1.00	0.040	0.040	0.20/0.35	—	0.70/0.90	—	—	5160
E50100	0.95/1.10	0.25/0.45	0.025	0.025	0.20/0.35	—	0.40/0.60	—	—	50100
E51100	0.95/1.10	0.25/0.45	0.025	0.025	0.20/0.35	—	0.90/1.15	—	—	51100
E52100	0.95/1.10	0.25/0.45	0.025	0.025	0.20/0.35	—	1.30/1.60	—	—	52100
6117	0.15/0.20	0.70/0.90	0.040	0.040	0.20/0.35	—	0.70/0.90	—	0.10	6117
6120	0.17/0.22	0.70/0.90	0.040	0.040	0.20/0.35	—	0.70/0.90	—	0.10	6120
6145	0.43/0.48	0.70/0.90	0.040	0.040	0.20/0.35	—	0.80/1.10	—	0.15	6145
6150	0.48/0.53	0.70/0.90	0.040	0.040	0.20/0.35	—	0.80/1.10	—	0.15	6150
TS 8115	0.13/0.18	0.70/0.90	0.040	0.040	0.20/0.35	0.20/0.40	0.30/0.50	0.08/0.15	—	—
TS 8120	0.18/0.23	0.70/0.90	0.040	0.040	0.20/0.35	0.20/0.40	0.30/0.50	0.08/0.15	—	—
TS 8122	0.20/0.25	0.70/0.90	0.040	0.040	0.20/0.35	0.20/0.40	0.30/0.50	0.08/0.15	—	—
TS 8125	0.23/0.28	0.70/0.90	0.040	0.040	0.20/0.35	0.20/0.40	0.30/0.50	0.08/0.15	—	—
TS 8127	0.25/0.30	0.70/0.90	0.040	0.040	0.20/0.35	0.20/0.40	0.30/0.50	0.08/0.15	—	—
8615	0.13/0.18	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8615
TS 8615	0.13/0.18	0.70/0.90	0.040	0.040	0.20/0.35	0.30/0.60	0.55/0.75	0.08/0.15	—	—
8617	0.15/0.20	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8617
TS 8617	0.15/0.20	0.70/0.90	0.040	0.040	0.20/0.35	0.30/0.60	0.55/0.75	0.08/0.15	—	—
8620	0.18/0.23	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8620
TS 8620	0.18/0.23	0.70/0.90	0.040	0.040	0.20/0.35	0.30/0.60	0.55/0.75	0.08/0.15	—	—
8622	0.20/0.25	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8622
8625	0.23/0.28	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8625
8627	0.25/0.30	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8627
8630	0.28/0.33	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8630
8635	0.33/0.38	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8635
8637	0.35/0.40	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8637
8640	0.38/0.43	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8640
8641	0.38/0.43	0.75/1.00	0.040	0.040/0.060	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8641
8642	0.40/0.45	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8642
8645	0.43/0.48	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8645
8650	0.48/0.53	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8650
8653	0.50/0.56	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.50/0.80	0.15/0.25	—	8653
8655	0.50/0.60	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8655
8660	0.55/0.65	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—	8660
8715	0.13/0.18	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.20/0.30	—	8715
8717	0.15/0.20	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.20/0.30	—	8717
8720	0.18/0.23	0.70/0.90	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.20/0.30	—	8720
8735	0.33/0.38	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.20/0.30	—	—
8740	0.38/0.43	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.20/0.30	—	8740
8742	0.40/0.45	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.20/0.30	—	8742
8750	0.48/0.53	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.20/0.30	—	8750
9255	0.50/0.60	0.70/0.95	0.040	0.040	1.80/2.20	—	—	—	—	9255
9260	0.55/0.65	0.70/1.00	0.040	0.040	1.80/2.20	—	—	—	—	9260
9261	0.55/0.65	0.75/1.00	0.040	0.040	1.80/2.20	—	0.10/0.25	—	—	9261
9262	0.55/0.65	0.75/1.00	0.040	0.040	1.80/2.20	—	0.25/0.40	—	—	9262
E9310	0.08/0.13	0.45/0.65	0.025	0.025	0.20/0.35	3.00/3.50	1.00/1.40	0.08/0.15	—	9310
E9314	0.11/0.17	0.40/0.70	0.025	0.025	0.20/0.35	3.00/3.50	1.00/1.40	0.08/0.15	—	—

(Continued on next page)

Table 44—I (Continued). Chemical Compositions of Open Hearth and Electric Furnace Alloy Steels
(Bars, Billets, Blooms and Slabs).

(The ranges and limits in this table apply to steel not exceeding 200 sq. in. in cross-sectional area.)

AISI Number	Chemical Composition Ranges and Limits, per cent								Corre- spond- ing SAE Number
	C	Mn	P (Max.)	S (Max.)	Si	Ni	Cr	Mo	V (Min.)
9840	0.38/0.43	0.70/0.90	0.040	0.010	0.20/0.35	0.85/1.15	0.70/0.90	0.20/0.30	—
9845	0.43/0.48	0.70/0.90	0.040	0.040	0.20/0.35	0.85/1.15	0.70/0.90	0.20/0.30	—
9850	0.48/0.53	0.70/0.90	0.040	0.040	0.20/0.35	0.85/1.15	0.70/0.90	0.20/0.30	—

BORON STEELS

(These steels can be expected to have 0.0005 per cent minimum boron content.)

AISI Number	Chemical Composition Ranges and Limits, per cent								Corre- spond- ing SAE Number
	C	Mn	P (Max.)	S (Max.)	Si	Ni	Cr	Mo	V (Min.)
TS 14B35	0.33/0.38	0.75/1.00	0.040	0.010	0.20/0.35	—	—	—	—
TS 14B50	0.48/0.53	0.75/1.00	0.040	0.040	0.20/0.35	—	—	—	—
TS 43BV12	0.08/0.13	0.75/1.00	0.040	0.040	0.20/0.40	1.65/2.00	0.40/0.60	0.20/0.30	0.03
TS 43BV14	0.10/0.15	0.45/0.65	0.010	0.010	0.20/0.35	1.65/2.00	0.40/0.60	0.08/0.15	0.03
46B12	0.10/0.15	0.45/0.65	0.040	0.040	0.20/0.35	1.65/2.00	—	0.20/0.30	—
50B30	0.27/0.34	0.70/1.00	0.010	0.010	0.20/0.35	—	0.35/0.60	—	—
50B35	0.32/0.39	0.70/1.00	0.010	0.040	0.20/0.35	—	0.35/0.60	—	—
50B40	0.37/0.45	0.70/1.00	0.010	0.010	0.20/0.35	—	0.35/0.60	—	—
50B44	0.42/0.50	0.70/1.00	0.010	0.010	0.20/0.35	—	0.35/0.60	—	—
50B46	0.43/0.50	0.75/1.00	0.010	0.040	0.20/0.35	—	0.20/0.35	—	—
50B50	0.48/0.53	0.75/1.00	0.040	0.040	0.20/0.35	—	0.40/0.60	—	—
50B60	0.55/0.65	0.75/1.00	0.010	0.010	0.20/0.35	—	0.40/0.60	—	—
51B60	0.55/0.65	0.75/1.00	0.040	0.040	0.20/0.35	—	0.70/0.90	—	—
81B45	0.43/0.48	0.75/1.00	0.010	0.040	0.20/0.35	0.20/0.10	0.35/0.55	0.08/0.15	—
86B45	0.43/0.48	0.75/1.00	0.040	0.040	0.20/0.35	0.40/0.70	0.40/0.60	0.15/0.25	—
94B15	0.13/0.18	0.75/1.00	0.040	0.040	0.20/0.35	0.30/0.60	0.30/0.50	0.08/0.15	—
94B17	0.15/0.20	0.75/1.00	0.040	0.010	0.20/0.35	0.30/0.60	0.30/0.50	0.08/0.15	—
TS 94B30	0.28/0.33	0.75/1.00	0.040	0.040	0.20/0.35	0.30/0.60	0.30/0.50	0.08/0.15	—
TS 94B40	0.38/0.43	0.75/1.00	0.040	0.010	0.20/0.35	0.30/0.60	0.30/0.50	0.08/0.15	—

NOTE 1. Grades shown in the above list with prefix letter E generally are manufactured by the basic electric furnace process. All others are normally manufactured by the basic open hearth process but may be manufactured by the basic electric furnace process with adjustments in phosphorus and sulphur.

NOTE 2. The phosphorus and sulphur limitations for each process are as follows:

Basic electric furnace—0.025 maximum per cent

Basic open hearth —0.040 maximum per cent

Acid electric furnace—0.050 maximum per cent

Acid open hearth —0.050 maximum per cent

NOTE 3. Minimum silicon limit for acid open hearth or acid electric furnace alloy steel is 0.15 per cent.

NOTE 4. Small quantities of certain elements are present in alloy steels which are not specified or required. These elements are considered as incidental and may be present to the following maximum amounts: Copper, 0.35 per cent; Nickel, 0.25 per cent; Chromium, 0.20 per cent and Molybdenum, 0.06 per cent.

NOTE 5. Where minimum and maximum sulphur content is shown it is indicative of resulphurized steels.

NOTE 6. TS denotes tentative standard steels.

the composition was constant except for the phosphorus. It will be noted that the hardenability increases regularly with increasing phosphorus content, and that the rate of increase is more rapid for the steel of the higher base hardenability. In order to obtain a numerical evaluation of the effect of phosphorus on hardenability, the hardenabilities of the steels containing phosphorus may be divided by the base hardenability of the steel containing no phosphorus. This value expressing the effect of the element on hardenability is known as a **multiplying factor**, and the multiplying factors for phos-

phorus, derived from the series in Figure 44—1, are plotted in Figure 44—2. It will be seen that a steel with 0.020 per cent phosphorus will have 1.05 times the hardenability of a steel with no phosphorus, while a steel with 0.100 per cent phosphorus would have roughly one and one quarter times the base hardenability. Also, with this information, the effect of increasing the phosphorus content from 0.020 to 0.100 per cent can be evaluated by multiplying the hardenability of the 0.020 per cent phosphorus steel by $\frac{1.27}{1.05}$ or 1.21.

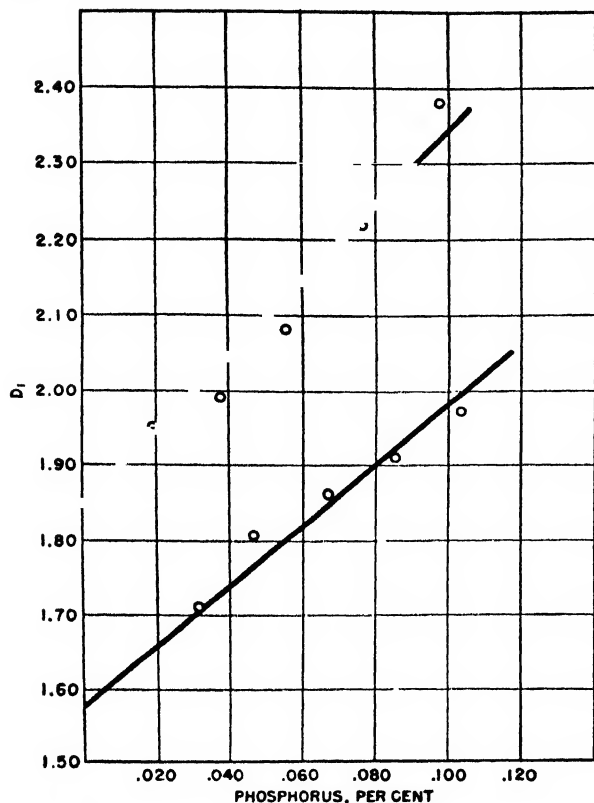


FIG. 44-1. Hardenability as a function of phosphorus content in two series of steels.

The pioneer work of this nature was done by Grossmann in 1941. Hardenability factors for many of the common alloying elements, as well as hardenability values for pure iron-carbon alloys and the effect of grain size, were evaluated at that time. These factors have, in some cases, been modified by the work of other investigators and the values as published by the American Iron and Steel Institute are shown for alloying elements in Figure 44-3, and for grain size in Figure 44-4.

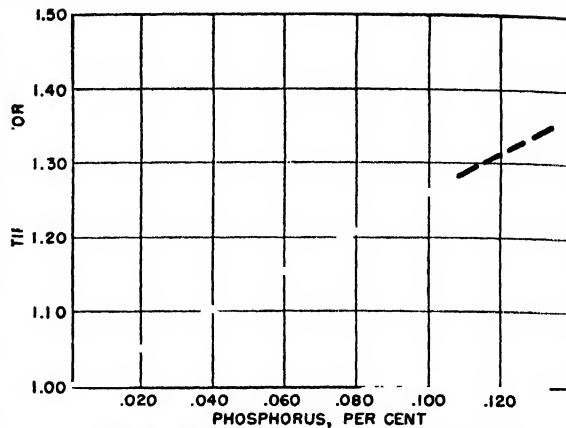


FIG. 44-2. Multiplying factors for phosphorus.

Grossmann and his associates further found that the cumulative effects of alloying elements on hardenability could be evaluated by multiplying the base hardenability of the iron-carbon alloy progressively by the multiplying factors for the elements. Thus the chart of Figure 44-3 enables one to calculate the hardenability in terms of 50 per cent martensite microstructures of a given alloy combination.

The element boron increases the hardenability of steel but its effect differs from that of the other alloying elements since additions over a certain optimum amount produce no further increase in hardenability. Experience has shown that the optimum content is 0.0005 to 0.004 per cent boron. The multiplying factor for boron depends not on boron content, but on carbon content and varies from approximately 2.00 for 0.20 per cent carbon steels to nearly zero for 0.90 per cent carbon.

The multiplying factor principle is of importance not only as a means of predicting the approximate hardenability of a steel from its composition, but also since it shows that, in general, the addition of relatively small amounts of several alloying elements is more effective in increasing hardenability than a relatively large amount of a single element. This principle was also widely applied during World War II to the development of substitute compositions which would utilize to the

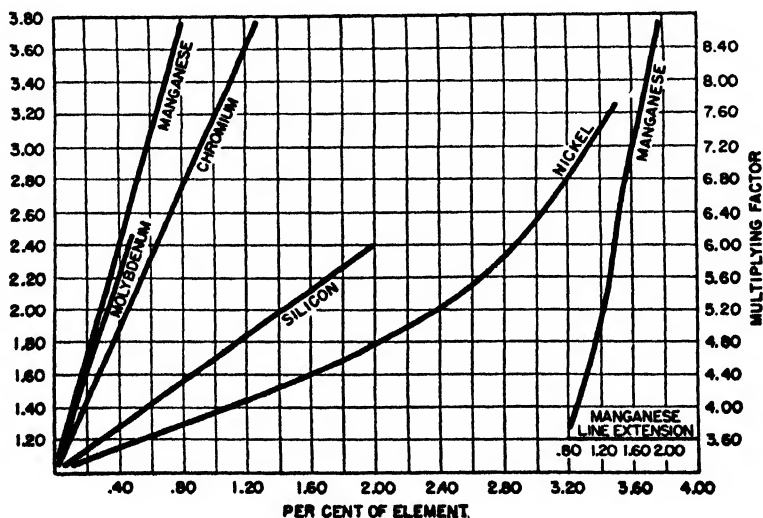


FIG. 44-3. Multiplying factors for a variety of alloying elements (American Iron and Steel Institute).

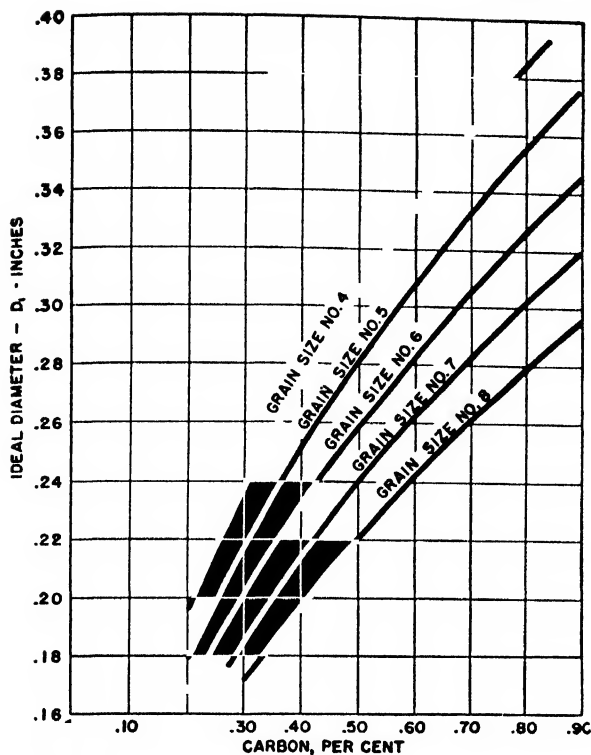


FIG. 44-4. Effect of grain size on the hardenability of pure iron-carbon alloys, expressed as ideal critical diameter, D_i .

fullest extent the hardenability effects of the alloying elements most available.

Effects of the Alloys in Tempering—While the martensitic microstructure has been stressed as a criterion of hardenability, it must of course be realized that martensite itself is very brittle, and that in order to realize the properties generally sought in machine parts, this martensite must be reheated or tempered. It is essential to an understanding of alloy steels, therefore, to consider the roles played by the alloying elements in this tempering process, and the manner in which they will affect the behavior on tempering.

The primary purpose of tempering is to impart a degree of plasticity or toughness to the steel to alleviate the brittleness of the martensite. Although this process may, and usually does, soften the steel, this softening is only incidental to the very important increase in toughness. This increase in toughness after tempering reflects two effects of tempering: (1) the relief of internal stresses set up by the quenching operation; and (2) a crystallization, coalescence and spheroidization of iron and alloy carbides, resulting in a microstructure of greater plasticity.

The addition of alloying elements which increase hardenability may be very helpful in decreasing the magnitude of the internal stresses resulting from the quench, since they will permit the attainment of a martensitic microstructure with a less drastic quench. For this reason, the use of an alloy steel and a mild quench for an application requiring high hardness, and, therefore, a low tempering temperature with an accompanying relatively low degree of stress relief, may be very advantageous. It should be noted, however, that this is only secondarily an effect of alloys in tempering; here again the primary function of the alloying elements is to increase hardenability.

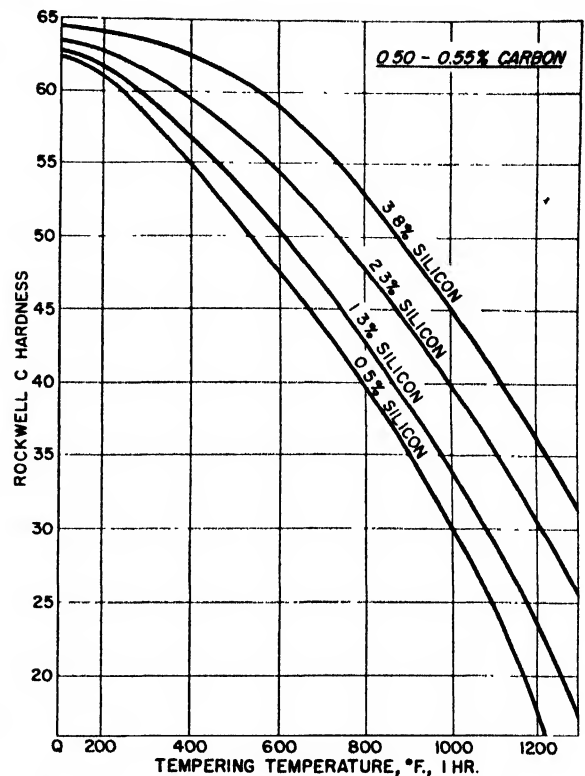


FIG. 44-5. The effect of silicon on tempering rate of 0.50-0.55% carbon steel, tempered for 1 hour at indicated temperatures.

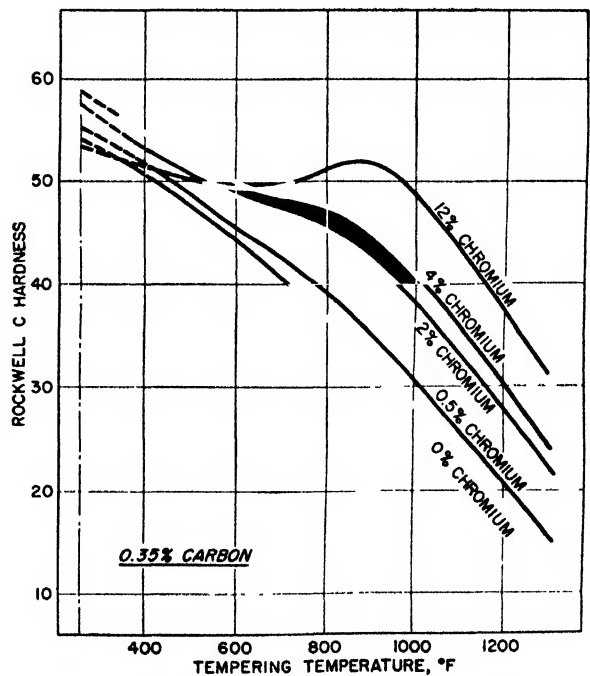


FIG. 44-6. The effect of chromium on tempering rate of 0.35% carbon steel, tempered for 1 hour at indicated temperatures.

The alloying elements will, however, have a direct and significant effect upon the second behavior, that of crystallization and coalescence of the carbides. In general, the effect of alloying elements will be to slow up the processes of crystallization and coalescence. This means that an alloy steel will customarily require higher tempering temperatures, or longer times at temperature, to obtain a given hardness.

The effects of some of the individual alloying elements on the tempering rate are illustrated in Figure 44-5 for silicon, Figure 44-6 for chromium, Figure 44-7 for molybdenum and Figure 44-8 for vanadium. These charts show the hardness of tempered martensite in these steels after tempering one hour at the indicated tempering temperature.

The effects of nickel and manganese, as in the case of silicon, while they are significant, are quite moderate, and the hardness changes are nearly a direct function of the tempering temperature. This type of behavior is characteristic of alloys which dissolve largely in the ferrite phase, and do not tend to form carbides.

Boron in the amount used to increase hardenability has no perceptible effect on hardness changes during tempering.

The carbide-forming elements, such as chromium, molybdenum or vanadium, however, have very marked effects on the tempering behavior. Elements of this type not only raise the tempering temperature to obtain a given hardness, but with the higher percentages of these elements, the rate of softening is no longer a continuous function of the tempering temperature. In steels of this type, such as the 0.5 per cent molybdenum steel of Figure 44-7, there is a tempering temperature range in which the softening is retarded or, with still higher alloy content as in the 2.0 per cent molybdenum steel, in which the hardness first decreases, then increases somewhat before continuing the decline with increasing tempering temperature. The retardation is also observed in steels containing more than one carbide-forming ele-

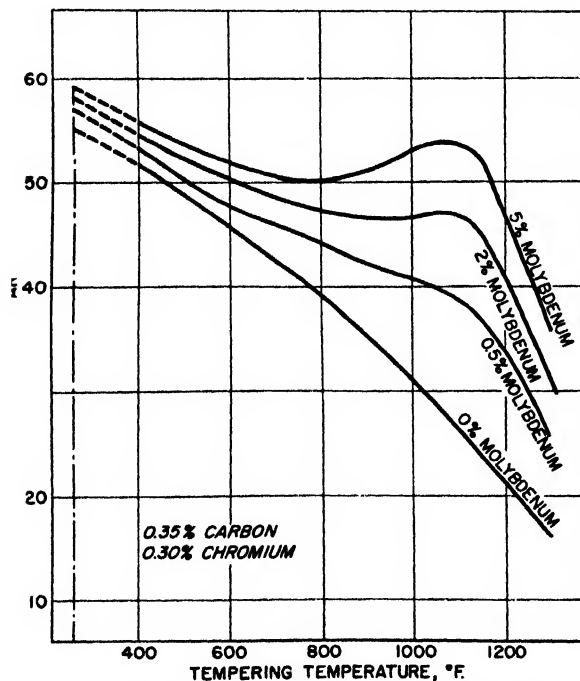


FIG. 44-7. The effect of molybdenum on tempering rate of 0.35% carbon, 0.30% chromium steel, tempered for 1 hour at the indicated temperatures.

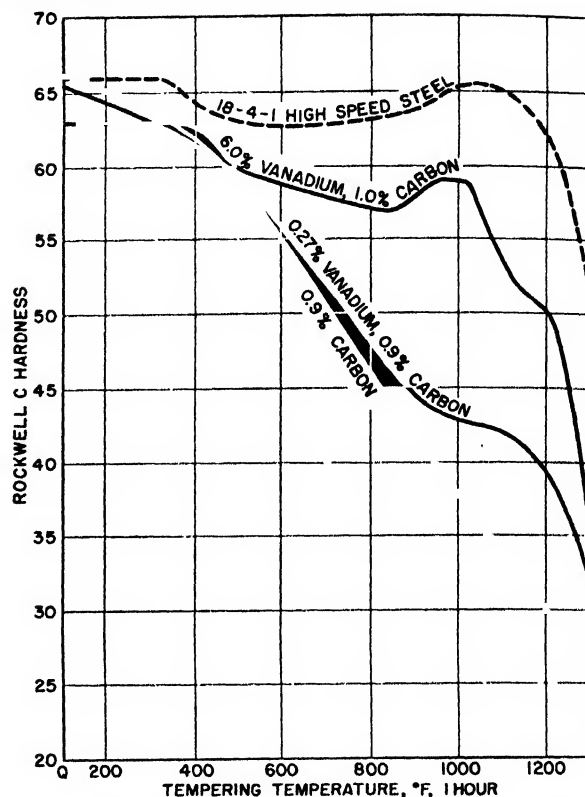


FIG. 44-8. The effect of vanadium on tempering rate of steels tempered for 1 hour at the indicated temperatures.

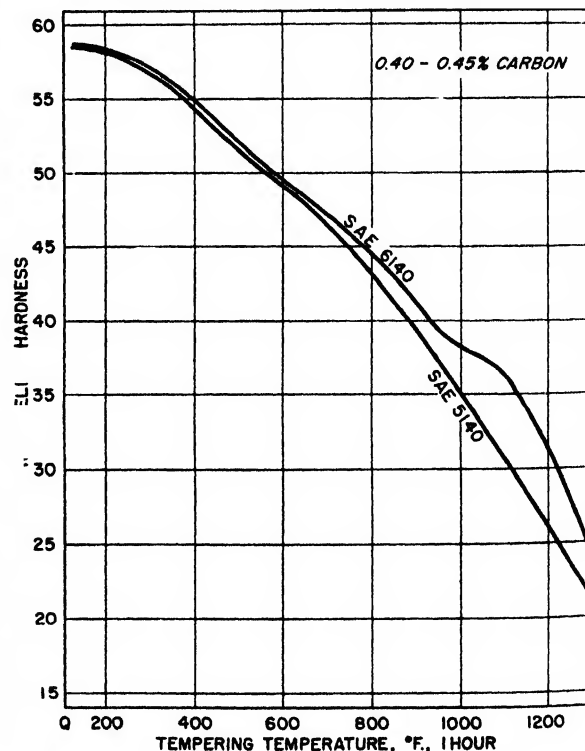


FIG. 44-9. The combined effect of chromium and vanadium on tempering rate of 0.40-0.45% carbon steel tempered for 1 hour at the indicated temperatures. The 6140 steel contains both Cr and V; the 5140, only Cr.

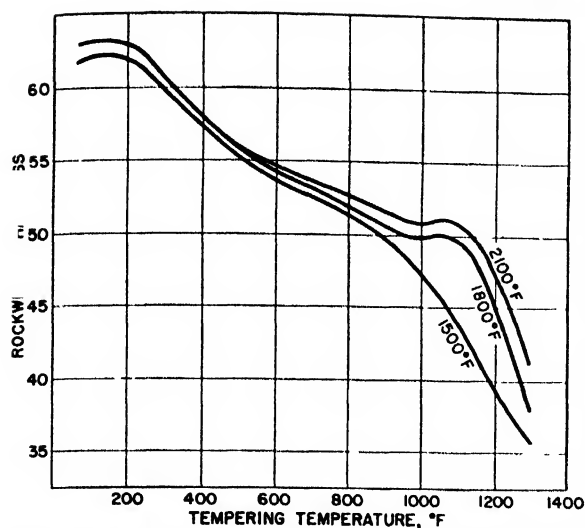


FIG. 44—10. The effect of carbide solution on tempering rate of a Cr-Mo-V steel quenched from 2100°, 1800° and 1500° F.

ment, as illustrated by Figure 44—9, which compares the tempering curve for a 1 per cent chromium steel with that for a chrome-vanadium steel.

In order that a carbide forming element may manifest its full effect upon the tempering behavior, it must dissolve in the austenite at the heating temperature. This is illustrated by Figure 44—10, which shows the tempering behavior of a chrome-molybdenum-vanadium steel after quenching from 1500°, 1800° and 2100° F. It will be noted that the secondary hardening behavior is very marked in the steels quenched from

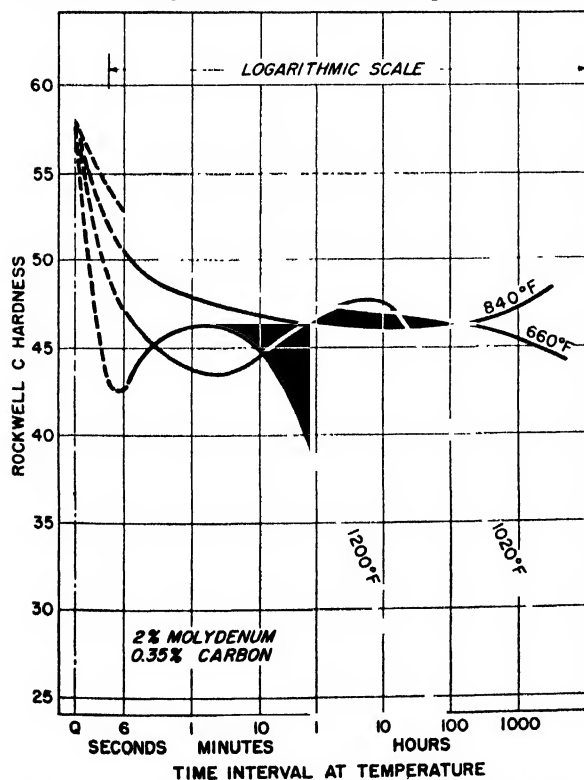


FIG. 44—11. The effect of time of tempering on the secondary hardening behavior in tempering.

the higher temperature, but is almost absent in the steel quenched from 1500° F in which a considerable proportion of the carbides is undissolved.

This secondary hardening effect is also evident in studies of the effect of time at a given tempering temperature on the hardness of alloy steels with carbide forming elements. As an illustration of this, Figure 44—11 shows the manner in which the hardness of a 2.0 per cent molybdenum steel varies with time at several different tempering temperatures. It will be noted that at the highest tempering temperature (1200° F), the secondary hardening effect occurs at times of from 10 seconds to 10 minutes, while at the lowest tempering temperature (660° F), there is no indication of this effect in 1000 hours.

This phenomenon can best be explained on the basis of a delayed precipitation of alloy carbide. Because of the relatively small number of alloy atoms in comparison to the iron atoms, and because of the slow diffusion rate of the alloying elements, the first precipitate to form on tempering will certainly be iron carbide, and the initially rapid drop in hardness represents the coalescence of these iron-carbide particles. However, with longer times and particularly with higher temperatures at which the diffusion rate of the alloys becomes more rapid, some alloy carbide will precipitate, and since this occurs after the coagulation of the iron carbide has progressed to a considerable extent, these fine particles will result in a reversal of the softening action. With relatively low alloy content, this may be manifested as only a decrease in the rate of softening while with high alloy content, an actual increase in hardness may occur as this secondary precipitation proceeds.

In the case of modern alloy steels which are of such a composition as to consist of a matrix of considerable plasticity with a dispersion of rather sizeable undissolved carbides after heat treatment, full advantage is taken of this secondary hardening effect. The composition and heat treatment of these steels is such that, although a number of the alloy carbides remain undissolved, enough are taken into solution to bring about a marked resistance to softening at temperatures up to 1100° F. Such tools can then be used for high-speed machining at relatively high operating temperature without softening.

It might further be mentioned that this effect of the alloying elements, and particularly the carbide-forming elements, on tempering may be reflected in an increased toughness of high-alloy steels. We have seen that the toughness of tempered martensite results from both the relief of internal stresses, and the formation of a desirable carbide dispersion. The higher tempering temperatures for a given strength level, characteristic of these high-alloy steels, will permit a greater degree of stress relief with an accompanying increase in toughness. Furthermore, as we have seen, a given hardness level in the tempered alloy steels reflects not only the state of dispersion and coagulation of the iron carbides, but also that of the alloy carbides, and thus coagulation and spheroidization of the iron carbides must progress further for a given hardness to offset the secondary hardening effect of the alloy carbides. This more completely spheroidized microstructure would also favor plasticity, particularly at hardness levels at which a moderate coalescence of the alloy-carbide dispersion has taken place.

The increase in plasticity on tempering has so far been considered as though it were a continuous effect, that is, as if the steel softened and became more ductile continuously as the tempering temperature is increased. However, this is not altogether true, as many steels ex-

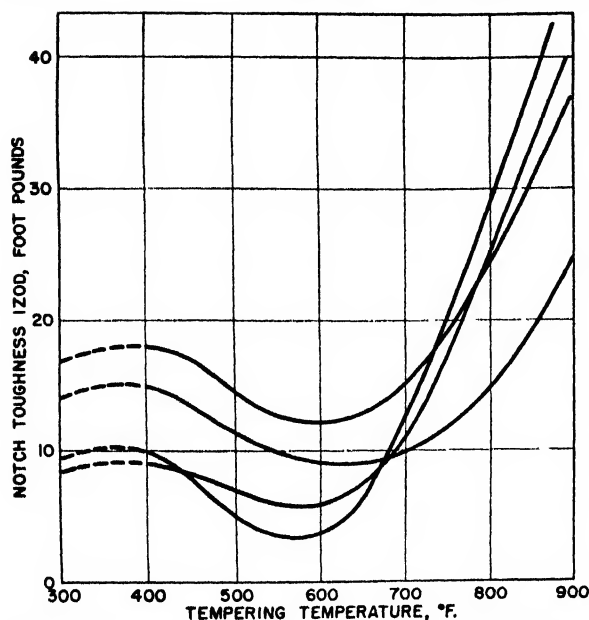


Fig. 44-12. The loss of notch toughness in the Izod test in several alloy steels on tempering at about 600° F.

hibit a minimum toughness on tempering at temperatures of 500°–600° F. This behavior is illustrated in Figure 44-12, which shows the impact values of several alloy steels as a function of tempering temperature. This phenomenon is not fully understood, nor can the effects of the alloying elements on this behavior be evaluated. However, it should be realized that, in general, tempering temperatures in this range (500°–600° F) should be avoided wherever possible.

Temper brittleness in alloy steels is another common example of a discontinuous increase in plasticity on tempering. This phenomenon is manifested as a loss of toughness on slow cooling after tempering at temperatures of 1100° F or above, or on tempering in the temperature range of approximately 850° to 1100° F. Thus, a steel which is susceptible to this type of embrittlement may lose much of its ductility as indicated by a notched-bar impact test on slow cooling from a tempering temperature of 1150° F, although the same steel will be very tough if it is quenched from the same tempering temperature, and this expedient of quenching from the tempering temperature is a common practice to insure freedom from this embrittlement. However, in such steels, embrittlement will also occur on tempering at 850°–1050° F, particularly if the tempering times are protracted and quenching from the tempering temperature will, in such cases, never completely restore the toughness. This phenomenon also is not completely understood, although the behavior suggests that something which dissolves at temperatures of 1100° F and above precipitates in a damaging form at the lower temperatures, either during slow cooling or on reheating to these temperatures. High manganese, phosphorus and chromium contents appear to accentuate this behavior, and molybdenum seems to have a definite retarding effect.

OTHER FUNCTIONS OF THE ALLOYING ELEMENTS

The primary function of the alloying elements in alloy steels has been seen to be that of enhancing the properties through control of the microstructure, particu-

larly in conjunction with suitable heat treatments, and this function has been discussed in considerable detail. However, the alloying elements may exert other useful influences, particularly in steels which may be classed as special purpose steels. These functions will now be briefly discussed.

Ferrite Strengthening—Alloying elements dissolved in pure iron will increase its hardness, and this furnishes a method of increasing the strength of steels in the unhardened state. This ferrite-strengthening function of the alloys is, thus, independent of the effect of the alloys on microstructure, and may be utilized to increase the strength of steels which essentially receive no heat treatment, except for the cooling after the hot-working operation. This hardening effect is, of course, small as compared with that obtainable by changes in the dispersion of the carbide. This is illustrated by Figure 44-13, which shows the hardness of chromium steels as

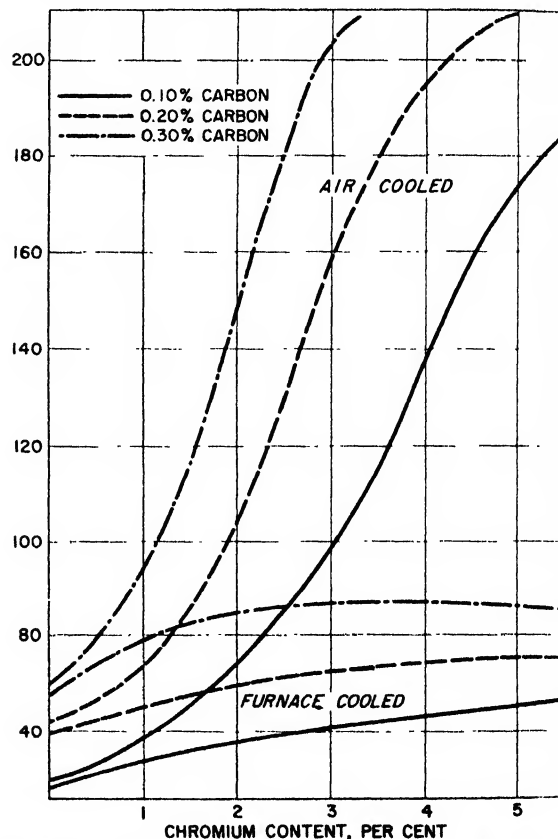


Fig. 44-13. The minor effect of chromium in furnace-cooled steels compared with its strong effect as a strengthener through its influence upon structure in air-cooled steels.

a function of chromium content in a series of steels which have been very slowly cooled, in which the ferrite-strengthening effect will be predominant, as compared with a rapidly cooled series in which the effects of microstructural changes will be predominant. Each of the alloying elements will exert its individual effect in ferrite strengthening. The relative effectiveness of the alloying elements in this respect is indicated by Figure 44-14. The order of increasing effectiveness of these elements appears to be as follows: chromium, tungsten, vanadium, molybdenum, nickel, manganese, silicon, phosphorus. Significant increases in strength may be

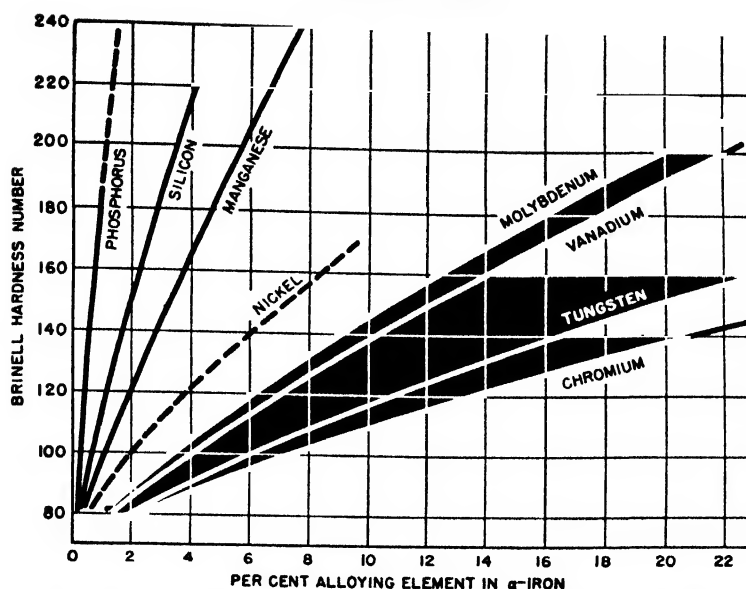


FIG. 44-14. The relative effectiveness of the alloying elements as ferrite strengtheners.

obtained by the use of several elements in a single composition. In general, the higher strength levels obtained by this method of ferrite strengthening will be accompanied by a relatively small loss in plasticity, as compared with the considerable loss in ductility accompanying the hardness increases resulting from microstructural changes.

As discussed in the chapter on the low-alloy high-strength steels, these ferrite-strengthening effects have been utilized to their fullest extent in such steels.

Corrosion Resistance—Another function of some of the alloying elements in steel is to increase its resistance to corrosive attack. This function is also utilized in the low-alloy high-strength steels, in which chromium, copper, and phosphorus have been found very effective in increasing resistance to atmospheric corrosion, thereby permitting the use of the lighter sections made possible by their higher strengths, without decreased life from corrosion.

Chromium in high percentages imparts an extraordinary corrosion resistance to steel, and the stainless steels are largely based on this effect of chromium. This effect is discussed in detail in Chapter 47 on the stainless steels and need not be elaborated on here.

Abrasion Resistance—The compositions of alloy tool steels are such that at the heat-treating temperature many of the alloy carbides remain undissolved in the

austenite. These hard carbide particles serve to increase the abrasion resistance of the steel, and this represents another function of the alloying elements. The elements commonly used for this purpose are tungsten, molybdenum, vanadium, titanium and chromium.

The Hadfield manganese steels represent another class of abrasion-resistant steels in which the function of the element is to stabilize austenite in order to produce austenitic steels which harden on cold working.

Magnetic (Electrical) Characteristics—The addition of alloying elements may greatly modify the characteristics of steel used for electrical equipment. Improved electrical characteristics for a desired application may be obtained by utilizing these effects (see Chapter 46).

One example of this function of the alloying elements is represented by the silicon steels used for transformer cores. These steels, containing silicon up to 5 per cent, have a greatly increased electrical resistivity and, as annealed, high permeability. When used as transformer cores, these properties result in greatly reduced core losses.

The magnet steels, for permanent magnets, are another example of alloy electrical steels. The outstanding property of these steels is their retentivity or ability to retain magnetism. Cobalt, chromium, and tungsten are the alloying elements commonly used to enhance this characteristic.

SECTION 2

THE AISI ALLOY STEELS

Classification and Standardization of the AISI Steels

—The alloy steels most commonly used for heat-treated parts have been classified by the American Iron and Steel Institute and the Society of Automotive Engineers. The composition ranges of these steels have been listed in Table 44-I.

These steels are identified by a numerical index system that is partially descriptive of the composition. The first digit indicates the type to which the steel belongs; thus "1" indicates a carbon steel; "2" indicates a

nickel steel; "3" indicates a nickel-chromium steel. In the case of the simple alloy steels, the second number usually indicates the percentage of the predominating alloying element. Usually the last two or three digits indicate the average carbon content in "points," or hundredths of a per cent. Thus, "2340" indicates a nickel steel of approximately 3 per cent nickel (3.25 to 3.75) and 0.40 per cent carbon (0.35 to 0.45).

The basic numerals for the various types of AISI steels (including plain-carbon steels) are:

Series Designation	Types
10xx	Nonresulphurized basic open hearth and acid bessemer carbon steel grades
11xx	Resulphurized basic open hearth and acid bessemer carbon steel grades
12xx	Rephosphorized and resulphurized basic open hearth carbon steel grades
13xx	Manganese 1.75 per cent
23xx	Nickel 3.50 per cent
25xx	Nickel 5.00 per cent
31xx	Nickel 1.25 per cent—Chromium 0.65 per cent
33xx	Nickel 3.50 per cent—Chromium 1.55 per cent
40xx	Molybdenum 0.25 per cent
41xx	Chromium 0.50 or 0.95 per cent—Molybdenum 0.12 or 0.20 per cent
43xx	Nickel 1.80 per cent—Chromium 0.50 or 0.80 per cent—Molybdenum 0.25 per cent
46xx	Nickel 1.55 or 1.80 per cent—Molybdenum 0.20 or 0.25 per cent
47xx	Nickel 1.05 per cent—Chromium 0.45 per cent—Molybdenum 0.20 per cent
48xx	Nickel 3.50 per cent—Molybdenum 0.25 per cent
50xx	Chromium 0.28 or 0.40 per cent
51xx	Chromium 0.80, 0.90, 0.95, 1.00 or 1.05 per cent
5xxxx	Carbon 1.00 per cent—Chromium 0.50, 1.00 or 1.45 per cent
61xx	Chromium 0.80 or 0.95 per cent—Vanadium 0.10 per cent or 0.15 per cent min.
86xx	Nickel 0.55 per cent—Chromium 0.50 or 0.65 per cent—Molybdenum 0.20 per cent
87xx	Nickel 0.55 per cent—Chromium 0.50 per cent—Molybdenum 0.25 per cent
92xx	Manganese 0.85 per cent—Silicon 2.00 per cent
93xx	Nickel 3.25 per cent—Chromium 1.20 per cent—Molybdenum 0.12 per cent
98xx	Nickel 1.00 per cent—Chromium 0.80 per cent—Molybdenum 0.25 per cent
TS	denotes Tentative Standard Steel, as in TS 4130 and others.
B	denotes Boron Steel, as in 46B12 and others.
BV	denotes Boron Vanadium Steel, as in TS 43BV12 and TS 43BV14.

Needless to say, this list, representing as it does, a standardization and simplification of thousands of alloy-steel compositions, is a very valuable aid to the specification and choice of alloy steels for various applications. Many of these steels were developed for specific applications, and their continued satisfactory performance has resulted in a considerable degree of standardization of application among these compositions.

Applications of the AISI Alloy Steels—The low-carbon steels (0.10-0.25 per cent carbon) in this classification are designated as carburizing steels and they are applied almost exclusively to carburized parts. However, the choice of a steel within this group is determined largely by the core properties desired for the specific application. The lower alloy combinations, such as 4023, 4118 or 5015 are used where somewhat better core properties than those obtainable with the plain carbon compositions such as C1018 or C1117 are desired. They have the further advantage of being hardenable in oil in moderate

sections, and therefore can be heat treated with less distortion than the types requiring water quenching. The higher manganese and sulphur steels are used where superior machinability is required. Typical applications of these low-alloy carburizing grades would be for the production of cam shafts, wrist pins, clutch fingers, and other automotive parts in which high strength and optimum core properties are not required.

The higher alloy carburizing steels, such as 3120, 3310, 4320, 4620, 4815, 5120, 6120, 8620, 9310 or 94B17 are used where superior case hardness or core properties are desired. The choice of steels within this group is determined primarily by the hardenability necessary to obtain the desired core properties under the given conditions of section size and heat treatment. The low nickel-chromium (3120), nickel-molybdenum (4620), plain chromium (5120), chromium-vanadium (6120) or low nickel-chromium-molybdenum (8620) steels are customarily used for such applications as automotive gears, universal joints, small hand tools, piston pins, and similar parts of moderate section for relatively severe service. The higher alloy steels 3310, 4815, 9310 or 94B17 are used for severe service applications or heavy sections. Typical applications of these steels are aircraft-engine parts, truck transmissions and differentials, rotary rock-bit cutters and large antifriction bearings.

Similarly, the choice of the higher carbon alloy steels is based largely on the hardenability requirements of the specific applications. This will, of course, be a function of the heat treatment and section size. Intricate sections or higher carbon materials (over 0.40 per cent carbon) which must be oil quenched to prevent danger of quench cracking may frequently require higher alloy compositions than the simpler sections or low-carbon materials which can be safely heat treated under more drastic quenching conditions.

As with the carburizing steels, the lower alloy higher carbon steels, such as the manganese (1330-45), plain molybdenum (4037-47), plain chromium (5130-50), or the low nickel-chromium-molybdenum (8630-50), are used for applications involving relatively small sections, but which are subject to severe service conditions, or in larger sections which may not necessitate optimum properties, but in which advantage is taken of the weight saving derived from the higher strength of the alloy steels. Typical applications are the use of the manganese steels for automotive axles and high-strength bolts, molybdenum steels and chromium steels for automotive steering parts and low nickel-chromium-molybdenum steels for small machinery axles and shafts. These lower alloy steels are also widely used for high-quality small tools.

The higher alloy AISI steels, such as the 4137-50, 4337-40, 9840-50 or 86B45 compositions are used for heavy sections or for parts subject to particularly severe service conditions or for which very mild quenches must be used to prevent distortion. Typical uses would be for relatively heavy aircraft or truck parts or for ordnance materials.

In addition to the more or less general uses described above, some of the AISI steels have quite specialized applications. Thus the 52100 steels are used almost exclusively for ball- and roller-bearing applications, and the chromium steels (5155 and 5160) were developed for and are used almost entirely for spring-steel applications.

Hardenabilities of the AISI Alloy Steels—Hardenability has been stressed as the most important function of the alloying elements in these steels, and the above discussion of their applications has shown that the choice of the alloy steel to be used for a given applica-

FIG. 44-15. Typical hardenability band (4140H steel).

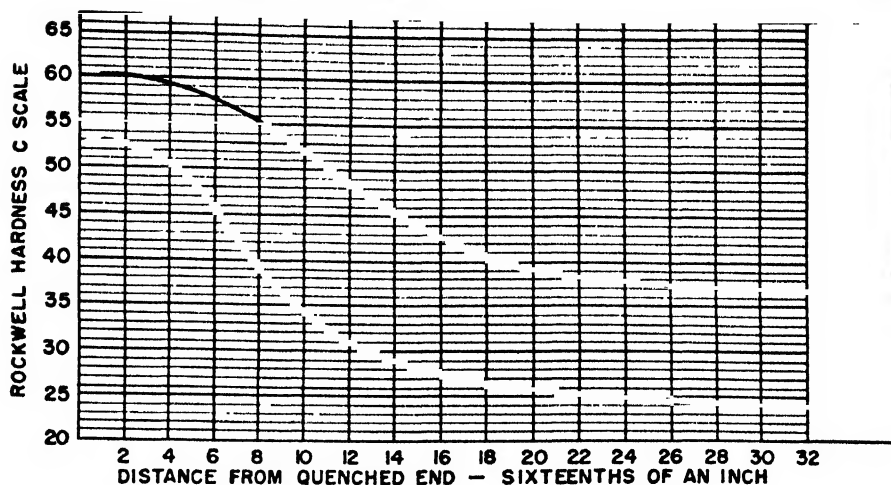


FIG. 44-16. Minimum hardenability limits for 4140H, 4340H, 5140H, 8640H and 9840H, comparing minimum hardenabilities for steels of different alloy content but of the same carbon content.

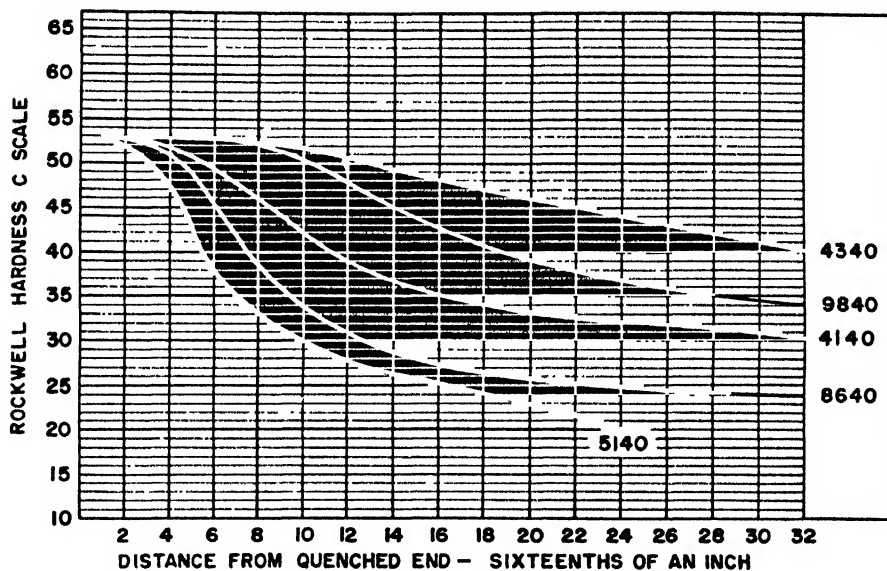
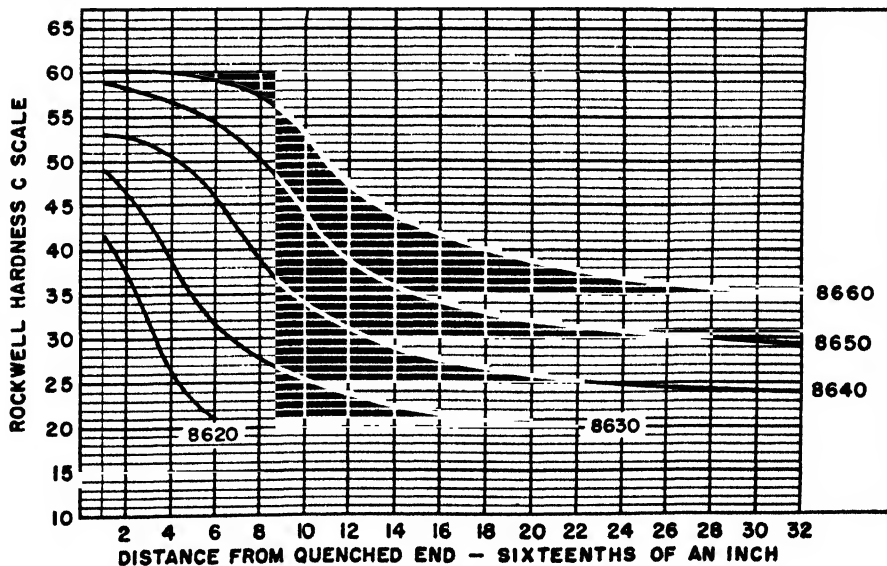


FIG. 44-17. Minimum hardenability limits of a group of steels of the 8600 (nickel-chromium-molybdenum) series, showing the increase in minimum hardenability with increasing carbon content.



tion is largely based on its hardenability. Realizing the importance of hardenability, the American Iron and Steel Institute, together with the American Society of Automotive Engineers, have established minimum and maximum end-quench hardenability curves, known as hardenability bands, for most of these alloy steels. These bands or hardenability limits, typified by Figure 44-15, are based on the analysis of data collected from hundreds of heats of each grade of steel. Such information permits these steels to be sold on the basis of this most important property, hardenability, and steels sold to such

a hardenability specification are known as "H" steels.

Since, for application purposes, the minimum hardenability values for a given steel are usually the most pertinent, the minimum hardenability limits for numerous "H" steels are shown in Figures 44-16 and 44-17. Figure 44-16 also compares the minimum hardenability limits for a group of steels of different alloy content but having essentially the same carbon content. Figure 44-17 shows the effect on the minimum hardenability limit of increasing carbon content in steels of the 8600 (nickel-chromium-molybdenum) series.

SECTION 3

ALLOY TOOL STEELS

Compositions and Applications—The principal functions of the alloying elements in tool steels are to increase hardenability, to form hard, wear-resisting alloy carbides, and to increase resistance to softening on tempering. The alloy tool steels may be roughly classified according to the extent of their utilization of these three functions. On this basis, the three classes would be as follows:

1. **Relatively low-alloy tool steels.** These are of higher hardenability than the plain carbon tool steels in order that they may be hardened in heavier sec-

tions or with less drastic quenches and thereby less distortion.

2. **Intermediate alloy tool steels.** These steels usually contain elements such as tungsten, molybdenum or vanadium, which form hard, wear-resisting carbides. They are employed in the manufacture of fast-finishing tools in which the retention of a smooth cutting edge is of particular importance.
3. **High-speed tool steels.** These contain large amounts of the carbide-forming elements which serve not only to furnish wear-resisting carbides but also to

Table 44—II. Compositions of Some Tool Steels (Per Cent)

Class	Type	C	Mn	Si	Ni	Cr	Mo	W	V	Co
1	Non-Deforming ...	0.90	1.60
1	Chromium	0.90-1.10	0.25	0.25	1.0-2.0
1	Chromium-Vanadium	0.50-1.10	0.25	0.25	0.75-1.50	0.20
1	Chromium-Nickel .	0.40-0.75	0.40	0.25	1.0-2.0	0.75-2.25
1	Chromium-Molybdenum ...	0.40-0.75	0.40	0.25	0.75-1.50	0.20-0.50
1	Nickel-Chromium-Molybdenum ...	0.40-0.75	0.40	0.25	1.0-2.0	0.75-1.50	0.20-0.50
2	Tungsten Finishing.	1.35	0.25	0.25	3.0-4.0
2	Tungsten Finishing.	1.35	0.25	0.25	0.50-1.00	3.0-4.0
2	Tungsten Finishing.	1.35	0.25	0.25	0.50-1.00	3.0-4.0
2	Tungsten Chisel ...	0.50	0.25	0.25	1.50	2.10	0.25
3	18-4-1 High Speed .	0.50-0.80	0.25	0.25	4.00	18.00	1.00
3	18-4-3 High Speed .	0.95	0.25	0.25	4.00	18.00	3.00
3	Tungsten Cobalt ...	0.75	0.25	0.25	4.00	18.00	1.00	5.00
3	Molybdenum High Speed	0.75	0.25	0.25	4.00	8.50	1.50	1.00
3	Tungsten-Molybdenum High Speed	0.80	0.25	0.25	4.00	4.75	5.75	1.50

promote secondary hardening and thereby to increase resistance to softening at elevated temperature.

Typical compositions of these three classes of tool steels are given in Table 44—II. More detailed information on types of tool steels, their composition, treatments and applications, may be found in the 1948 Edition of the American Society for Metals' Handbook, pages 653-676, and in "Tool Steels" by Gill, Rose, Roberts, Johnston and George, published by the American Society for Metals.

Table 44—II is illustrative of the types of compositions used for alloy tool steels, of which there are hundreds of variations. The uses of tool steels are so diversified and the compositions and types that may be used for a single application overlap to such an extent that a satisfactory general classification on this basis is impossible. However, the high-speed steels obviously are used for application requiring long life at relatively high operating temperatures such as for heavy cuts or high-speed machining, the intermediate alloy types are used for finishing operations in which extreme wear resistance and the ability to retain a smooth cutting edge on light cuts is necessary, and the first class of steels are general purpose tool steels, the choice of which is based primarily on section size, permissible distortion, intricacy of design, and the hardness and toughness requirements of the application; all of which are, to a considerable extent, functions of the hardenability. The higher hardenability steels are used in cases where a low "movement" (change of dimension) in hardening is required, since relatively slow oil or even air quenches may be used. These steels are also designed to be capable of hardening from relatively low quenching temperatures which also tends to decrease distortion and danger of quench cracking. Within this class, the higher carbon steels are used for applications requiring high resistance to wear or abrasion and the lower carbon steels for applications in which resistance to shock or impact is of particular importance.

HEAT TREATMENT OF ALLOY TOOL STEELS

General—The general principles of heat treatment, as described in Chapter 42, naturally apply to the heat treatment of the alloy tool steels. The alloy tool steels are generally high carbon, and many of them are relatively high-alloy steels so that their heat treatment necessarily involves special precautions to avoid distortion, cracking, and decarburization. The heating operation must be conducted at a slow rate to minimize thermal stresses, and relatively low austenitizing temperatures are usual to minimize distortion and cracking. It is a common practice to carry out the heating in two stages: a preliminary preheat to an intermediate temperature preceding the heating to the final temperatures. Decarburization is usually particularly harmful in tool steels, and the practices mentioned in the chapter on heat treatment, such as use of controlled atmospheres, packing in cast-iron chips, or heating in neutral liquid baths, are very commonly employed.

Because of their sensitivity to cracking and the dangers of distortion, relatively mild quenches are commonly used in the heat treatment of alloy tool steels, and many of them are of high enough hardenability to permit air quenching.

Since the residual stresses are high in these high-carbon steels after quenching, the stress-relieving function of the tempering operation is of particular importance. Tempering to relieve these stresses and

toughen the steels is therefore an essential part of the heat-treating operation and should immediately follow the quench. Since high hardness is usually desired, the tempering temperatures are generally low (250° to 450° F), although in cases such as some of the die applications, in which resistance to shock and impact are important, higher temperatures may be used. Tempering in the temperature range of 500° to 600° F should generally be avoided.

High-Speed Steel—As mentioned above, the high-speed steels, typified by the 18 per cent tungsten, 4 per cent chromium, 1 per cent vanadium composition, differ from the lower alloy tool steels, not only in the presence of higher percentages of carbide-forming elements, but also in the fact that the secondary hardening effects of these elements are much more fully realized and a high resistance to softening at elevated temperature is obtained. These steels require a special heat treatment in order that their unique properties may be fully realized. In outline, this procedure consists of heating to a high temperature (2150° to 2400° F) to obtain solution of a substantial percentage of the alloy carbides, quenching to room temperature, at which stage a considerable amount of austenite is retained, tempering at 1000° to 1150° F, and again cooling to room temperature. During the tempering operation, alloy carbides are precipitated, resulting in a marked secondary hardening and in a reduction of alloy content in the retained austenite, which then transforms to martensite on cooling to room temperature and results in a still greater hardness increase. It is often desirable to temper a second time to temper the martensite formed on cooling from the original tempering.

In order to prevent excessive grain growth and decarburization, the steels are held at the high quenching temperature for only a few minutes before quenching. Steels are customarily preheated to between 1400° and 1600° F before transferring to the high-heat furnace. This serves the dual purpose of eliminating the severe thermal shock of placing the cold tool into the high-temperature furnace, and of decreasing the decarburization because of the shorter time of exposure to the high temperature. The use of controlled-atmosphere furnaces, or of neutral liquid baths for the high-heat treatment is also very desirable, and in many cases essential, to minimize decarburization. The time in the high-heat furnace will, of course, vary with the heating rate and size and shape of the piece. Typical hardening temperatures are 2250° to 2350° F for the tungsten types, 2150° to 2250° F for the molybdenum types, and 2325° to 2400° F for the cobalt types.

Quenching may be in air, oil, or in liquid baths. Air cooling has the disadvantage of the formation of a tightly adherent scale during cooling. Oil quenching, while it facilitates the removal of this scale, results in higher stresses. These may be minimized, however, by removing the piece from the oil at the flash point and then air cooling. By this method, the tools will be air cooled through the temperature range in which transformation to martensite occurs. The third method, which is very commonly practiced to insure low quenching stresses, is to quench into a liquid bath at 1000° F, hold until equalized, and then slowly air cool to between 200° and 300° F and immediately temper. This procedure is particularly applicable to the hardening without undue distortion or cracking of intricate tools.

Tempering may be carried out in salt baths, lead baths, or circulating-air furnaces. The latter are particularly desirable because of their adaptability to close control of temperature and uniformity of heating the work. The rapid heating of high-speed tool steels im-

mersed in lead baths is undesirable since it may set up stresses which would lead to cracking. The maximum hardness will usually be developed at temperatures of 1000° to 1100° F, and the holding times in this temperature range usually are from 1 to 4 hours. The specific time and temperature will vary with the hardness and toughness desired.

As mentioned above, a second tempering at a relatively low temperature (600° to 650° F) will temper the martensite formed in cooling from the first tempering operation and will serve to increase the toughness of these steels without causing appreciable softening.

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Chapter 45

HIGH-STRENGTH LOW-ALLOY STEELS

The steels which are the subject of discussion in this chapter are categorized upon the basis of their mechanical properties. This is in contrast to the usual method of classification, which is generally based on chemical composition; for example, the designations "plain carbon steels," "alloy steels" and "stainless steels" are based upon the presence (or absence) of added alloying elements in the particular steel. Thus steels of a particular range of composition may be specified so that the purchaser will be able to obtain certain desired properties in a part that may be heat treated by him. On occasion, plain carbon, alloy and stainless steels are furnished to specifications involving a prescribed combination of mechanical properties; since, however, chemical composition is the primary controlling factor in obtaining these required properties, such steels automatically fall into one of the above compositional classifications.

The steels classified as "high-strength low-alloy steels" have strengths somewhat greater than those exhibited by the structural carbon steels and, since enhanced mechanical properties can be obtained with a number of different combinations of alloy contents (other than carbon), this group of steels consists of various chemical compositions, but each steel in the group meets essentially the same minimum mechanical-property requirements. For the sake of brevity, they are referred to in the text only as high-strength steels.

Engineering Function of the High-Strength Steels—Modern trends toward larger and larger structures have made design and construction engineers aware of the advantages to be gained from a steel inherently stronger than structural carbon steel in order that excessively large sections might be avoided. It was realized that the advantages would be greatest in the transportation field since thinner sections with increased strength could be employed to eliminate the hauling of excess dead weight. Since carbon and manganese are the main strength-producing elements in structural carbon steel, it is obvious that the desired increase in strength could be obtained simply by increasing the amounts of either or both of these elements in the steel. It has been found, however, that while these additions will produce the higher strength, the resulting steels often cannot be handled so readily by ordinary methods of fabrication (including welding) and even when successfully fabricated, the utility of the finished part may be seriously impaired. It was found that other alloying elements, such as copper, phosphorus, nickel, chromium, vanadium and silicon, could be added in such amounts that, with a lower carbon content, the desired increase in strength was obtained, at relatively low cost, without greatly decreasing the ductility of the steel. The result of this experience has been the development of a group of steels which possess the required high strengths and which can still be fabricated by standard shop processes on conventional equipment.

The use of thinner sections to eliminate excess weight, and the desire for longer life with decreased mainte-

nance emphasized the need for improved resistance to corrosion. Materials which are exposed to corrosive environments, of which the atmosphere is one, are subject to loss of thickness and, consequently, to a decrease in their capacity to carry load. It may readily be seen that, if the high-strength low-alloy steels corroded at the same rate as structural carbon steel, thinner sections of high-strength steel would be reduced to the permissible minimum thicknesses in less time than would the thicker structural carbon steel sections. It is, therefore, necessary that the improvement in the atmospheric-corrosion resistance of the high-strength steels at least be in proportion to the increase in their strength so as to obtain satisfactory performance from the thinner sections. Similarly, increased resistance to corrosion is required to improve the service life of equipment made from sections of standard thicknesses.

As the foregoing discussion intimates, the high-strength low-alloy steels are intended for general structural applications. They should not, therefore, be considered as special-purpose steels nor as steels requiring, or adapted for, heat treatment.

Historical Background—Before 1900, the only widely available ferrous construction material, such as used in buildings and bridges, was mild carbon steel having an ultimate tensile strength of about 60,000 pounds per square inch. In 1902, the design engineers of the Queensboro Bridge, which was to span the East River in New York City, requested the development of a stronger steel so that the number and size of supporting members could be held within the space and weight limitations of their design. Carnegie Steel Company, which now is part of the United States Steel Corporation, supplied 3.25 per cent nickel steel for this application. Because of the satisfactory results obtained on the Queensboro Bridge, nickel steel was used for the stiffening trusses of the Manhattan Bridge in 1906. Although this material was satisfactory for riveted structures from the standpoint of strength, it was an expensive material, economical for use only in structures in which reductions in the weight or size of members was a necessity.

Silicon steel containing about 1.00 per cent silicon and 0.25 per cent carbon, cheaper than the nickel steel, was suggested for ship construction as early as the 1880's. However, the first application of this steel in ship-hull construction was in 1907 as plates in the S. S. MAURETANIA. The manufacture of this steel was ultimately discontinued because of the many difficulties encountered. Later on, a so-called "silicon structural steel" was used which, however, contained only a nominal amount of silicon. This material depended for its strength mostly on its high carbon content, usually 0.30 per cent minimum. It was first used in 1915 in a bridge spanning the Ohio River at Metropolis, Illinois. This grade, under the misnomer Silicon Structural Steel, has been one of the most widely used materials for riveted structures, but it is rapidly being superseded by carbon structural steel and high-strength steel.

In 1927, as an alternate for expensive nickel steel, the American Bridge Company, now a division of United States Steel, used a 1.60 per cent manganese steel for the lower chord members of the Kill van Kull Bridge connecting Staten Island with the mainland at Bayonne, New Jersey.

Similar developments had taken place in other countries. Engineers in Great Britain attempting through weight reduction to effect economies in ocean freights and handling charges, had used carbon steels containing generous amounts of silicon and manganese. Engineers on the Continent made use of similar steels, and early in 1933 the Germans completed the construction of "The Flying Dutchman," the first light-weight "streamlined" train built of high-strength steel. This composition contained additions of silicon, manganese and copper and was known as ST. 52.

Two New Problems: Corrosion and Welding—Each of the high-strength steels which had been developed before the year 1930 was found to lack one or more of the qualities required for most constructional applications. The potential benefits to be derived from the use of such materials, however, were indicated by these early steels, and the experience gained in their use had pointed out the specific characteristics needed in steels for this field of application. It had been discovered that improved resistance to corrosion was required if section thicknesses were to be reduced without the sacrifice of serviceability, or if increased service life were to be obtained. This discovery led to the addition of elements for the express purpose of improving the atmospheric corrosion resistance. A great deal of corrosion testing followed, which in turn led to the evaluation of the effects of the individual elements and of combinations of the elements on resistance to corrosion.

At about the same time, it was realized that good weldability of the high-strength steels was required to permit the greatest possible weight reduction, and to thus enhance the economical use of this type of material. The development of the present day high-strength low-alloy steels has paralleled the adoption of the various welding processes and much metallurgical ingenuity has been shown during the development of these steels to make certain that they possessed good welding characteristics.

The earliest of the present-day high-strength steel brands was "COR-TEN," which was introduced by United States Steel in 1933. This brand of steel has been continuously improved to meet the increasingly severe demands of fabricators, an experience common with most new steels. The great demand for steels of this type is indicated by the fact that, since "COR-TEN" steel was introduced, many other steel-producing companies have developed their own grades and there are at present a considerable number of high-strength low-alloy steels. These steels have found wide acceptance in the metal-working industry and have come into general production and almost universal use for many applications.

Definition—This group of steels has been termed "High Tensile Steels" and "Low Alloy Steels" but the name "High-Strength Low-Alloy Steels" is gradually being adopted as its designation.

There have been numerous attempts to write suitable definitions for this class of steels and the following is one that has been adopted by the American Iron and Steel Institute for its product manual on this grade of steel:

"High strength low alloy steel comprises a specific group of steels with chemical compositions specially developed to impart higher mechanical-property values

and greater resistance to atmospheric corrosion than are obtainable from conventional carbon structural steels containing copper. High strength low alloy steel is generally produced to mechanical property requirements rather than to chemical composition limits.

"High strength low alloy steel is generally intended for application where savings in weight can be effected by reason of its greater strength and atmospheric corrosion resistance and where better durability is desired."

These steels are normally furnished in the hot-rolled (as rolled) or cold-rolled, annealed or normalized condition, and are intended for use without further heat treatment, particularly not by quenching and tempering.

These steels are readily adaptable to fabrication by forming, riveting and welding; in the latter case no preheat or postheat is required. In certain complex structures, stress relieving after welding may be desirable.

Fundamental Characteristics—Steels of the high-strength low-alloy type, to be of interest as construction materials, must have characteristics and properties which result in economies to the user when the steels are properly applied. Such materials should be considerably stronger and, in many instances, tougher than structural carbon steel. They must have sufficient ductility, formability and weldability to be successfully fabricated by customary shop methods. In addition, high-strength steel should have sufficient resistance to corrosion so that equal life in a thinner section or longer life in the same section obtains when compared to that of a structural carbon steel member. Table 45—I lists the combination of important characteristics which an all-purpose high-strength steel should possess, and indicates the methods by which these characteristics are commonly determined.

Table 45—I. Important Characteristics of an All-Purpose High-Strength Steel

Property	Method of Determination
High strength ..	.Yield point in tension test.
Good corrosion resistanceAtmospheric-corrosion resistance judged from weight loss in exposure rack test, and useful life judged from service performance.
Good formability.	.Bend test, tensile elongation and fabrication performance.
Good weldability.	.Weldment performance judged by various weld-bending tests.
Good toughness under adverse conditionsTemperature of transition from tough to brittle behavior in notched-bar impact test.
Good resistance to repeated loadingEndurance tests on simple specimens or on built-up structures.
Good abrasion resistance ..	.Service performance.

Since the yield point of the constructional material determines the stress to which a structure may be subjected without permanent deformation, the unit working stresses of each structure are based upon this important property. The minimum yield point of structural carbon steel is 33,000 lb. per sq. in.; that of high-strength steel is generally 50,000 lb. per sq. in. Hence, on the basis of the proportionality of their yield points, the unit working stress employed with high-strength steel may

be increased to one and one-half times that used with structural carbon steel. The use of higher unit working stresses permits reductions in the thicknesses of section in the structure and this results in a decrease in weight. This saving in weight is of utmost importance in mobile structures, both in the economy and speed of transportation, and in the ability of these structures to carry greater payload. Frequently high-strength steels are substituted for structural carbon steel without change in section, the sole purpose being to produce a stronger and more durable structure with no increase in weight.

Corrosion Resistance—When high-strength steels are employed to save weight, it is desirable to make the maximum reduction in section. The engineer, however, must consider the structure not only at the time it is built but also as it will exist perhaps twenty years later when the members have been thinned by corrosion. Since the structural members are already relatively thin by virtue of the desired weight-saving, it is necessary that further thinning from the ravages of corrosion be prevented, at least in so far as possible, else the structure will become too weak to serve its intended purpose. Similarly, when high-strength steels are employed to obtain increased service life, corrosive attack must be retarded. Thus, it may be said that improved corrosion resistance is equal in importance to higher strength and that the corrosion resistance of the high-strength steels should be as high as is economically feasible without, at the same time, adversely affecting the other properties and characteristics to a marked extent. Unfortunately, the corrosion resistance of a material cannot be expressed quantitatively (as the yield point is described, for instance) because it is only a relative term. Furthermore, no material is equally resistant to all the corrosive conditions to which it might conceivably be exposed. Its performance can only be compared with that of other materials under similar conditions. A large number of corrosion tests under many conditions of exposure, particularly atmospheric rack tests, have been conducted for the purpose of determining these relative values. Such tests differentiate between the performances of various steels and permit a close evaluation of the trends in weight loss due to corrosion. The atmospheric-corrosion resistance of high-strength steels varies with the combination and content of those alloying elements most effective in building up this resistance. Several steels of this type possess four to six times the atmospheric-corrosion resistance of structural carbon

steel having a low copper content. The superior atmospheric corrosion resistance which most of the high-strength steels have shown in rack tests has been confirmed by their performances in many different kinds of service. This superiority is particularly evident in those applications in which the materials are subjected principally to atmospheric corrosion. Much has been written concerning the details and results of corrosion tests and a number of references are included in the bibliography for the use of those who wish to pursue the subject further. Figure 45-1 shows time-corrosion curves for plain carbon steel, structural copper steel, and three high-strength, low-alloy steels exposed to an industrial atmosphere.

Formability—High-strength, low-alloy steels must have suitable plastic properties so that they can be hot or cold worked readily and economically into various commodities for use in engineering structures. These materials must also be capable of being readily hot formed, cold formed, sheared, punched, reamed, coped, sawed, milled, drilled, riveted and welded by the fabricator. These operations can be performed on high-strength steels with almost as much ease as on regular structural carbon steel. High-strength steels, despite their high yield points, can be satisfactorily worked in pressbrakes, drawbenches, presses and other equipment used for cold forming, even when these forming operations are quite severe.

There are some inherent differences between the cold-forming characteristics of high-strength and plain carbon steels. First of all, more force is required to produce a given amount of permanent set in a high-strength steel section than in a plain carbon steel section of the same thickness. This is attributable to the greater strength of the high-strength steel, and the difference in force is approximately proportional to the difference in the yield points of the two materials.

Another difference is in the amount of springback encountered after forming the two materials. When like parts are formed from the same thickness of a high-strength steel and of a plain carbon steel, the relative springback of the two materials will be about proportional to the amount of force required to produce the part; therefore, a somewhat greater allowance for springback should be provided when forming the high-strength steel. When the high-strength steels are employed in thinner sections, the effect of reductions in thickness upon the springback allowance must also be

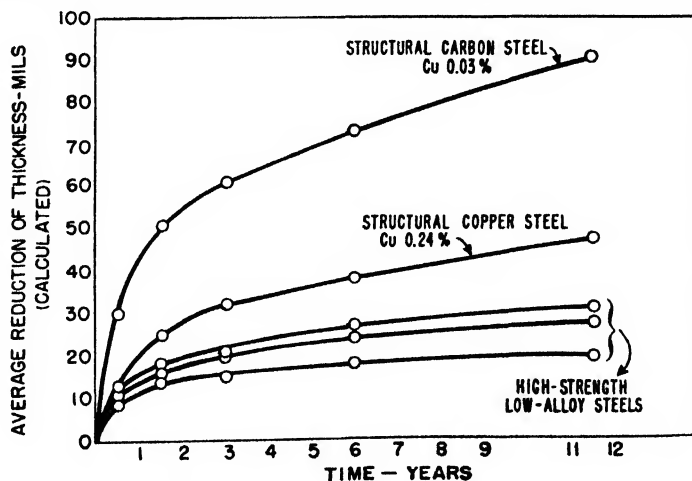


FIG. 45-1. Comparative reductions of thickness by atmospheric corrosion of steels exposed to an industrial atmosphere.

considered. In bending operations, the amount of elastic deformation is increased considerably by reductions in section thickness. Therefore, when lighter sections of high-strength steel are used, more springback is obtained even though the force to produce the part is little or no greater than that required for the original part of plain carbon steel.

Experience has shown that more liberal radii of bend must be used with a high-strength steel than with plain carbon steel for successful cold forming. This again is ascribed to the greater strength of the high-strength steel.

Weldability—Since welding is generally employed in fabricating structural steel, it is important that high-strength steels be readily weldable by metal-arc welding and by gas welding in plate thicknesses and by all the resistance welding processes in sheet and strip thicknesses. It is equally important that the welds in fabricated structures have the required strength and ductility to withstand the most adverse conditions in the contemplated service. As mentioned before, the development of the present-day high-strength steels has paralleled the adoption of the various welding processes and particular care was exercised to make certain that these steels possessed good welding characteristics. Suffice it to say that the high-strength low-alloy steels are considered readily weldable by the usual processes.

Notch toughness, measured in a notched-bar impact test, is of interest because it reflects the behavior of the materials at notches in actual structures. High-strength steels exhibit notch toughness superior to that of structural carbon steels, whether the notch toughness be considered in terms of the amount of energy absorbed in breaking a specimen at room temperature or in terms of the refrigerated temperature to which they preserve their toughness.

These advantages in notch toughness are reflected in superior performance of actual structures under adverse conditions of stress.

Fatigue Resistance—The resistance to repeated loading, or the fatigue resistance, of materials is generally determined by testing polished specimens. The endurance limit determined by this method of testing, therefore, is considered to have little bearing on the fatigue resistance of full-size structures since failures of structural members, subject to alternating or pulsating stresses, generally originate at some surface notch or discontinuity acting as a stress raiser. Laboratory tests of polished specimens do, however, indicate that the high-strength steels are superior to structural carbon steels. The ratio of endurance limit to tensile strength is greater for high-strength steels than for structural carbon steel.

Abrasion Resistance—There is general agreement that the resistance of various steels to abrasive action increases with strength or hardness and, to some extent, with carbon content. Service tests have demonstrated that the abrasion resistance of the high-strength steels, with their inherently greater strength, is somewhat higher than that of structural steel containing 0.15 to 0.20 per cent carbon.

Composition, Properties and Characteristics, and Applications—In the development of a chemical composition to obtain the desired properties in a steel, it was, of course, imperative that strength be given first consideration. Since increased strength can be obtained with various combinations of alloying elements, a number of different compositions have been produced which offer interesting combinations of other properties and characteristics in addition to the required minimum strength. This is well illustrated by the compositions of a representative group of current high-strength steels

Table 45—II. Composition Ranges of Some Representative High-Strength Low-Alloy Steels.

Brand (and Type)	Composition, Per Cent								Other Elements
	C	Mn	P	S (max.)	Si	Cu	Ni	Cr	
COR-TEN (Cr-Si-Ni-Cu-P)	0.12 max.	0.20/0.50	0.07/0.15	0.05	0.25/0.75	0.25/0.55	0.65 max.	0.30/1.25	—
TRI-TEN "E" (Mn-Cu-V)	0.22 max.	1.25 max.	0.04 max.	0.05	0.30 max.	0.20 min.	—	—	V, 0.02 min.
TRI-TEN (Mn-Ni-Cu)	0.25 max.	1.35 max.	0.045 max.	0.05	0.30 max.	0.30/0.60	0.50/0.90	—	—
MAN-TEN (Mn-Cu)	0.25 max.	1.10/1.60	0.045 max.	0.05	0.30 max.	0.20 min.	—	—	—
Aldecor (Si-Cu-Mo-P)	0.12 max.	0.15/0.40	0.08/0.15	0.05	0.35/0.75	0.35/0.60	—	—	Mo, 0.16/0.28
Republic Double Strength (Ni-Cu-Mo)	0.12 max.	0.50/1.00	0.04 max.	0.05	—	0.50/1.00	0.50/1.10	—	Mo, 0.10 min.
Jalten No. 2 (Mn-Cu-P)	0.15 max.	0.90/1.40	0.08/0.14	0.05	0.10 max.	0.30 min.	Residual	Residual	—
Hi-Steel (Cu-Ni-Mo-P)	0.12 max.	0.50/0.90	0.05/0.12	0.05	0.15 max.	0.95/1.30	0.45/0.75	—	Mo, 0.08/0.18 Al, 0.12/0.27
Mayari R (Cu-Ni-Cr-Si-P)	0.12 max.	0.50/1.00	0.06/0.12	0.05	0.10/0.50	0.30/0.70	0.25/0.75	0.40/1.00	—
Dynalloy (Ni-Cu-P-Mo)	0.15 max.	0.60/1.00	0.050/0.100	0.05	0.30 max.	0.30/0.60	0.40/0.70	—	Mo, 0.05/0.15
Yoloy (Ni-Cu-P)	0.15 max.	0.60 max.	0.05/0.10	0.05	—	0.75/1.25	0.50/2.00	—	—
50Y (Ni-Cu-P)	0.12 max.	0.50/0.75	0.04/0.07	0.04	0.10 max.	0.50/0.70	0.60/0.90	—	—
N-A-X High Tensile (Si-Cr-Zr)	0.08/0.15	0.50/0.75	0.04 max.	0.05	0.60/0.90	Residual	Residual	0.50/0.65	Zr, 0.05/0.15

Table 45—III. Comparative Properties and Engineering Data for Structural Carbon Steels and Typical High-Strength Steel*

Mechanical Properties in Thicknesses ½" and Under	Typical High Strength Steel	Structural Carbon Steel	
		A.S.T.M. A-7	A.S.T.M. A-113
Yield Point, (lb. per sq. in.)	50,000 Min.	33,000 Min.	27,000 Min.
Tensile Strength, (lb. per sq. in.)	70,000 Min.	60,000 to 72,000	50,000 to 62,000
% Elongation in 2"	22 Min.	—	—
% Elongation in 8", 0.180" and heavier....	18	21	24
Cold Bend	180° D = 1T	180° D = ½ T	180° Flat
Resistance to Atmospheric Corrosion (Comparative)	4 to 6	1 (or 2 with Copper 0.20% Min.)	1 (or 2 with Copper 0.20% Min.)
Modulus of Elasticity	28/30,000,000	28/30,000,000	28/30,000,000
Endurance Limit (as Rolled), (lb. per sq. in.)	42,000	28,000	26,000
Charpy Impact, Keyhole Notch, (as Rolled—Room Temp., Avg.) Ft. Lbs....	40	25	30
Coefficient of Expansion per Degree F., 70° to 200° F.	0.0000063	0.0000063	0.0000063

* ASTM Standard Specimens, minimum number of tests, and ductility modifications apply.

shown in Table 45—II, and by the comparison of the properties of a typical high-strength steel with those of two plain carbon steels given in Table 45—III.

EFFECT OF CHEMICAL COMPOSITION ON PROPERTIES AND CHARACTERISTICS

Carbon—It will be noted that a number of features are common to the compositions of many of the current high-strength low-alloy steels. For example, the carbon contents are generally lower than those found in structural carbon steels. Carbon is one of the more potent strengthening elements, but the increase in strength which it produces is accompanied by a marked decrease in ductility, notch toughness, and weldability.

In so far as metal-arc welding is concerned, it is generally desirable to keep the carbon content below certain maximum values because of the high rates of cooling which obtain in the heat-affected zone of a weldment and the consequent danger of embrittlement if the carbon content becomes too high. Carbon has two distinct effects in this connection: (1), an increase in carbon increases the hardenability and therefore increases the tendency for hard, brittle martensite to form adjacent to the weld; and (2), when martensite does form, the steel is much more brittle if the carbon content is high than if it is low. Thus, if other alloying elements are added, for example, to increase the corrosion resistance or strength, and the hardenability is consequently increased, the carbon content must be reduced to avoid embrittlement.

For satisfactory metal-arc welding of plain carbon steels with manganese contents below about 0.60 per cent, the upper limit of carbon content is about 0.30 per cent. With high-strength steels, in which hardenability is increased by addition of other elements, this upper limit is reduced; for example, to 0.12 per cent carbon in a steel for metal-arc welding containing 0.80 per cent chromium, 0.50 per cent nickel, 0.40 per cent silicon, 0.40 per cent copper and 0.10 per cent phosphorus. Of course, with less alloy content, the carbon content may be somewhat higher.

In the case of spot welding, where quenching rates are even more drastic than in metal-arc welding, the maximum carbon content in plain carbon steel is still further reduced and would, for example, be approxi-

mately 0.15 per cent. Spot welds made in high-strength steels have considerably less ductility than spot welds in plain low-carbon steels, even when the former contain only a maximum of about 0.10 per cent carbon.

The addition of carbon improves the endurance limit in direct proportion to the amount it raises the tensile strength. It is reported that variations in carbon content or conditions of heat treatment have little effect on the rates of corrosion of steels.

Manganese—The manganese contents of most of the high-strength steels, as is shown in the table, are somewhat higher than those of carbon structural steel. The effect of manganese on the mechanical properties of these types of steel is very much like that produced by carbon, but to a milder degree. The addition of manganese to steel results in an increase in strength accompanied by a decrease in ductility. There appears to be a slight improvement in notch toughness and a slight impairment in weldability with increase in manganese content.

In regard to metal-arc welding, it is desirable to keep the manganese content below certain values. These vary considerably, depending on the amounts of other elements present, particularly the carbon content. Thus, if the carbon content is about 0.25 per cent, manganese contents up to about 1.00 per cent are acceptable, whereas, if carbon is 0.20 per cent or lower, manganese somewhat over 1.00 per cent would be satisfactory. These permissible maximum amounts of carbon and manganese will be further reduced by approximately the extent to which the hardenability is increased by the addition of other alloying elements used to obtain the desired type of high-strength steel composition.

In the case of spot welding, the maximum permissible manganese content is again dependent upon the amount of carbon present. As mentioned previously, a carbon content of about 0.15 per cent can be tolerated in a steel having a manganese content below about 0.60 per cent. However, if the carbon content is reduced to 0.10 per cent or lower, the permissible manganese content may be 1.00 per cent or a little higher.

The addition of manganese, as is the case with carbon, increases the endurance limit in direct proportion to the amount it raises the tensile strength. It is reported that an increase in the manganese content has a very

small beneficial effect on the atmospheric corrosion resistance of steel.

Phosphorus—Many of the present day high-strength low-alloy steels have phosphorus contents in the range of 0.04 to 0.15 per cent. Additions of phosphorus markedly increase the strength properties of steel but this increase is accompanied by a decrease in ductility. Phosphorus was formerly considered to cause embrittlement in steels when present in amounts exceeding about 0.10 per cent. It has been found, however, that the embrittling effect of phosphorus is influenced markedly by the carbon content and that this effect is not pronounced in the low range of carbon content generally employed in the manufacture of high-strength steels. It has been further shown that grain-refining additions of aluminum improve somewhat the toughness of phosphorus-bearing steels. Thus added benefit is gained from the silicon-aluminum deoxidation practice employed in the manufacture of many high-strength steels. With respect to metal-arc welding, the effect of phosphorus is similar to that of manganese except that the influence of phosphorus is greater. Since the heat-affected parent metal is of principal concern when a steel is welded, the carbon content must be kept low enough so that the phosphorus which is present will not increase the hardenability of the steel by an amount sufficient to produce martensite. The results of several investigations indicate that a permissible upper limit of 0.20 per cent phosphorus may be set for a plain carbon steel containing 0.10 per cent carbon. These maxima will, of course, be reduced by approximately the extent to which the hardenability is increased by the addition of other alloying elements used to obtain the desired type of high-strength steel composition. The results of a few investigations to determine the effect of phosphorus content on the spot-welding characteristics of low-carbon steel indicate that cracking occurs in the welds if the phosphorus content is 0.20 per cent or higher. Addition of phosphorus, as in the case of carbon and manganese, increases the endurance limit in approximately direct proportion to the increase in the tensile strength. The atmospheric corrosion resistance of steel is considerably increased by the addition of phosphorus alone, and it has been shown that, when small amounts of copper are present in the steel, the effect is greatly enhanced. In other words, a given amount of phosphorus and copper has a beneficial effect which is greater than the effect produced by the corresponding amount of either of the individual elements alone.

Copper—Copper in limited quantities is beneficial to steels of the high-strength low-alloy type. Most present-day high-strength steels contain copper in amounts ranging from 0.20 to 1.30 per cent. Copper markedly increases the strength properties and the hardness of low- and medium-carbon steels with only a slight accompanying decrease in ductility. In contrast to the usual effect of alloying elements on strength properties, copper additions cause a more rapid rise in yield point than in tensile strength. The relative rates of increase are such that at about 3.00 per cent copper the yield point would closely approach the tensile strength. The endurance limit of steel is improved by the addition of copper, at least in proportion to the increase produced in the tensile strength. Copper up to at least 0.75 per cent is considered to have little effect on notched-bar toughness or welding performance. Steels containing over about 0.60 per cent are susceptible to precipitation hardening. This type of hardening is most pronounced in the range from 1.20 to 1.50 per cent copper, where increases of some 20,000 lb. per sq. in. in both yield point and tensile strength are possible. Steels containing

about 0.50 per cent or more of copper frequently exhibit "hot shortness" during hot working. This is evidenced by the occurrence of cracks or extremely roughened surfaces, sometimes referred to as "checking," which may develop during hot deforming at too high a temperature or during hot working after overheating. These defects are caused by preferential oxidation of the iron and by penetration of the remaining copper-rich alloy into the grain boundaries of the steel beneath the scale. The occurrence of these undesirable surface conditions can be minimized by careful control of oxidation during heating and by hot working below the melting temperature of the copper-rich solid solution. There has also been considerable experience which indicates that the addition of nickel in an amount equal to at least one-half the copper content is very beneficial to the surface quality of steels containing copper. Copper, in the concentrations used, is by far the most potent of all the common alloying elements in improving atmospheric corrosion resistance. Copper is especially effective in small amounts—as low as 0.10 per cent or less—in regular carbon steel. Continued improvement may be obtained up to 1.00 per cent copper, but the effect is not nearly so marked with additions over about 0.20 per cent copper.

Other Elements—Examination of Table 45—II discloses that other elements besides those discussed above are added to the various proprietary high-strength steels. Space does not permit a full discussion of the effect of the individual elements, but the amount of each element present in the steels listed in Table 45—II has been added to obtain a desired improvement in one or more of the essential characteristics and properties. From an examination of the compositions shown in Table 45—II and of the properties shown in Table 45—III, it will be seen that the high strength has been obtained in these steels by the use of alloying elements other than carbon. In spite of the fact that an increase in strength is usually accompanied by a decrease in toughness, the high-strength steels, having a lowered carbon content, are both stronger and tougher than structural carbon steels. Generally, some sacrifice has been made in weldability, and even occasionally in formability, in order to obtain this higher strength and increased resistance to atmospheric corrosion, along with higher endurance limit and greater abrasion resistance. The specific combinations of elements employed in the individual high-strength steels have resulted in various compromises of the properties and characteristics previously discussed.

Many articles and papers have been published which deal with the properties, characteristics and fabrication of the various steel compositions which are termed high-strength low-alloy steels, and a number of references which cover the subject in greater detail will be found at the end of this chapter.

Applications—While high-strength low-alloy steels find application in all recognized market classifications, the largest single field of application has been in the construction of transportation equipment. From 1934 to 1954 more than 190,000 railroad freight cars and 6,000 railroad passenger units have been constructed with but one of the leading grades of high-strength steel. If all of the prominent grades are considered, the number of high-strength steel freight cars in service would doubtless exceed 300,000 or 17 per cent of the current car ownership by American Class I railroads. Until quite recently, the emphasis, in railroad freight car usage, was on the substantial economies in operating costs obtained by using high-strength steels in somewhat reduced thicknesses to decrease the dead weight

of the cars. In some instances, the railroads were also able to increase the revenue per car trip by enlarging the capacity of the lightweight car so that it could carry an additional amount of pay load equal to the reduction made in the dead weight.

Similar benefits from weight reduction and some increase in capacity have been obtained with other mobile equipment, such as various types of trucks, trailers and buses.

In recent years, however, the emphasis has been shifted, in many types of applications, from weight reduction to stronger, more durable equipment, with no appreciable decrease in weight.

In addition to decreased maintenance costs, utilization of this principle has resulted in improved service performance. Examples are found in railroad freight cars, mine cars, power shovels and many other types of earth-moving equipment, off-highway trucks, and many other types of material-handling equipment, all of which are subjected to severe abuse in service.

High-strength steels, surface ground while hot and subsequently cold formed into the more elaborate types of automotive bumpers prevalently in use, have largely supplanted high- and medium-carbon steels previously used in making bumpers by hot forming which required costly hand grinding of the formed bumper before plating.

In bridges, designers are giving increased recognition to the importance of reducing dead load by using high-strength steels, particularly when long spans are involved and where a reduction of weight at the center permits additional saving in the weight of the supporting members. High-strength steels also lend themselves to economical tower construction where their properties permit the use of smaller sections than would be required in plain carbon steel. This advantage is important in tall television towers where wind resistance is lessened by use of smaller sections, and in transmission towers where lighter weight is a substantial advantage in reducing freight and handling costs.

Multistory building construction does not generally provide a suitable field of application for high-strength steel because weight reduction in the structural members is not usually important or economical. However, lightweight cold-formed sections of high-strength steel are being used to advantage in the framing members of industrial and farm buildings.

The weight of portable and semi-portable containers for liquefied petroleum gas has been reduced substantially by the use of high-strength steel, making them easier and less costly to handle and ship. Almost all such containers are now made of high-strength steel.

Other applications of high-strength steels include: the inner bottoms, floors, tanks and hatch covers of ore boats; hulls and other structural members and accessories of small tankers, barges, tugs, launches and river boats; coal bunkers, street-lighting poles, portable oil-drilling rigs, cable reels, pole-line hardware, air-conditioning equipment, oil-storage tanks, stokers, agricultural-machinery parts, military shipping containers, and air-preheater heating elements.

Chapter 46

SILICON-STEEL ELECTRICAL SHEETS

Introductory—When an alternating electric current is passed through a wire coiled around an iron bar, a magnetic field is induced in the bar, or core as it is commonly called. This magnetic field will, in turn, induce an electric current in a second wire coiled around the same bar or core, but not connected to the first wire. During this process, some of the electrical energy is converted to heat which is wastefully dissipated. If the core is constructed of thin laminations insulated from each other, the amount of energy lost as heat will be considerably reduced because the stray magnetic fields that cause part of the energy loss cannot flow as freely in a laminated structure as they do in a solid bar. An additional, and far greater, reduction in energy loss can be obtained, however, by replacing the iron in the core with silicon steel. The relatively high permeability, high electrical resistance, and low hysteresis loss of the silicon steels, which contain $\frac{1}{2}$ per cent to 5 per cent silicon, account for the beneficial reduction in energy loss.

The silicon steels (first patented by Sir Robert Hadfield about 1900) have made possible the development of more efficient and more powerful electrical equipment and have played an important role in the rapid growth of the electrical power industry. It has been estimated that this industry uses approximately 750,000 tons of silicon steel sheets annually.

In this chapter, reference is made only to steels produced in sheet form for the magnetic cores of electrical equipment and containing up to approximately 5 per cent silicon, which is the upper limit for commercial materials. Such products are referred to in the industry as electrical sheets.

CLASSIFICATION AND USES OF ELECTRICAL SHEETS

Silicon-steel electrical sheets may be divided into two general classifications, (1) grain-oriented steels, and (2) non-oriented steels. The grain-oriented steels, containing about $3\frac{1}{4}$ per cent silicon, are used in the highest-efficiency distribution and power transformers and in large turbine-generators. The non-oriented steels may be further subdivided into three classes on the basis of composition and application.

1. **Low-silicon steels** containing about $\frac{1}{2}$ per cent to $1\frac{1}{2}$ per cent silicon. These steels are used principally in the rotors and stators of motors and generators. Steels containing about 1 per cent are also used for reactors, relays, and small intermittent-duty transformers.
2. **Intermediate-silicon steels** containing about $2\frac{1}{2}$ per cent to $3\frac{1}{2}$ per cent silicon. These steels are used in motors and generators of average to high efficiency and in small to medium-sized intermittent-duty transformers, reactors, and motors.
3. **High-silicon steels** containing about $3\frac{3}{4}$ per

cent to 5 per cent silicon. This class is used in power transformers and the highest-efficiency motors, generators, and radio-transformers, and in other communications equipment.

There are a number of grades of electrical sheets in each of the above classifications. Specifically, there are three grain-oriented grades, typified by USS Transformer 80, 73, and 66, and eight non-oriented grades, USS Armature, USS Electrical (low-silicon), USS Motor, USS Dynamo (intermediate-silicon), USS Transformer 72, 65, 58, and 52 (high-silicon). The sheets are "graded" on the basis of a maximum specified core loss * at an induction of either 10 or 15 kilogausses. Specified maximum values for the grain-oriented grades have been established only at 15 kilogausses. Core loss values are expressed as watts per pound at 60 cycles and vary for each grade and thickness as shown in Table 46—I. It can be seen that the core loss of the non-oriented grades decreases (improves) as silicon content increases and thickness decreases.

PROCESSING OF ELECTRICAL SHEETS

Grain-Oriented—These grades are made from open hearth steel containing about $3\frac{1}{4}$ per cent silicon. The selection of the charge for the heats, and the melting and refining are under very close controls to insure the cleanest and purest steel possible. Otherwise, the furnace practice is similar to that used for other low-carbon steels. The alloying addition of silicon in the form of ferrosilicon is made to the steel in the ladle. Pouring practices are much like those used for other fully-killed steels and the ingots are rolled to slabs by common practices.

The slabs are next hot rolled and coiled in continuous hot strip mills. After rolling, the coils are pickled in continuous, single-strand picklers to remove the surface oxides formed during hot rolling. Subsequent steps in the processing may vary somewhat among the various manufacturers, but generally they consist of two cold reductions, each followed by a suitable continuous anneal, and a final high-temperature box anneal. The steel may be used directly from the coil after box annealing for wound-core distribution transformers in spite of slight curvature ("coil-set") caused by annealing in coil form. If the steel is to be used for laminations in power transformers or turbine-generators, where even this slight curvature is undesirable, the coil-set must be removed by continuous annealing.

Non-Oriented—Two general processing methods are used in the production of non-oriented electrical sheets. The older method consists of hot rolling the sheets on hand mills; it is being supplanted by the newer continuous cold-reduction method. The inherent brittleness

*Definitions for core loss and other magnetic terms are given at the end of this chapter.

Table 46—I. Maximum Core Losses for Electrical Sheets, Based on Epstein Test, A.S.T.M. Standard Method A-34.
(Watts per Pound at 60 Cycles and 10 Kilogausses)

Grade	Electrical Sheet Gage No.	29	28	27	26	25	24	23	22
	Gage Thickness, In.	0.0140	0.0155	0.0170	0.0185	0.0220	0.0250	0.0280	0.0310
USS Armature .AISI M-43		1.30	1.38	1.46	1.55	1.75	1.98	2.23	2.50
USS Electrical .AISI M-36		1.17	1.23	1.29	1.35	1.50	1.70	1.94	2.17
USS MotorAISI M-27		1.01	1.05	1.09	1.14	1.22	1.30	1.44	1.60
USS Dynamo ..AISI M-22		0.82	0.86	0.90	0.94	1.02	1.10	—	—
USS Transformer 72									
..... AISI M-19		0.72	0.76	0.80	0.83	0.90	0.97	—	—
USS Transformer 65									
..... AISI M-17		0.65	0.68	0.72	0.75	—	—	—	—
USS Transformer 58									
..... AISI M-15		0.58	0.61	0.65	0.68	—	—	—	—
USS Transformer 52									
..... AISI M-14		0.52	—	—	—	—	—	—	—

(Watts per Pound at 60 Cycles and 15 Kilogausses)

USS Armature .AISI M-43	3.90	4.00	4.10	4.20	4.70	5.25	5.80	6.40
USS Electrical .AISI M-36	3.30	3.40	3.50	3.60	3.95	4.40	4.95	5.50
USS MotorAISI M-27	2.46	2.55	2.63	2.74	2.92	3.09	3.41	3.78
USS Dynamo ..AISI M-22	2.00	2.09	2.18	2.26	2.45	2.63	—	—
USS Transformer 72								
..... AISI M-19	1.78	1.87	1.96	2.03	2.19	2.35	—	—
USS Transformer 65								
..... AISI M-17	1.62	1.69	1.79	1.86	—	—	—	—
USS Transformer 58								
..... AISI M-15	1.46	1.54	1.63	1.71	—	—	—	—
USS Transformer 52								
..... AISI M-14	1.33	—	—	—	—	—	—	—
USS Transformer 80								
..... AISI M-8	0.80	—	—	—	—	—	—	—
USS Transformer 73								
..... AISI M-7	0.73	—	—	—	—	—	—	—
USS Transformer 66								
..... AISI M-6	0.66	—	—	—	—	—	—	—

of silicon steel at room temperature, however, normally limits the grades that can be cold reduced to those containing less than 3½ per cent silicon. Thus, it appears that production of high-silicon sheets in the hand mills will be continued until such an operation becomes uneconomical, or these grades can be made by the cold-reduction method.

The same grades of steel, with the exception noted above and also with some minor adjustments in composition, are used in both the hot-rolling and the cold-reduction methods. The practices for steelmaking, pouring, slabbing and rolling on a hot-strip mill are practically the same for both methods and are similar to those previously described for the grain-oriented grades. Beyond the hot-strip-mill rolling stage, however, the hot-rolling method and the cold-reduction method differ materially.

Hot-rolled strip coils intended for processing into sheets by the hot-rolling method are uncoiled and sheared into suitable lengths. The sheared pieces are doubled, reheated, and hot rolled in packs on hand-mills to a predetermined length. The length is such that the thickness of the individual sheets comprising the pack is reduced to the ordered thickness during the hot-rolling. After rolling, the packs are sheared to the ordered length and width and the sheets in the packs are separated from each other. Subsequent processing of

the sheets may vary somewhat, depending on the magnetic quality desired, but generally the treatments will include pickling, cold-rolling to flatten (no substantial reduction in thickness), and box annealing at a suitable temperature to develop the required magnetic properties.

The hot-rolled coils that are to be processed by the cold-reduction method are pickled (continuous) and then cold-reduced to the ordered thickness. The cold-reduction operation is accomplished in tandem mills or in single-stand reversing mills. The processing subsequent to cold-reduction includes suitable continuous-annealing and box-annealing treatments to develop the required magnetic properties. If the steel has been box-annealed in the form of coils, it is usually "thermally flattened" by continuous annealing.

The processing of grain-oriented and non-oriented electrical sheets described above may be followed by core plating if this is desired by the customer. The core plating operation is discussed in the following paragraphs.

CORE PLATING

The cores of transformers and other electrical structures are constructed of laminations to restrict the flow of eddy currents and thereby reduce this component of the core loss. For this to be successful, however, it is

necessary that each lamination be insulated from the others so that the eddy currents will not flow readily from one lamination to another. The normal surface oxide on fully-annealed silicon-steel sheets provides a certain amount of this necessary insulation or inter-lamination resistance, but in many cases it must be supplemented. The additional insulation is supplied by coating the sheets or laminations with a thin coating of varnish or core plate, which has good electrical resistance and which is capable of maintaining this resistance under normal operating conditions of temperature and pressure. When used in oil-immersed transformer cores, the core plate must not react with the oil because the products of this reaction would interfere with the circulation of the oil and thus cause excessive heating of the transformer.

There are two main types of core plate in common use, organic and inorganic, although there are many different classes of these two types. The usual method of applying either type is to pass the sheets or laminations through rolls coated with the core plate, and then through a combination flashing and baking oven. The thickness of the coating must be controlled, since the heavier the coating the greater will be the insulating properties. The coating cannot be too thick, however, because this will in effect reduce the amount of steel in a core of a specific height. Baking temperatures differ, depending on the composition and type of coating used. In general, the organic varnishes require more care in baking, since it is essential to volatilize off most of the vehicle and thus obtain a coating free from tackiness. Under-baking results in a soft, tacky coating which will not have satisfactory insulating properties at the pressure present in the core after assembling. Care also must be taken to prevent over-baking because the varnish will become carbonized and the insulation characteristics of the coating will be thereby impaired.

The inorganic type core plates are relatively free of volatile oils and organic matter and therefore require less exacting baking conditions. One of the requisites of an inorganic core plate is that it must be capable of withstanding annealing temperatures up to about 1500° F. Transformer manufacturers specify this type of core plate when they anneal the laminations before assembling the core.

FACTORS AFFECTING MAGNETIC PROPERTIES

Among the many factors that affect the magnetic properties of silicon-steel electrical sheets, the most important are (1) composition, (2) internal stress, and (3) grain orientation.

Composition—Electrical sheets composed of pure silicon ferrite, if obtainable, would be superior to the best commercial grades of silicon steel manufactured today. Practically all elements other than silicon and aluminum, when added to iron, adversely affect the magnetic properties desired in these "soft" magnetic materials. The absolute influence of each element is masked by the effects of other elements, but it is generally agreed that carbon is the most detrimental, followed in order by sulphur, oxygen, and nitrogen. Manganese and phosphorus apparently have little or no effect on magnetic properties, at least in the quantities normally present in commercial silicon steels. Consideration must be given to the state or form in which the impurity is present, however, as this may greatly alter its effect on magnetic properties. For example, widely-dispersed, fine particles of an impurity are more harmful than an agglomeration of the same impurity into a few relatively large particles.

Internal Stress—Three of the more important mag-

netic properties, permeability, coercive force, and hysteresis loss, are adversely affected by internal stress. For that reason, every effort is made in the processing of electrical sheets to produce a stress-free product. There are two main sources of internal stresses, (1) impurities that cause dislocations in the crystal lattice, and (2) mechanical stresses introduced during the rolling operations that are not completely removed during subsequent annealing or that are introduced by cooling too rapidly from the annealing temperature. Precautions are taken throughout the processing to reduce impurities such as carbon, sulphur, nitrogen, and oxygen to the lowest possible level to avoid internal stresses from this source. Mechanical stresses are minimized by annealing at high temperatures to completely remove the stresses introduced during working and by cooling very slowly from the annealing temperature.

Grain Orientation—Most magnetic properties are markedly affected by crystal orientation. That is, such properties are better in one of the three principal crystallographic directions of the unit cubes* than they are in the other two directions. This directionality of magnetic properties is undesirable in many applications such as in rotating machinery, but it has definite advantages in other applications. The cores of distribution and power transformers can be wound or constructed from laminations cut from the sheets to take advantage of such directionality. Consequently, the manufacturers of grain-oriented electrical sheets strive to develop this characteristic to a high degree. The processing of grain-oriented sheets was described in an earlier section of this chapter.

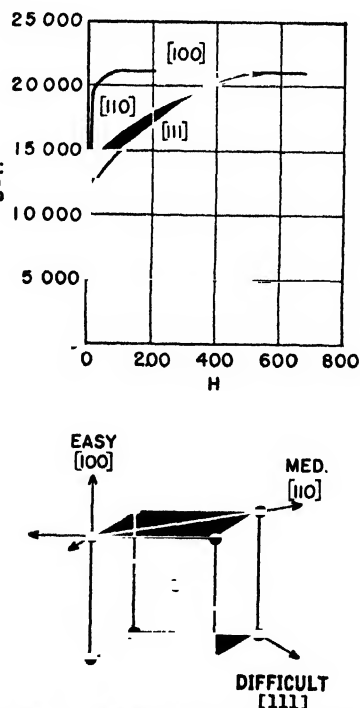


FIG. 46-1. Effect of orientation on the magnetic properties of a crystal, showing relative ease with which the cubes comprising the iron-silicon space lattice can be magnetized in different directions.

*A discussion of crystallography is found in Chapter 2 of this book.

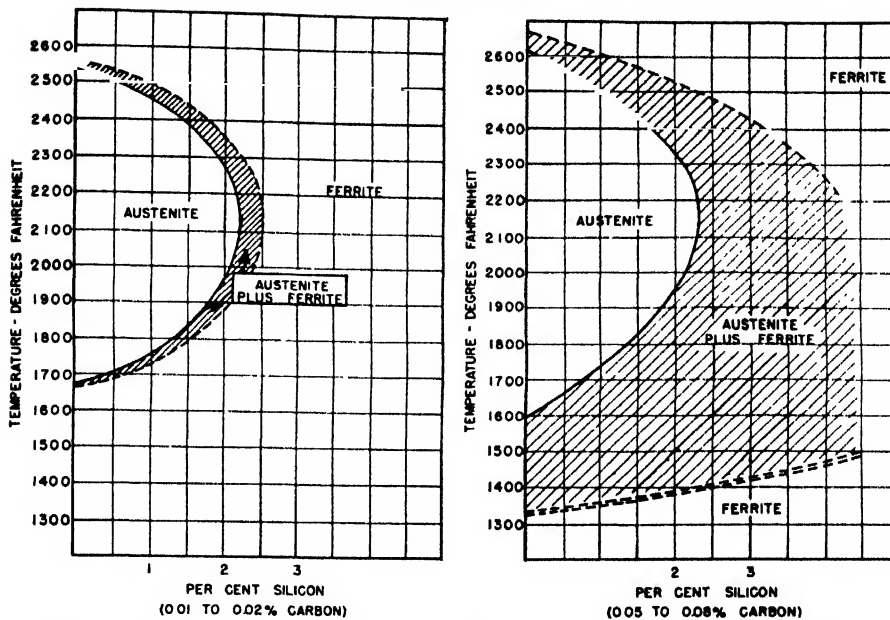


FIG. 46-2. (Left) Section of the ternary iron-silicon-carbon diagram at the 0.01 to 0.02 per cent carbon level; (right) same at the 0.05 to 0.08 per cent carbon level.

After completion of this processing, most of the grains are so arranged that edges of the unit cubes comprising each grain are aligned parallel to the rolling direction and face diagonals are aligned in the transverse direction. Because each cube is most easily magnetized along its edge, the [100] direction, the magnetic properties of grain-oriented sheets are best in the rolling direction. As shown in Figure 46-1, the face diagonal, [110] direction, of each cube is more difficult to magnetize than the cube edge, and the cube diagonal, [111] direction, is the most difficult to magnetize. The magnitude of these differences is also illustrated in Figure 46-1. Thus, the magnetic properties of grain-oriented sheets are best in the rolling direction, poorer at 90 degrees to the rolling direction, and poorest at 55 degrees.

Consumers of grain-oriented sheets take advantage of this directionality effect by constructing magnetic cores in such a manner that the main flux path is in the rolling direction. As examples of the extent of the directional effect, the core loss and permeability at 15 kilogausses in the rolling direction may be as much as $2\frac{1}{2}$ and 50 times better, respectively, than the same properties in the transverse direction.

EFFECTS OF SILICON ON IRON-CARBON ALLOYS

In addition to improving magnetic properties by decreasing eddy-current and hysteresis losses and by increasing permeability, silicon also has the following effects on metallographic, physical, and mechanical characteristics:

1. Alloys of iron and silicon form a metallographic gamma loop which is closed at a silicon content of about $2\frac{1}{4}$ per cent. In the complete absence of carbon, only alpha iron (or delta iron at high temperatures) is present in alloys containing more than $2\frac{1}{4}$ per cent silicon. The addition of even as little as 0.05 per cent carbon, however, causes some gamma iron (austenite) to be present up to at least 5 per cent silicon, as shown in Figure 46-2.

2. The magnetic transformation (A_s) is depressed about 9°C (16°F) for each per cent of silicon up to about 4 per cent silicon. At a silicon content of about 4 per cent, the magnetic transformation (A_s) occurs at approximately 730°C (1346°F).
3. Silicon increases the electrical resistivity of iron 11.4 microhms per cubic centimeter for each added per cent of silicon.
4. The addition of silicon to iron reduces the density of the resulting alloy.
5. The addition of silicon to a low-carbon steel decreases the tendency for the material to age (impairment of magnetic properties with increased time and temperature), provided other factors are normal and properly controlled.
6. Brittleness, or lack of ductility, increases as the percentage of silicon increases.

MECHANICAL PROPERTIES

Although no appreciable drawing operations are involved in fabricating articles from silicon-steel sheets, the material must have good punching and shearing qualities and must be reasonably flat so that motor, generator, and transformer laminations may be punched or sheared therefrom without difficulty. Typical mechanical properties of the various silicon-steel grades are listed in Table 46-II.

In general, as the silicon content increases, elongation, Erichsen ductility, and Amsler bend values decrease, and hardness, yield point, and tensile strength increase. In the transformer grades, however, differences in processing treatments may alter these trends to some extent.

DEFINITIONS OF TERMS AND METHODS OF TESTING

The practical value of electrical sheets is determined principally by their magnetic characteristics. The most

THE MAKING, SHAPING AND TREATING OF STEEL

Table 46—II. Typical Mechanical Properties of Electrical Sheets

Grade	Approximate Si Content (Per Cent)	Approximate Resistivity (Microhms per cm ²)	Yield Point Longitudinal (Lb. per Sq. In.)	Tensile Strength Longitudinal (Lb. per Sq. In.)	Per Cent Elongation in 2 In. Longitudinal	Rockwell Hardness B	Erichsen Cup (mm.)	Amsler Bends
Armature	¾	21	27,000	46,000	29	37	7.3	—
Electrical	1½	28	35,000	51,000	28	51	6.5	—
Motor	2¾	45	46,000	63,000	21	66	6.8	21
Dynamo	3¼	50	51,000	69,000	18	74	6.5	12
Transformer 72	3¾	56	59,000	76,000	12		3.6	13
Transformer 65	4	58	59,000	66,000	5	70	4.0	7
Transformer 58	4¼	61	60,000	61,600	4	75	3.5	5
Transformer 52	4½	64	60,000	69,500	2	74	2.7	3
Grain-Oriented (Transformer 73)	3¼	50	47,700	58,300	8	80	3.1	10

important characteristics that are used to evaluate this product are defined as follows:

Magnetic Aging is defined as the deterioration of magnetic properties of a material with increased time and temperature. Thus, as a material ages magnetically, its core loss increases and its permeability decreases. The American Society for Testing Materials Standard Aging Test consists of measuring the magnetic properties of a sample before and after heating it to 100° C for 600 hours and then determining the change in properties.

Core Loss is defined as the energy expended in magnetizing a material with an alternating electric current. The core loss of a low-loss material such as grain-oriented steel is composed of hysteresis, eddy-current and residual losses. Only the hysteresis and eddy-current losses are present in higher-loss material.

Core loss is measured on 3 by 50 centimeter specimens assembled with butt joints, or on 3 by 28 centimeter specimens assembled with lap and butt joints in the standard Epstein testing apparatus. The resulting values are expressed as watts per pound at a given flux density (usually 10 or 15 kilogausses) at a frequency of 60 cycles per second.

Eddy-Current Loss is that portion of the core loss caused by the circulation of stray magnetic currents in the magnetic material. These stray or eddy currents are generated in the core by the oscillation of the magnetic field in phase with the alternating electric current in the coil surrounding the core.

Eddy-current loss can be calculated from Maxwell's equation, or it can be determined by "separating" core loss into its components by the graphical method.

Hysteresis Loss is the power expended in a magnetic material, as a result of magnetic hysteresis, when the magnetic induction is cyclic. Hysteresis loss is proportional to the area of the hysteresis loop and can be determined by measuring the area of a hysteresis loop plotted from data obtained in a permeameter.

Permeability is a measure of the ease with which the magnetic lines of force can pass through a substance magnetized with a given magnetizing force. Quantitatively, it is expressed as the ratio between the magnetic flux density (B) produced and the magnetizing force (H) producing this flux density. The Greek letter μ (mu) is used to designate this ratio. Thus,

$$\text{Permeability, } \mu = \frac{B}{H}$$

Alternating-current permeability is measured in the Epstein apparatus on 3 by 28 centimeter specimens assembled with lap and butt joints. Direct-current permeability is measured in a permeameter, such as the

Fahy Simplex Permeameter in which the specimen size is 3 by 25 centimeters or longer.

Magnetic Flux Density, or Induction is the number of lines of magnetic flux per unit area at right angles to the direction of the flux.

When a magnetic core having a closed magnetic circuit is magnetized by current flowing in the windings of the coil which enclose the core, magnetic lines of force are generated which are designated as magnetic flux. The total magnetic flux in the core, designated as ϕ (phi), divided by the cross-sectional area of the core in square centimeters, gives the flux density (B), in lines per square centimeter, or gauss.

Magnetomotive Force is defined as the force which tends to produce a magnetic field. This force, when generated by an electric current flowing through a coil of wire, is proportional to the current and to the number of turns of wire in the coil. The term is not commonly used except in defining magnetizing force.

Magnetizing Force is defined as the magnetomotive force per unit of core length and is designated by the letter H. When the length of the core is expressed in centimeters, the unit of magnetizing force is the oersted. Another unit of magnetizing force sometimes used is ampere turns per inch which is 2.02 times greater than an oersted.

Saturation is the maximum flux density that can be obtained in a magnetic material. Further increases in magnetic force over that necessary to achieve this saturation flux density will not cause the generation of any additional flux within the material. Saturation occurs at a flux density of about 21,500 gauss in low-silicon steels and about 19,500 gauss in high-silicon steels.

Interlamination Resistance is the term applied to the electrical resistance measured perpendicular to the lamination plane in a stack of laminations under pressure. It indicates the effectiveness of surface oxides or core-plate coatings on the laminations in reducing interlamination (eddy-current) losses. Interlamination resistance values are generally expressed as ohms per square centimeter per strip at a given pressure in psi.

Lamination Factor or Space Factor is the ratio of the volume of a stack of laminations under a given pressure to that of the solid material of the same mass, assuming a definite density based on the chemical composition. Thus, the factor indicates the deficiency of effective steel volume due to the surface roughness and lack of flatness of the laminations, or to the presence of oxides and core-plate coatings on the surface of the laminations. Space factor values are generally expressed as a percentage which is obtained by multiplying the volume ratio by 100.

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Chapter 47

STAINLESS STEELS

General—As the name implies, stainless steels are more resistant to rusting and staining than are plain carbon and lower alloy steels. This superior corrosion resistance is brought about by addition of the element chromium to alloys of iron and carbon. Although other elements, such as copper, aluminum, and silicon, nickel and molybdenum, also increase the corrosion resistance of steel, they are limited in their usefulness, and the discussion in this chapter will be confined to the iron-chromium and iron-chromium-nickel steels in which chromium is the major element for conferring corrosion resistance.

The minimum amount of chromium necessary to confer this superior corrosion resistance depends upon the corroding agent. The American Iron and Steel Institute has chosen 4 per cent chromium as the dividing line between "alloy" steel and "stainless" steel and for this discussion, the AISI views will be adopted. The standard types of stainless steels listed by the AISI in Supplementary Information (dated April, 1955) to Section 24 of the "Steel Products Manual" are shown in Table 47—1. (The information in these manuals is revised from time to time.) Most of these types are available in the main product forms such as plates, bars, shapes, sheet, strip, and tubes. A detailed list of the products and sizes available for each type of stainless steel is given in the AISI Steel Products Manual.⁽¹⁾

As is true of all scientific developments, no single nation can claim credit for the stainless steels; Germany, England, and the United States shared alike in the development of these materials.

In 1912 Harry Brearley, head of the Brown-Firth Research Laboratory in England, while attempting to develop steels to resist the fouling and corrosion encountered in gun barrels, reported that a composition of 12.8 per cent chromium and 0.24 per cent carbon was quite resistant to corrosion. Brearley suggested that this composition be used for cutlery. In fact, our present cutlery steel, AISI Type 420 (12 to 14 per cent chromium, over 0.15 per cent carbon), is similar to the steel suggested by Brearley.

The development of the higher chromium-iron alloys was due to the work of F. M. Becket in the United States who, from 1903 on, was continuously attacking the problem of producing low-carbon ferrochromium from chromium ores. While investigating the effect of chromium on oxidation resistance at 2000° F (1095° C), Becket noted a marked increase in resistance as the chromium content was raised above 20 per cent. It is significant that even now and with steels containing appreciable quantities of nickel, 20 per cent seems to be the minimum amount of chromium necessary for oxidation resistance at 2000° F (1095° C).

The austenitic iron-chromium-nickel alloys were developed in Germany during the years 1909-1912 by Benno Strauss and Edward Maurer while searching for

materials for use in pyrometer tubes. Further work by Strauss and others ultimately led to the versatile 18 per cent chromium, 8 per cent nickel steels (popularly called 18-8) which are used so widely today.

CONSTITUTION

As mentioned above, the corrosion resistance or "stainlessness" of the stainless steels is primarily a function of their chromium content. Therefore, in order to understand the structures, heat treatments, and properties of the present commercial steels, a working knowledge of the iron-chromium, the iron-chromium-nickel, and iron-chromium-carbon diagrams is needed.

Iron-Chromium System—At the present time, the iron-chromium diagram is known only on a semi-quantitative basis. The diagram illustrated by Figure 47—1 is a compromise diagram based on the work of many investigators.^(2,3) The difficulties of establishing a precise equilibrium diagram are due primarily to two causes:

1. The difficulty of preparing pure alloys of iron and chromium (nitrogen and carbon seem to be the chief impurities).
2. The sluggishness of the iron-chromium alloys to respond to heat treatment.

The first difficulty is relatively unimportant because melting practices have improved and purer alloys may be expected. However, the second difficulty often may be overlooked and thus lead to false conclusions.

The high-temperature portion of the diagram is based on the data of Adcock reported in 1931.⁽⁴⁾ Adcock's data were accepted because of the high purity of his alloys and the techniques he employed.

The existence of the gamma loop was discovered in 1926 by Bain,⁽⁵⁾ who also showed the effect of higher carbon contents on enlarging the austenite field. The intermetallic compound, sigma phase, was discovered in 1927 by Bain and Griffiths.⁽⁶⁾

These two regions were extensively studied by other investigators and their data differed widely. This difference was especially marked for the sigma region. The most careful investigation, that of Adcock, who used the purest alloys of all, failed to detect the phase and Adcock came to the conclusion that the phase did not exist.

However, the evidence of other investigators was overwhelmingly in favor of the existence of sigma phase. In 1936, Jette and Foote,⁽⁷⁾ using alloys similar in purity to those of Adcock, were able to produce and identify sigma phase. Surprisingly enough, when they used treatments similar to those of Adcock, no sigma phase was formed. They concluded that Adcock's alloys were too pure and too well annealed for transformation to take place in the time allowed. Finally, the entire matter was conclusively settled in 1943 by Cook and Jones⁽⁸⁾ who, using Adcock's original alloys and longer transformation times, reported the sigma limit as shown. The boundary lines as shown by Cook and Jones are based

⁽¹⁾ References are at end of chapter.

STAINLESS STEELS

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Table 47—1. American Iron and Steel Institute Standard Type Numbers, Chemical Composition Limits and Ranges for Stainless Steels.

Chemical Composition, Per Cent								
Type Number	C	Mn Max.	Si Max.	P Max.	S Max.	Cr	Ni	Other Elements
301	0.15 Max.	2.00	1.00	0.045	0.030	16.00-18.00	6.00-8.00	Mo, Zr: 0.60 Max.* Se: 0.15 Min.
302	0.15 Max.	2.00	1.00	0.045	0.030	17.00-19.00	8.00-10.00	
302B	0.15 Max.	2.00	2.00-3.00	0.045	0.030	17.00-19.00	8.00-10.00	
303	0.15 Max.	2.00	1.00	0.20	0.15 Min.	17.00-19.00	8.00-10.00	
303 Se	0.15 Max.	2.00	1.00	0.20	0.06	17.00-19.00	8.00-10.00	
304	0.08 Max.	2.00	1.00	0.045	0.030	18.00-20.00	8.00-12.00	
304L	0.03 Max.	2.00	1.00	0.045	0.030	18.00-20.00	8.00-12.00	
305	0.12 Max.	2.00	1.00	0.045	0.030	17.00-19.00	10.00-13.00	
308	0.08 Max.	2.00	1.00	0.045	0.030	19.00-21.00	10.00-12.00	
309	0.20 Max.	2.00	1.00	0.045	0.030	22.00-24.00	12.00-15.00	
309S	0.08 Max.	2.00	1.00	0.045	0.030	22.00-24.00	12.00-15.00	
310	0.25 Max.	2.00	1.50	0.045	0.030	24.00-26.00	19.00-22.00	
310S	0.08 Max.	2.00	1.50	0.045	0.030	24.00-26.00	19.00-22.00	
314	0.25 Max.	2.00	1.50-3.00	0.045	0.030	23.00-26.00	19.00-22.00	
316	0.08 Max.	2.00	1.00	0.045	0.030	16.00-18.00	10.00-14.00	
316L	0.03 Max.	2.00	1.00	0.045	0.030	16.00-18.00	10.00-14.00	
317	0.08 Max.	2.00	1.00	0.045	0.030	18.00-20.00	11.00-15.00	
321	0.08 Max.	2.00	1.00	0.045	0.030	17.00-19.00	9.00-12.00	
347	0.08 Max.	2.00	1.00	0.045	0.030	17.00-19.00	9.00-13.00	
348	0.08 Max.	2.00	1.00	0.045	0.030	17.00-19.00	9.00-13.00	
403	0.15 Max.	1.00	0.50	0.040	0.030	11.50-13.00	1.25-2.50	Mo, Zr: 0.60 Max.* Se: 0.15 Min.
405	0.08 Max.	1.00	1.00	0.040	0.030	11.50-14.50		
410	0.15 Max.	1.00	1.00	0.040	0.030	11.50-13.50		
414	0.15 Max.	1.00	1.00	0.040	0.030	11.50-13.50		
416	0.15 Max.	1.25	1.00	0.06	0.15 Min.	12.00-14.00		
416 Se	0.15 Max.	1.25	1.00	0.06	0.06	12.00-14.00		
420	Over 0.15	1.00	1.00	0.040	0.030	12.00-14.00		
430	0.12 Max.	1.00	1.00	0.040	0.030	14.00-18.00		
430F	0.12 Max.	1.25	1.00	0.06	0.15 Min.	14.00-18.00		
430F Se	0.12 Max.	1.25	1.00	0.06	0.06	14.00-18.00		
431	0.20 Max.	1.00	1.00	0.040	0.030	15.00-17.00	1.25-2.50	Mo: 0.75 Max. Mo: 0.75 Max. Mo: 0.75 Max. N: 0.25 Max. Mo: 0.40-0.65 Mo: 0.40-0.65
440A	0.60-0.75	1.00	1.00	0.040	0.030	16.00-18.00		
440B	0.75-0.95	1.00	1.00	0.040	0.030	16.00-18.00		
440C	0.95-1.20	1.00	1.00	0.040	0.030	16.00-18.00		
446	0.20 Max.	1.50	1.00	0.040	0.030	23.00-27.00		
501	Over 0.10	1.00	1.00	0.040	0.030	4.00-6.00		
502	0.10 Max.	1.00	1.00	0.040	0.030	4.00-6.00		

* At producer's option; reported only when intentionally added.

† Columbium is also called niobium.

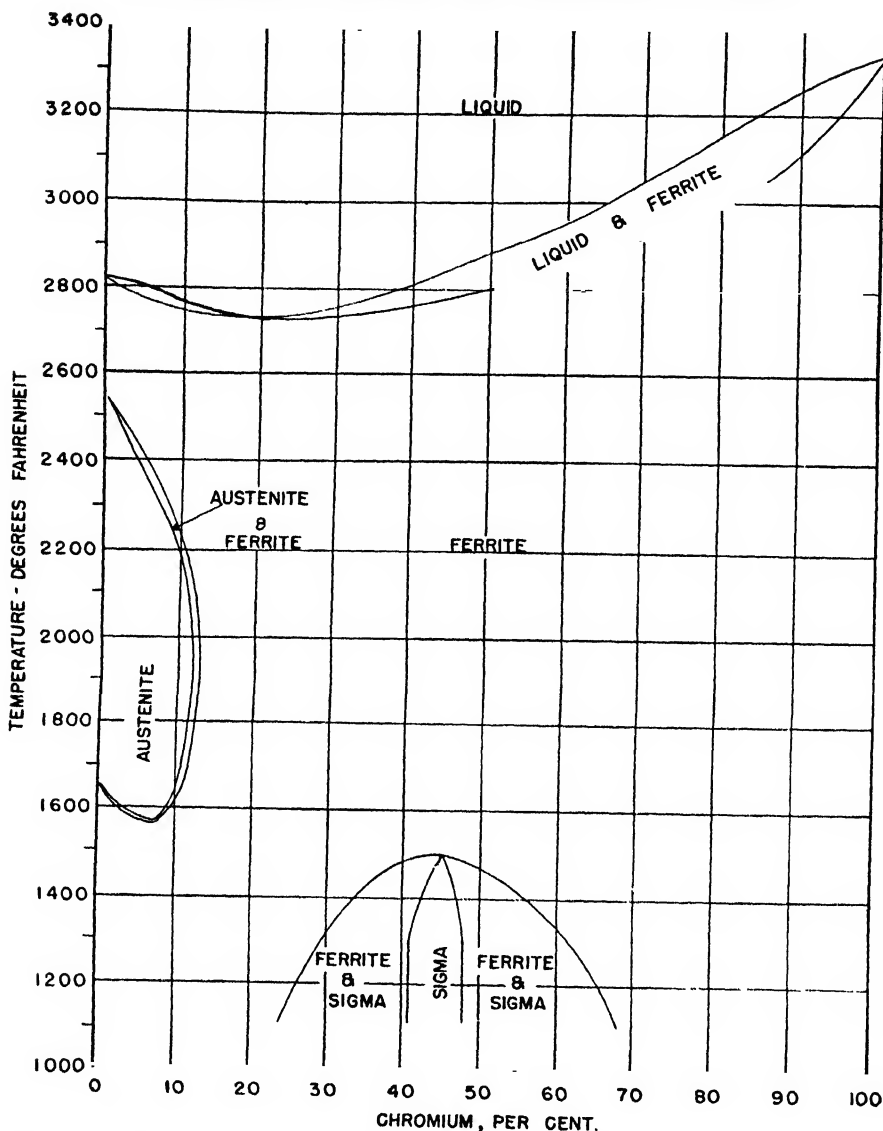


FIG. 47—1. The iron-chromium equilibrium diagram. (Adapted from American Society for Metals Handbook, 1948 Edition.)

on X-ray data but may still be questioned because equilibrium may not have been attained and longer annealing times may widen the ferrite + sigma region. An excellent bibliography pertaining to more recent studies of sigma phase in stainless steels has been published by Smith.⁽⁴⁰⁾

The diagram itself shows that as the chromium content increases, the austenite region is decreased until, above 12.5 per cent chromium, austenite no longer exists. After normal heat treatment, alloys above this chromium content consist entirely of a single phase—alpha, but as shown, sigma phase is stable and will form after long times at temperatures as low as 900° F (480° C) and below the upper limit.

Very little is known quantitatively about the effect of impurities upon this diagram. Bain⁽⁴¹⁾ has reported on the effect of carbon on the austenite region. Andersen and Jette⁽⁴²⁾ reported that silicon increases the sigma region. Bain and Aborn⁽⁴³⁾ reported that nickel also increases the sigma region. Aluminum, molybdenum, ti-

tanium, and columbium (niobium) probably increase it.

Iron-Chromium-Nickel System—Like the iron-chromium system, the iron-chromium-nickel system is known only semi-quantitatively but the existing diagrams are useful for interpretation of what otherwise would be mysterious behavior. The investigation of this system also has been hampered by slow reaction rates.

The first investigation of the iron-chromium-nickel system was conducted in 1909-1912 by Strauss⁽⁴⁴⁾ who studied the iron-rich corner. He found that the alloy having 20 per cent chromium and 7 per cent nickel contained free carbide which could be dissolved at 2280° F (1250° C) and retained in solution at room temperature by rapid cooling. He also noted a grain boundary precipitation at 1290° F (700° C).

The first comprehensive study was the heretofore mentioned investigation of Bain and Griffiths.⁽⁴⁵⁾ These investigators were the first to report sigma phase and they noted that nickel raised the upper temperature limit of this phase.

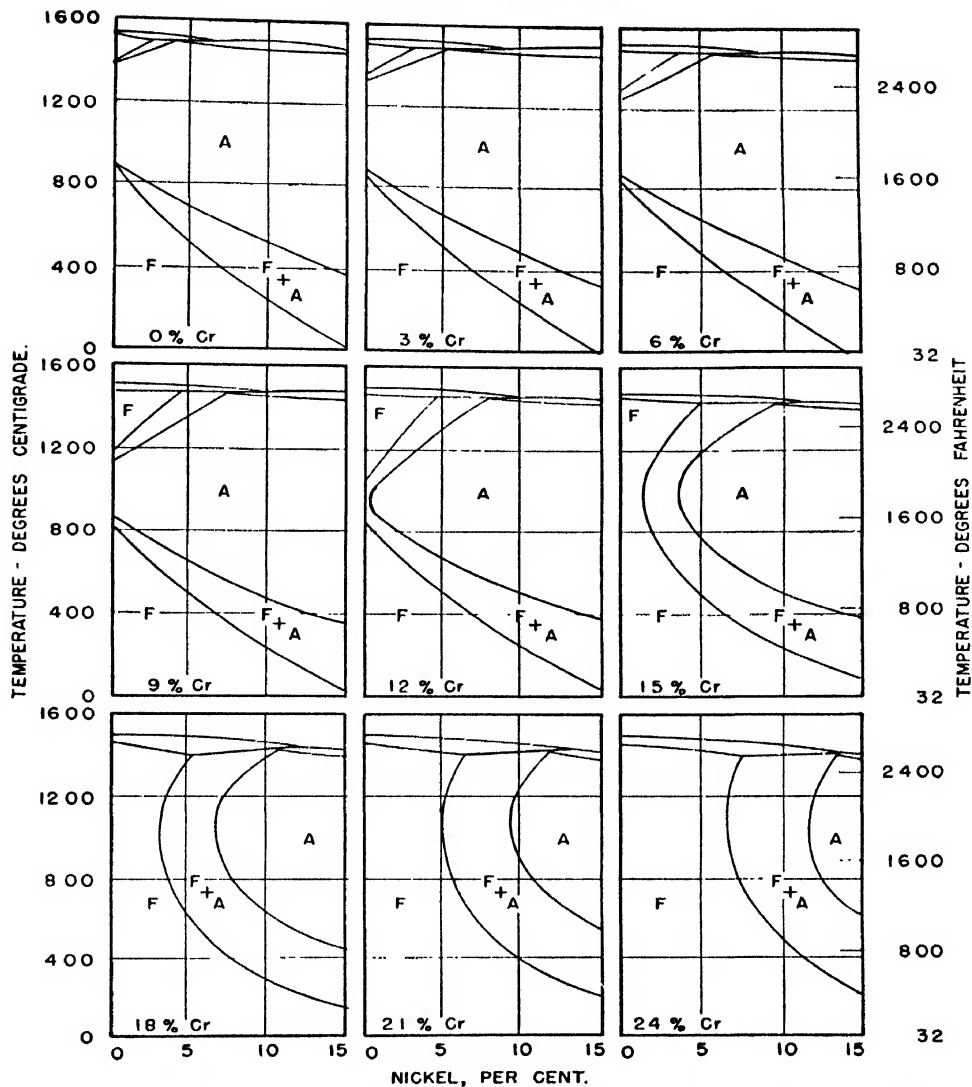


FIG. 47—2. The iron-chromium-nickel diagram at constant chromium contents.⁽¹⁰⁾ The symbols "A" and "F" stand for austenite and ferrite, respectively.

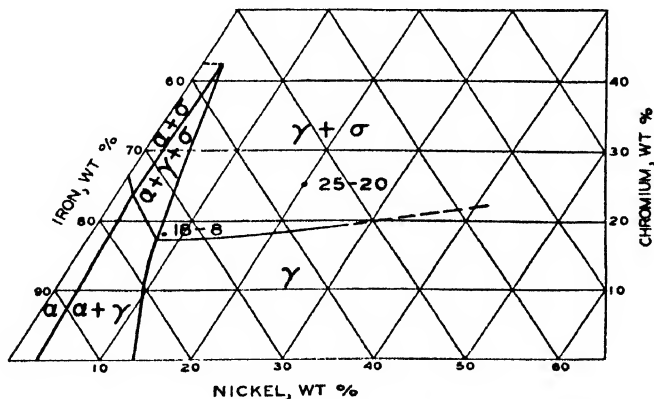


FIG. 47—3. The iron-chromium-nickel diagram. Isothermal section at 1200° F.⁽¹¹⁾ α , γ and σ represent alpha (ferrite), gamma (austenite), and sigma phases, respectively.

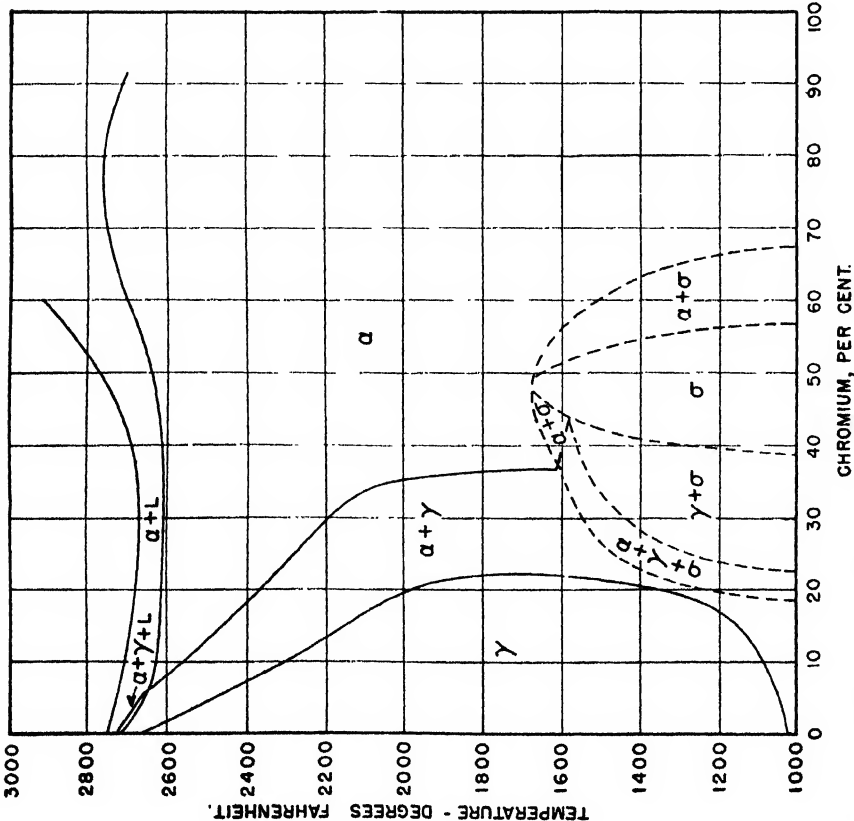


FIG. 47-4. (Above) The iron-chromium-nickel diagram. Section at 8 per cent nickel.^{as} α , γ and σ represent alpha (ferrite), gamma (austenite), and sigma phases, respectively. L signifies liquid.

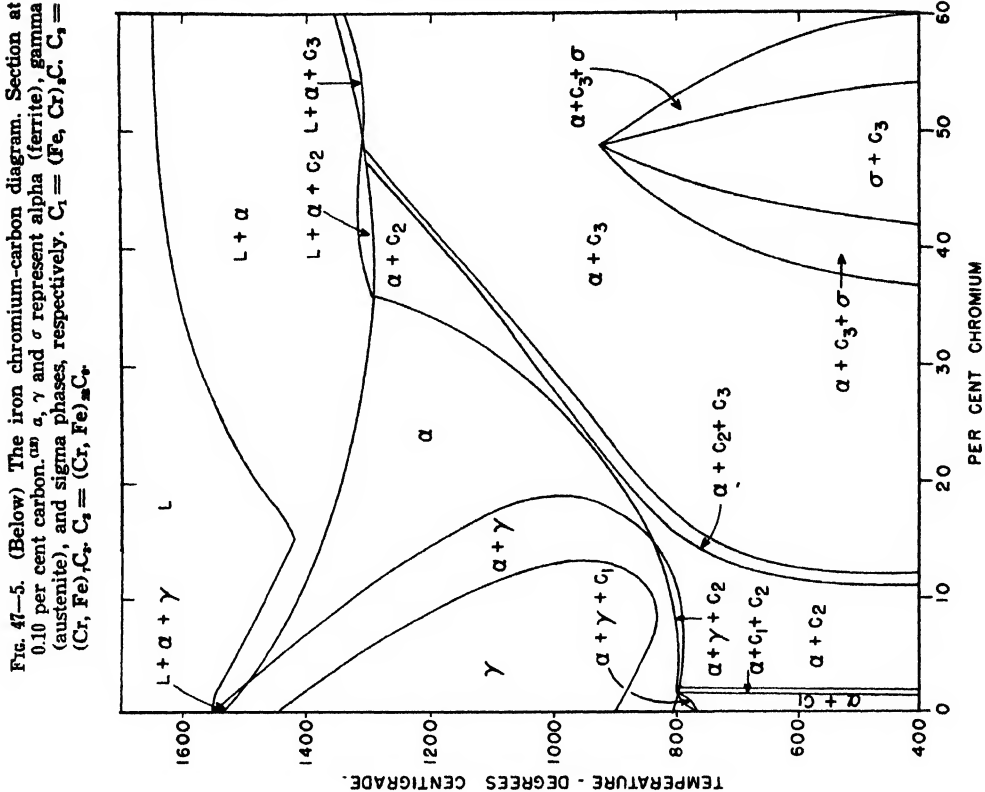


FIG. 47-5. (Below) The iron chromium-carbon diagram. Section at 0.10 per cent carbon.^{as} α , γ and σ represent alpha (ferrite), gamma (austenite), and sigma phases, respectively. $C_1 = (Fe, Cr)_C$, $C_2 = (Cr, Fe)_C$, $C_3 = (Cr, Fe)_C$.

Bain and Aborn⁹⁹ made a comprehensive review of the iron-chromium-nickel system, which is summarized in Figure 47-2.

The work of the British in this field has been very extensive. Figure 47-3 shows an isothermal section at 1202° F (650° C) based on the data of Rees, Burns and Cook.¹⁰¹

Figure 47-4 shows a diagram at constant nickel content.

Iron-Chromium-Carbon System—The most complete information on the iron-chromium-carbon system has been supplied by Tofaute and his co-workers.^{102, 103} Figure 47-5¹⁰³ shows a section through the iron-chromium-carbon diagram at 0.10 per cent carbon and, as may be seen, the diagram is quite complex. The three carbide phases shown in this diagram are:

1. (Fe, Cr)₃C—iron carbide (cementite) capable of dissolving up to 15 per cent chromium.
2. (Cr, Fe)₇C₃—chromium carbide capable of dissolving up to 50 per cent iron.
3. (Cr, Fe)₂₃C₆—chromium carbide capable of dissolving up to 25 per cent iron.

An excellent discussion of the various investigations which have been made on the iron-chromium-carbon alloys is given by Kinzel and Crafts.¹⁰⁴

MANUFACTURE AND FABRICATION

For the purpose of general discussion, the stainless steels are grouped into three classes:

1. **Martensitic**—Those iron-chromium alloys that lie within the gamma loop and thus are hardenable by heat treatment. Include Types 403, 410, 414, 416, 420, 431, 440A, 440B, 440C, 501 and 502.
2. **Ferritic**—Those iron-chromium alloys that are largely ferritic and not hardenable by heat treatment (ignoring the 885° F embrittlement). Include Types 405, 430, 430F and 446.
3. **Austenitic**—The iron-chromium-nickel alloys not hardenable by heat treatment and predominantly austenitic as commercially heat treated. Include Types 301, 302, 302B, 303, 304, 305, 308, 309, 310, 314, 316, 317, 321 and 347.

Melting—All of the stainless steels are melted in either the electric-arc or high-frequency induction furnace, the largest tonnages by far being melted in electric-arc furnaces. A description of arc-furnace melting practice is contained in earlier chapters.

Hot Working—General—Before discussing the details of any particular process used for the production of stainless steel, a few general remarks will be made concerning the hot- and cold-working characteristics of these materials. These general remarks, describing the salient differences in behavior between the carbon steels and the stainless steels, will apply to any hot- or cold-working operation to which the stainless steels may be subjected.

Unlike the performance characteristic of carbon steels, ruptures in the stainless steels do not "heal" easily, if at all, and more conditioning is necessary; in fact, after many of the hot-working operations, the surfaces of the resultant product are completely conditioned. All of the stainless steels have lower thermal conductivity than the carbon and low-alloy steels and, accordingly, precautions must be taken when heating, or surface burning will result. Also, for most of the stainless grades, the temperature ranges for optimum hot-working char-

acteristics are narrower than those for the carbon steels and, hence, closer temperature control is necessary when hot working the stainless steels.

For all grades of stainless steel, optimum hot workability is obtained when the structure of the steel at the hot-working temperature consists essentially of a single phase. Small amounts of ferrite in the structure of the austenitic and martensitic steels and small amounts of austenite in the structure of the ferritic steels can be tolerated, but must be kept within proper limits either by proper adjustment of the chemical composition of the steel or by adjustment of the hot-working temperature.

The martensitic stainless steels can be forged, pierced and rolled. However, because these steels are air hardening, they must be annealed after rolling before any subsequent operation such as conditioning or cold working. Their "as-rolled" hardness also makes for brittleness which must be taken into account when handling the hot-rolled product.

The ferritic stainless steels also can be forged, pierced and rolled. These steels are very soft when hot, thus they are easily marked by guides or rolls, and spread considerably during hot rolling. Over-heating these grades causes excessive grain growth, which makes the material susceptible to tears and cracks. Additions of nitrogen have helped somewhat in preventing grain growth. To refine the grain size, finishing temperatures are kept as low as possible. Annealing is necessary for those steels which are susceptible to the 885° F (475° C) embrittlement.

The austenitic stainless steels are generally stronger than ferritic steels at rolling temperature and, consequently, require more power for deformation. Like the ferritic steels, the austenitic steels are susceptible to grain growth and overheating should be avoided. Low finishing temperatures are not practicable because of the power required. During the heating of these nickel-bearing austenitic steels, special precautions are taken to keep the sulphur content of the furnace or soaking pit atmospheres at a minimum because these steels, after being heated in atmospheres containing sulphur, tend to tear and crack during rolling. Apparently, the sulphur in the atmosphere combines with the nickel in the steel to form nickel sulphide. This reaction usually occurs at the grain boundaries of the metal and, because the nickel sulphide is liquid at the rolling temperatures, the steels so attacked are weak and easily break apart during rolling. The steels which are completely austenitic after cooling from the rolling temperature to room temperature can be rolled without difficulty. The usual precautions, such as slow heating, must be observed. On the other hand, those steels such as 18-8 Mo (Type 316), 18-8 Cb (Type 347), 18-8 Ti (Type 321), and 25-12 (Type 309), show poor hot-working characteristics which are blamed on the presence of delta ferrite. The explanation for the poor working characteristics is that the difference in plasticity between the soft ferrite and the tough austenite causes ruptures.

Table 47-II lists the forging and annealing temperature ranges commonly used for the standard stainless steels, based on the 1950 AISI Steel Products Manual.

Cold Working—General—With the exception of the high-carbon hardenable steels, all of the stainless steels can be cold worked. However, certain precautions must be taken.

The ferritic stainless steels, especially those containing over 20 per cent chromium, are extremely notch sensitive at room temperature and care must be taken to avoid notches, otherwise considerable breakage will result. However, between 400° F (205° C) and 600° F

Table 47—II. Forging and Annealing Temperatures for Stainless Steels* From AISI Steel Products Manual, Section 24 (1950).

Type of Steel	Initial Forging Temperature Range (° F)	Annealing Temperature Range (° F)	Rate of Cool from Annealing Temperature
301	2100-2300	1850-2050	Rapid
302	2100-2300	1850-2050	Rapid
302B	2050-2250	1850-2050	Rapid
303	2100-2350	1850-2050	Rapid
304	2100-2300	1850-2050	Rapid
305	2100-2300	1850-2050	Rapid
308	2100-2300	1850-2050	Rapid
309	2050-2250	1900-2050	Rapid
310	2000-2250	1900-2100	Rapid
314	1900-2050	2100	Rapid
316	2100-2300	1850-2050	Rapid
317	2100-2300	1850-2050	Rapid
321	2100-2300	1750-1950	Rapid
347	2100-2300	1850-2050	Rapid
403	2000-2200	1500-1650	Slow
	(Retarded Cool)	or 1200-1400	
405	1950-2050	1350-1500	
410	2000-2200	1500-1650	Slow
	(Retarded Cool)	or 1200-1400	
414	2100-2200	1200-1300	
	(Retarded Cool)		
416	2100-2300	1500-1650	Slow
	(Retarded Cool)	or 1200-1400	
420	2000-2200	1500-1650	Slow
	(Retarded Cool)	or 1350-1450	
430	1900-2050	1400-1500	
430F	1950-2100	1250-1400	
431	2100-2250	1150-1225	
	(Retarded Cool)		
440A	1900-2200	1550-1650	Slow
	(Retarded Cool)	or 1350-1450	
440B	1900-2150	1550-1650	Slow
	(Retarded Cool)	or 1350-1450	
440C	1900-2100	1550-1650	Slow
	(Retarded Cool)	or 1350-1450	
446	1950-2050	1450-1600	Rapid
501	2100-2200	1525-1600	Slow
	(Retarded Cool)	or 1325-1375	
502	2100-2200	1525-1600	Slow
		or 1325-1375	

*Preheating time about twice that required for carbon steel.

(315° C) the steels are tough, and difficult cold-working operations are successfully accomplished by working the material in this temperature range.

Cold work causes some austenitic stainless steels to transform partially to a low-carbon martensite. This transformation, plus the effect of the strain hardening caused by the cold work itself, causes such austenitic steels to have a high rate of work hardening. More power is required to work these steels. Figure 47—6 shows flow

diagrams illustrating the various steps involved in the manufacture of stainless-steel products.

The Rolling of Stainless-Steel Ingots to Blooms and Slabs—The equipment used for the heating and rolling of stainless-steel ingots is the same as that used for carbon-steel ingots. However, as previously mentioned, close temperature control and avoidance of sulphur contamination are precautions which should be followed when heating the stainless steels. Also, the stainless steels require more conditioning than the carbon steels. The bloom and slab products are completely conditioned.

Rolling of Billets—The blooms used for the production of billets are also completely conditioned prior to heating for rolling. As was true for the rolling of ingots, the rolling of stainless-steel blooms to billets is performed on the same equipment used for carbon steels and the usual precautions of close temperature control and avoidance of sulphur contamination are taken.

After rolling, the air-hardenable martensitic grades must be cooled slowly in order to soften the material. This practice prevents thermal cracking during subsequent conditioning.

Rolling of Stainless-Steel Plates—The equipment used for the heating and rolling of stainless-steel plates is the same as that used for the heating and rolling of carbon-steel plates. However, because the austenitic stainless steels are very stiff at elevated temperatures, they require more power for rolling. Consequently, the amount of reduction per pass is smaller for the austenitic grades. Also, these steels spread less than do the ordinary steels and due allowances are made for this lack of spread in order that the resulting plate widths will satisfy dimensional requirements.

After rolling, the stainless-steel plates are annealed and pickled. As might be expected, the annealing temperatures employed depend upon the composition of the material, and the specific annealing temperatures used for the standard AISI grades are listed in Table 47—II. The pickling procedure used for stainless steel varies from plant to plant. One installation consists of a 10 per cent sulphuric acid bath operated at 140° to 160° F (60° to 70° C) and a 10 per cent nitric acid, 4 per cent hydrofluoric acid bath operated at 130° to 150° F (50° to 70° C). The first bath softens and loosens the scale but will not remove it completely; the second solution will remove the scale loosened by the first solution.

Most recent of the developments in the pickling of stainless steels is the use of molten salts consisting of sodium hydroxide to which is added some agent such as sodium hydride. These molten-salt descaling processes are rapid and efficient and have replaced many acid pickling installations.

After annealing and pickling, the stainless plates have what is called a No. 1 (hot-rolled, annealed, and pickled) finish and, in this condition, the plates are sheared to size, and are then suitable for shipment.

Rolling of Stainless-Steel Bars—The billets used for the rolling of stainless-steel bars are conditioned as the surface requires it. Martensitic steels must not exceed 275 Brinell hardness prior to conditioning, and if this hardness is exceeded, the billet must be annealed before swing grinding in order to prevent thermal cracking which might occur during the grinding operation or during heating for rolling. Prior to heating for rolling, the ends of the billets are pointed by a scarfing torch to prevent the splitting of the ends of the bar and also to decrease slippage when entering the mill.

During the rolling of the ferritic grades, spread control is important and is accomplished by providing a billet size slightly less in cross-section than that which

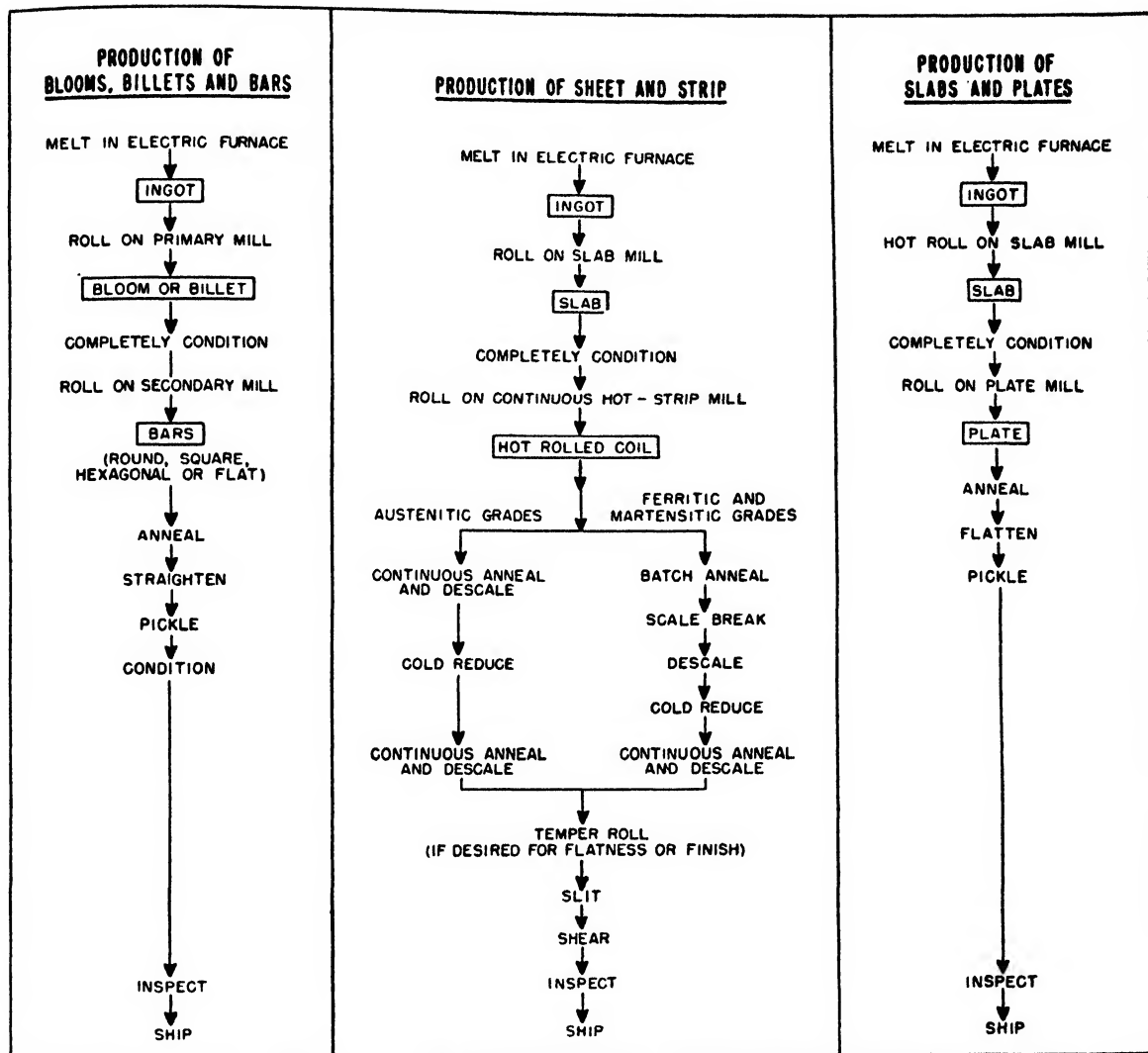


FIG. 47—6. Flow sheets indicating general principal steps in the production of stainless-steel products.

would be used for carbon or alloy steels. This smaller size permits lighter initial drafting.

On delivery from the rolling mill, the austenitic and ferritic stainless-steel bars are rapidly (air) cooled, while cooling of the martensitic stainless-steel bars is deliberately retarded. Sections of the latter two inches and over are cooled slowly in covered pits, while those under two inches have cooling retarded by packing on the hot bed.

After annealing and pickling, stainless-steel bars are straightened on standard equipment and may be shipped in this condition or finished by centerless grinding or cold drawing.

Rolling of Stainless-Steel Sheet and Strip—In this section, the discussion of the rolling of stainless-steel sheet and strip will be confined to the continuous method by which the largest tonnage of stainless steels is produced. However, recognition should be given to hand-mill methods which, although being gradually replaced by the continuous method, find importance for the production of those grades of stainless steel that are difficult to roll and also for the production of small lots not conveniently produced by the continuous method.

Figure 47—6 presents a flow chart indicating typical steps in the processing of stainless-steel sheet and strip.

The slab or billet used for the production of stainless-steel sheet or strip is completely conditioned, although some progress is being made toward the practice of conditioning only those areas containing defects ("spot conditioning"). As is true for the other products, stainless-steel sheet and strip is rolled on the same equipment as that used for carbon-steel sheet and strip.

After rolling and annealing, all of the stainless steels are descaled, usually by pickling in acids. For the plain-chromium grades of stainless steel containing up to 18 per cent chromium, the hot-rolled sheet or strip, in coil form, is batch annealed at sub-critical temperatures. On the other hand, the plain-chromium grades containing over 18 per cent chromium, and the austenitic grades, are annealed on a continuous unit and quenched from the annealing temperature. Often, for the austenitic steels, this annealing is performed in an oxidizing atmosphere which, by producing a heavy scale, "burns off" the defects and thus reduces the amount of conditioning necessary at some later stage. The quenching practice used depends upon the thickness of the material. For

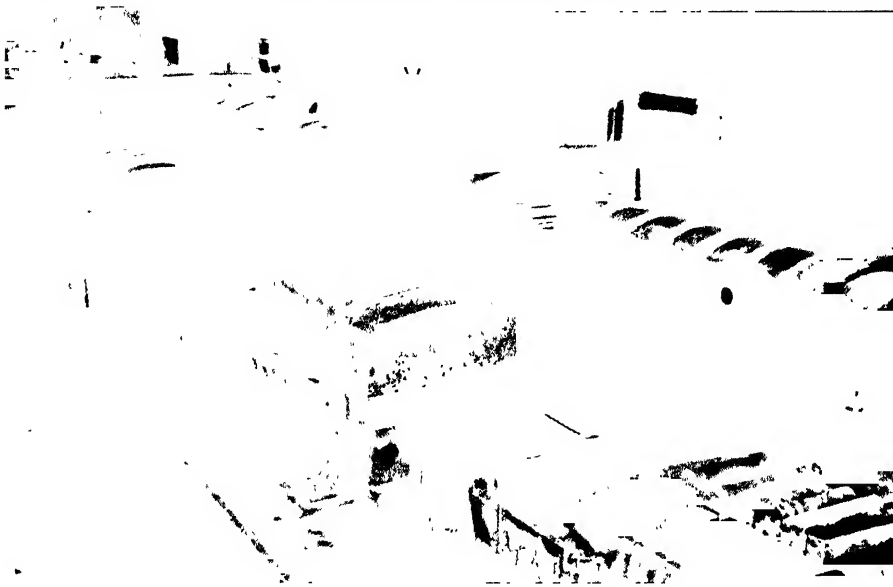


FIG. 47-7. Stainless-steel strip entering a continuous pickling line.

thick materials, high-pressure water sprays are used, but for thin materials, cooling in still air is sufficient. All of the stainless steels are descaled on continuous units, usually arranged in tandem with a continuous annealing unit.

A typical continuous descaling or pickling installation (see Figure 47-7) consists of two 35-foot long tanks containing approximately 15 per cent hydrochloric acid (HCl) at 160°F followed by a tank of similar size containing about 4 per cent hydrofluoric acid (HF) and 10 per cent nitric acid (HNO_3) at 150°F to 170°F . In some installations, electrolytic-pickling facilities using either cold sulphuric acid (H_2SO_4) or nitric acid (HNO_3) are substituted for the first two tanks.

Molten-salt descaling processes are now being used commercially. Normally a light acid pickle, usually hot nitric acid, follows the descaling treatment. These processes provide scale removal on all grades of stainless steel without metal loss and result in a smoother surface than is obtainable from acid pickling. The method requires more heat input than pickling, and care must be taken to prevent the introduction of any water.

After annealing and pickling, the material is inspected and the defects removed by grinding or by reannealing and repickling. At this stage, the material has what is known as a No. 1 or a hot-rolled, annealed and pickled finish and, after shearing to the required lengths and widths, may be shipped as hot-rolled, annealed and pickled sheets. However, in order to obtain lighter gages and improvements in surface, grain size and mechanical properties, the material is cold rolled in coil form.

Usually, the cold rolling of stainless-steel sheet and strip is performed on a reversing mill, although a tandem mill may be used. Depending upon the final thickness desired, an intermediate anneal may or may not be used. This intermediate anneal and pickle, as well as the final anneal and pickle, is performed on a continuous unit (see Figures 47-8 and 47-9).

After annealing and pickling, the surface of the cold-rolled material has what is called a No. 2-D (dull, cold-rolled) finish for sheet or a No. 1 cold-rolled finish for strip. In this condition, the material either may be sheared to desired lengths and shipped, or may remain

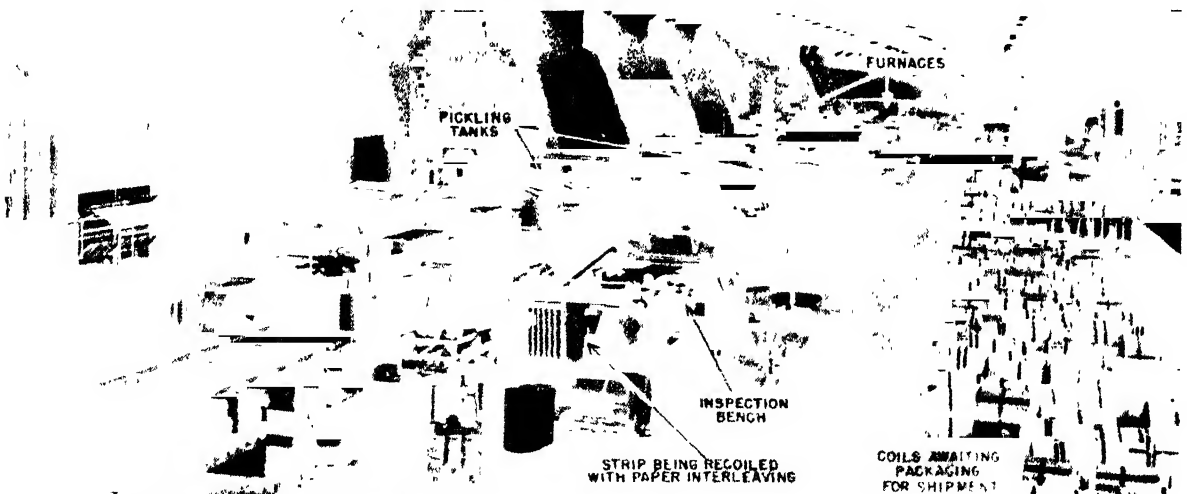


FIG. 47-8. General view of stainless-steel strip annealing and pickling lines.



FIG. 47-9. Close-up view of stainless-steel strip emerging from one of the continuous furnaces shown in Figure 47-8.

in coil form and be subjected to further processing. If a brighter finish is desired, the material is rolled on a temper mill; the finish resulting from this process is called a No. 2-B (bright, cold-rolled) finish for sheet or a No. 2 cold-rolled finish for strip. Higher finishes are obtained by mechanical polishing. Flattening, or leveling, is accomplished on standard units used for this purpose.

Of special interest is the work-hardening characteristic of the austenitic stainless steels which permits these grades to be produced to tensile strengths as high as 200,000 pounds per square inch. The composition best suited for the production of these high strengths is AISI Type 301 (17 per cent chromium, 7 per cent nickel). Usually, this grade is supplied to four standard minimum tensile strength levels of 125,000, 150,000, 175,000 and 185,000 pounds per square inch. Materials having these respective tensile strengths are designated commercially as having $\frac{1}{4}$, $\frac{1}{2}$, $\frac{3}{4}$ and full-hard temper. The amount of cold reduction necessary to produce these strengths depends upon the composition of the material. Generally, 10 per cent cold reduction is needed for producing $\frac{1}{4}$ hard, 22 per cent for $\frac{1}{2}$ hard, 32 per cent for $\frac{3}{4}$ hard, and 37 per cent for full-hard material.

HEAT TREATMENT

Heat Treatment of Iron-Chromium Stainless Steels—Knowing the phase diagrams of the stainless steels, an understanding of the philosophy of the commercial heat treatments employed for the various grades becomes possible.

From 5 per cent chromium to 12 per cent chromium, using 0.15 per cent carbon as a base, the steels are hardenable by the austenite-to-martensite transformation. As would be expected from the high chromium contents, the hardenability of these steels is good and increases with increasing chromium and carbon.

However, most of the applications for the 5 to 9 per cent chromium steels require the metal to be in the most ductile condition, and the "softening" practice employed consists of normalizing and tempering or one

of the various annealing cycles. Ductility is required because these steels, before they are placed into service, are subjected to various cold-fabrication practices. Because these steels are air hardening, welding must be performed with the necessary precautions such as preheating and post-heating. Often in field welding, these precautions cannot be taken and, when this situation is anticipated, stabilization with titanium or columbium (niobium) may be employed. Titanium and columbium form stable carbides and the material behaves like a low-carbon stainless steel and has lower hardness than material without stabilizing elements. Improvements in melting practices have led to the development of extremely-low-carbon grades which, theoretically, should not harden after welding.

Depending upon the intended application, the 12 per cent chromium steels are used in the fully-hardened and tempered as well as the annealed condition. These materials may be either air-cooled or oil-quenched from the hardening temperature. Oil quenching produces slightly higher hardnesses but air cooling is employed in order to minimize the danger of cracking or warping. Figure 47-10 shows the effect of tempering temperature on the hardness and tensile properties of hardened Type 410 (12 per cent chromium, 0.10 per cent carbon). The impact strength decreases in the temperature range 750° to 950° F; also maximum corrosion resistance is not obtained in the range 1000° to 1200° F, hence, the range 750° to 1200° F is to be avoided. Heating this type of steel below 750° F is commonly called "stress relieving" while heating above 1200° F is called "tempering." Annealing is usually accomplished by heating above the lower critical temperature and slow cooling, but also may be accomplished by sub-critical annealing. As is true for the lower chromium steels, the 12 per cent chromium steels must be welded with caution because of air-hardening. Unlike the lower chromium steels, the hardenability of the 12 per cent chromium steels is lowered by small additions of aluminum. These small amounts of aluminum (0.3 per cent) are sufficient to minimize austenite formation and make the steel

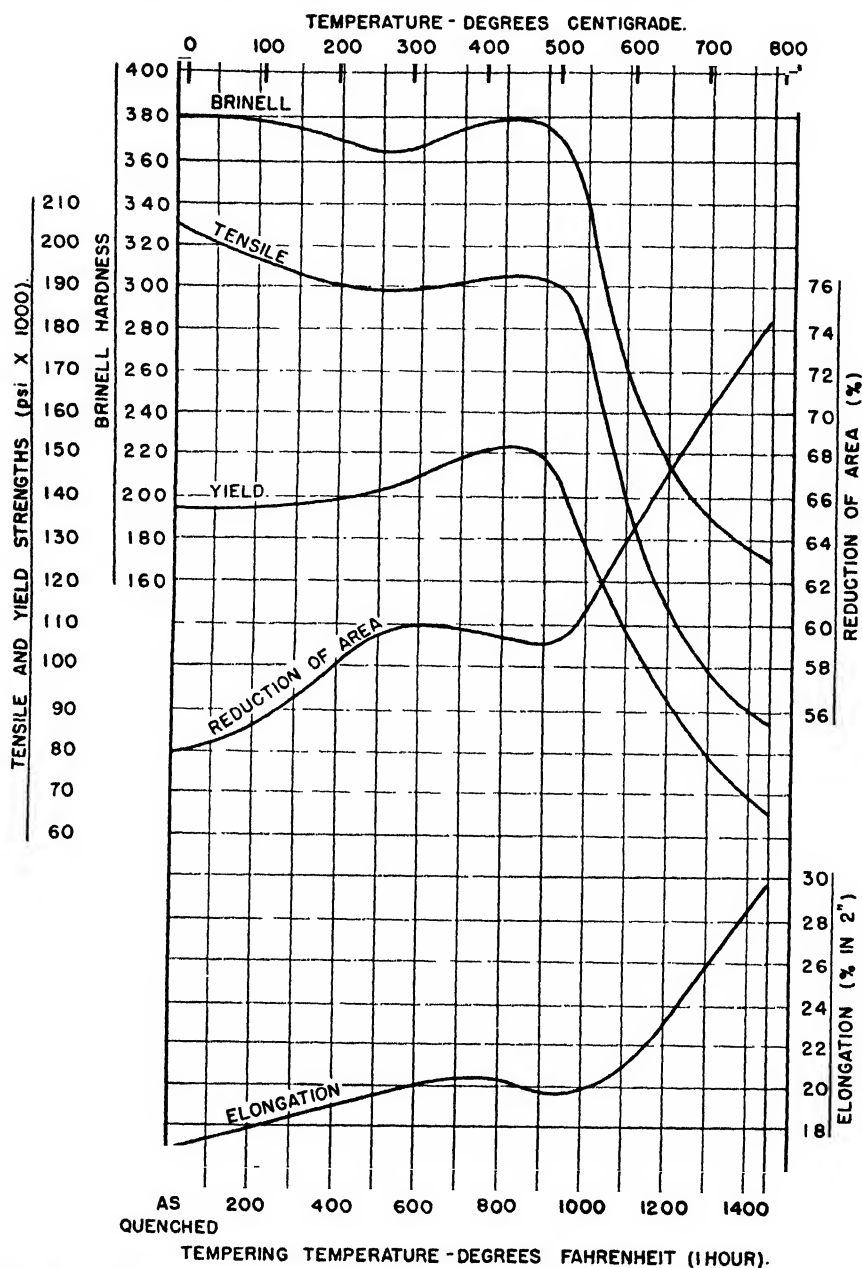


FIG. 47-10. The effect of tempering for one hour on the hardness and tensile properties of hardened Type 410 (12% Cr-0.10% C) stainless steel.

ferritic at all temperatures so that no phase transformation, and consequently no hardening, occurs.

The higher chromium steels (17 per cent chromium, high carbon excepted) are completely ferritic. Annealing would then seem to be a simple process of heating at a recommended temperature for a reasonable time and slow cooling. However, two difficulties arise. First, the alloys are single phase except for carbide, hence, no grain refinement by a phase transformation is possible. A large grain size once formed by improper heating procedure will be retained on cooling to room temperature. Only by cold work and recrystallization can the grain size be reduced. The fact that the high-chromium steels are inherently notch sensitive makes the effect

of grain coarsening even worse. Additions of nitrogen have been used to obtain a finer grain size. The nitrogen forms small pools of austenite which inhibit grain growth.

The second difficulty is that embrittlement occurs when the steels are heated in or slowly cooled through the temperature range of 800° to 1400° F. The embrittlement, which is actually an age-hardening phenomenon, is caused by the precipitation of a body-centered cubic phase of iron and chromium containing 70 to 80 per cent chromium.⁽⁵⁰⁾

Keeping in mind grain growth and embrittlement, the high-chromium steels are annealed by heating in the temperature range 1400° to 1700° F and cooling rapidly.

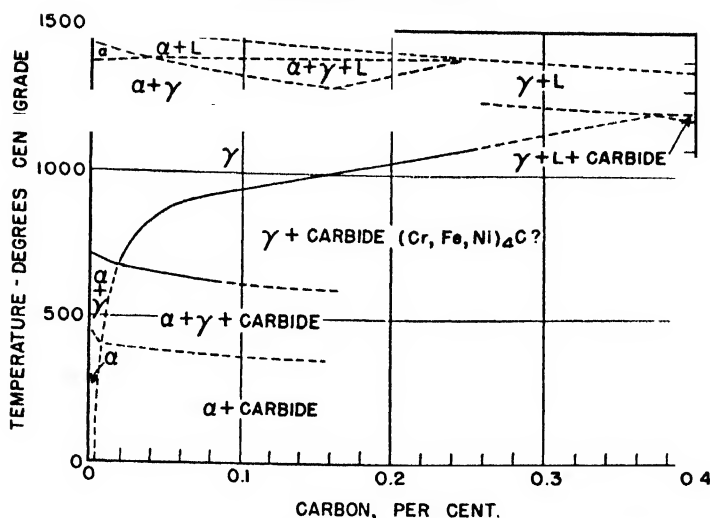


FIG. 47-11. The effect of carbon on the constitution of stainless steels containing 18 per cent chromium and 8 per cent nickel.⁽¹⁰⁾

Iron-Chromium-Nickel Stainless Steels—The austenitic stainless steels also are considered to be single phase although this belief is erroneous for two reasons.

First, in ordinary 18-8, austenite is not thermodynamically stable at room temperature. By the means of plastic deformation at or below room temperature, meta-stable austenite can be transformed, at least partially, to martensite. Furthermore, additions of aluminum and titanium in sufficient quantities cause the 18-8 steel to transform to a low-carbon martensite without benefit of cold work. This effect has been used advantageously in developing Stainless W, an age-hardening ferritic 17 per cent chromium, 7 per cent nickel steel.⁽¹⁴⁾

Second, the carbide phase, unfortunately, cannot be ignored. In Figure 47-11⁽¹⁰⁾ it will be noted that the carbide solubility changes abruptly with temperature. Therefore, during slow cooling a carbide precipitation occurs and these carbides, rich in chromium, precipitate at the grain boundaries. At the temperature where the precipitation occurs, chromium diffusion from the matrix is not rapid enough to replenish the chromium taken out of the immediate vicinity of the carbide and, consequently, this region is low in chromium. Because chromium is the element largely responsible for the excellent corrosion resistance, the region adjacent to the carbide becomes lower in corrosion resistance and the material is susceptible to intergranular corrosion.

The austenitic steels are, therefore, heat treated by an anneal at a temperature high enough to effect carbide solution but low enough to minimize grain growth, and then cooled to room temperature rapidly enough to keep the carbides in solution.

Such a treatment is not always possible, especially where these steels are welded in the field, and modifications of the austenitic grades have been developed. These modified steels contain titanium or columbium (niobium) which combine with the carbon and eliminate intergranular carbide precipitation and susceptibility to intergranular corrosion. Titanium in amounts of five times the carbon and columbium in amounts of ten times the carbon are considered to be sufficient, although the actual amounts necessarily depend upon the grain size and composition (other than carbon content) of the material (see also "Intergranular Corrosion").

The relationship of these variables to the amount of titanium required has been quantitatively evaluated, and a suitable formula developed.⁽¹⁴⁾

The titanium or columbium grades are sometimes given a stabilizing treatment at 1600° F to insure complete chemical combination of carbon with titanium or columbium.

CORROSION RESISTANCE

As was mentioned previously, the corrosion resistance of the stainless steels generally increases with increasing chromium content. There has been some speculation as to why chromium should impart stainlessness to steel. The popular concept is that when sufficient chromium is present, a thin, tight, chromium oxide is formed on the surface and this oxide prevents any further oxidation or corrosion. Environments which are oxidizing in nature strengthen this film while reducing environments tend to break down the film and cause the steel to corrode.

This theory of passivation by an oxide film now has considerable experimental support. Films were first stripped by Evans⁽¹⁷⁾ and his co-workers. More recently, Rhodin⁽¹⁸⁾ and his associates have isolated films to determine their structure and chemical composition. Rhodin's results show the films to be "jell-like" substances having no well-defined crystalline structure. A chemical composition corresponding to the approximate formula is:



where:

$$12\text{M} = 7 \text{ Fe} + 2 \text{ Ni} + 3 \text{ Cr}, n \text{ is approximately } 9.$$

Voluminous data have been published on the corrosion resistance of specific grades of stainless steel in specific environments. The data in the literature represent both controlled laboratory tests and actual service records. To discuss these corrosion data, the subject of corrosion is divided into four separate parts: atmospheric, elevated-temperature, intergranular, and pitting corrosion.

Atmospheric Corrosion—The most common type of corrosion encountered in steel is ordinary rusting. Recent investigations of the atmospheric corrosion of stainless steels have disclosed that ferritic steels above 12 per cent chromium are partially resistant, and above 18 per cent chromium are fully resistant to rusting.

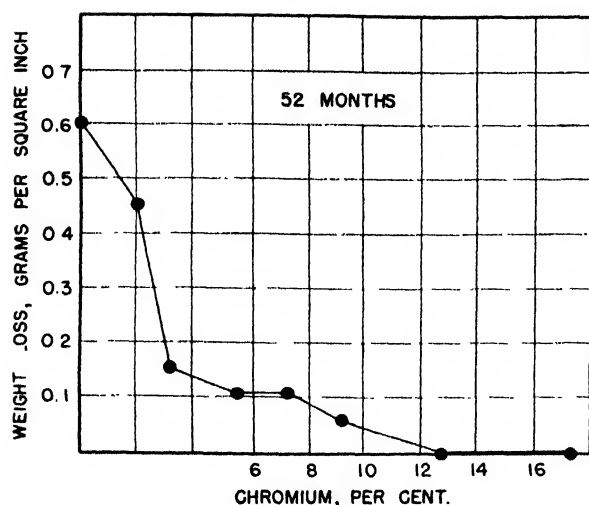


Fig. 47-12. The influence of chromium on the atmospheric corrosion of low-carbon steel.⁽¹⁰⁾

Figure 47-12⁽¹⁰⁾ shows graphically the effect of chromium content on the atmospheric corrosion of stainless steels. Noteworthy are the breaks in the curve at 3 per cent and 12 per cent chromium. Some variations from this behavior may be expected in different atmospheres and when the metal is in different conditions of heat treatment.

The austenitic steels also are very resistant to atmospheric corrosion. The results of a series of tests lasting over several years were reported in the 1946 Symposium sponsored by the American Society for Testing Materials. These results indicated that installations of 18-8 should be cleaned periodically especially in marine atmospheres where salt deposits might cause pitting. Of course, if the installation is fully exposed to the atmosphere, automatic cleaning occurs during rainfall, but installations which are fully or partially sheltered may be subject to pitting. An 18-8 stainless steel containing molybdenum was shown to have the best resistance to all conditions of corrosion.

The effect of surface was also studied and it was reported that the smoother the surface, the better the corrosion resistance.

The amount of work which has been done on the corrosion of stainless steels in various reagents is voluminous but, unfortunately, the data never have been properly organized. The effect of chromium on the resistance of steel to corrosion by these various reagents is similar to its effect on the resistance to atmospheric corrosion, although nickel and molybdenum enhance the resistance to corrosion by certain chemicals.

Elevated-Temperature Corrosion—Corrosion at elevated temperatures is usually one of three types: oxidation, carburization, and sulphidation.

As was true for atmospheric corrosion, the resistance of the stainless steels to oxidation is primarily a function of the chromium content. This effect of chromium on oxidation resistance is discussed in Chapter 48, entitled "Steels for Elevated-Temperature Service."

The chromium content also determines the rate of carburization. In general, ferritic steels are more resistant to carburization than austenitic steels. Addition of 2 per cent silicon improves resistance of 18-8 and 25-20 to carburization, but the effect of titanium, columbium (niobium) and molybdenum is negligible.

Usually, carburization rates are not appreciable below

1500° F in steels containing 18 per cent or more chromium. However, under special conditions, especially cyclic oxidation and carburization, 18-8 has been known to carburize at 1400° F.

The austenitic stainless steels, because of their nickel content, are susceptible to attack by reducing sulphurous gases, notably hydrogen sulphide. The ferritic stainless steels are more resistant.

Oxidizing sulphurous gases slightly increase the corrosion of the austenitic and the ferritic stainless steels.

These three types of elevated-temperature attack are very important but too often neglected when recommending materials. In many installations, elevated-temperature corrosion by oxygen, carbon, or sulphur causes more failures than stresses. Consequently, many of the steels such as 18-8, although possessing adequate strength, will not meet service requirements because of excessive oxidizing, carburizing or sulphidizing conditions.

Intergranular Corrosion—Bain, Aborn, and Rutherford⁽¹¹⁾ have ascribed intergranular corrosion to chromium impoverishment at the grain boundaries, caused by the grain boundary precipitation of chromium-rich carbides. There are two possible remedies for this difficulty. First, the carbon may be immunized, either by precipitating the carbide at a temperature where chromium diffusion will be rapid enough to restore passivity to the grain-boundary area, or by forming a stable carbide of some element other than chromium. Second, the steel may be made with less than 0.03 per cent carbon.

Probably the first method to be used commercially was that of forming a stable carbide of some element other than chromium. Bain, Rutherford and Aborn suggested the addition of titanium in amounts of four to five times the carbon content. By heating the titanium-bearing 18-8 at 1600° F, they obtained the complete fixing of carbon by titanium to form titanium carbide, and the steel was said to be stabilized. Concurrent with this development, the Union Carbide and Carbon Corporation developed a columbium-bearing (niobium-bearing) steel having a columbium content of eight times the carbon content. The columbium, like titanium, formed a stable carbide and thus prevented the formation of chromium carbide at elevated temperatures.

During World War II, United States Steel Corporation developed a method for producing 18-8 and 18-8 containing molybdenum, both containing 0.03 per cent maximum carbon. These "new" steels are more resistant to carbide precipitation and the resultant susceptibility to intergranular corrosion than regular grades of these steels, because of their lower carbon contents. These new grades, designated as Types 304L and 316L have been accepted commercially as replacements for the columbium- and titanium-bearing grades for service at temperatures below about 800° F.

Pitting Corrosion—Pitting usually occurs during continuous exposure to relatively-weak corroding media, such as chlorides, to which the steels are otherwise substantially resistant. Very little is known about the cause of this type of corrosion except that it probably occurs in certain vulnerable spots where the passivity is continuously destroyed. Pitting manifests itself by small holes distributed at random over the surface of the steel, or it may develop at points where the steel is in contact with other materials, such as leather, glass, and grease. The formation of an electrolytic cell may be the cause. The mechanism probably involves a lack of oxygen on the surface of the metal that is corroded.

Pitting may be inhibited somewhat by treating the environments with some strong oxidizing agents such

STAINLESS STEELS

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Table 47—III. Representative Mechanical Properties of Some Grades of Stainless Steel*

Type	Form and Treatment	Tensile Strength (Lb. per Sq. In.)	Yield Strength (Offset: 0.2 Per Cent) (Lb. per Sq. In.)	Elongation in 2 In. (Per Cent)	Reduction of Area (Per Cent)	Impact Resistance, Ft.-Lb.		Hardness Number	Endurance Limit (Fatigue) (Lb. per Sq. In.)	Cold Bend (Degrees)	Erichsen Value (mm.)	Olsen Value (In.)
						Charpy	Izod					
301	Sheet and Strip Annealed: 1/4 Hard to Full Hard: Plate Annealed:	110,000	40,000	60	35,000	180	10 to 14	0.4 to 0.5
		125,000 to 185,000	75,000 to 140,000	25 to 9	80,000 (Full Hard)
		105,000	40,000	55	70	110	165	39,000
410	Sheet and Strip Annealed: Wire Soft Temper: Tempered and Cold Drawn: Heat Treated: Plate Annealed: Bars Annealed: Tempered: Tempered and Cold Drawn:	65,000	35,000	25	180	7 to 8	0.3 to 0.4
		75,000	40,000	30	70
		95,000 to 135,000	80,000 to 105,000	15 to 10	60 to 50
		70,000	35,000	30	150	180
430	Sheet and Strip Annealed: Wire Soft Temper: Bright Soft Temper: Plate Annealed: Bars Annealed: Annealed and Cold Drawn:	75,000	45,000	25	180	7 to 9	0.3 to 0.4
		70,000	40,000	35	70
		85,000	70,000	15	70
		75,000	40,000	30	160	180
	Annealed: Annealed and Cold Drawn:	75,000	45,000	30	65	155	40,000
		85,000	70,000	20	65	185	46,000

*Ranges from individual data in American Iron and Steel Institute "Steel Products Manual, Section No. 24" (1950).

as some chromates or phosphates. Molybdenum additions to 18-8 also inhibit pitting.

MECHANICAL PROPERTIES

Low-Temperature Properties—At extremely low temperatures, the ferritic and martensitic steels are quite brittle, having impact strengths on the order of one foot pound. However, the austenitic stainless steels have impact strengths above 15 foot pounds at minus 320° F and are the best materials for use at the extremely low temperatures.

Room-Temperature Properties—The range of room-temperature mechanical properties of each grade of stainless steel is given in the AISI Steel Products Manual.⁽¹⁾ Table 47—III shows typical properties of Type 301 (18-8), Type 410 (12 per cent chromium), and Type 430 (17 per cent chromium).

As was mentioned, the martensitic grades are hardenable, and by proper selection of composition and heat treatment, their tensile properties can be made similar to those of the low-alloy steels. Ductility values and impact strengths are somewhat inferior to those of the low-alloy steels.

The ferritic grades cannot be hardened to any extent by heat treatment, although the 885° F embrittlement involves hardening. Table 47—III shows typical properties of 17 per cent chromium, Type 430. These steels are extremely notch sensitive. The notch sensitivity, or rather the transition temperature from brittle to ductile behavior, increases with chromium content, the 27 per cent chromium steel having a Charpy impact strength of only two foot pounds at room temperature. This notch sensitivity is undesirable, and when fabricating these steels, special precautions must be taken. Because of the notch sensitivity, aggravated by large grain size and 885° F embrittlement, both of which may result from welding, the ferritic steels have not found as wide usage at room temperatures as have the austenitic steels. However, the ferritic steels have as good corrosion resistance as the austenitic steels, and, if it were not for the disadvantage in ductility, they would be used almost as much as the more expensive austenitic steels.

The austenitic steels are not hardenable by heat treatment but they can be hardened by cold work which for some grades transforms the austenite (metastable at room temperature) to a low-carbon martensite. This transformation hardening by cold work is additive to the

strain hardening and the result is that an extremely wide range of tensile properties may be produced. As an example, Table 47—III shows typical properties of Type 301 (17-7). As would be expected, the degree of hardening obtained by cold work is a function of the composition, and extensive investigations have been made on the effect of various elements on the work hardenability of 17-7.^(1a)

The disadvantage of the austenitic stainless steels is that the work hardening is confined to those shapes which can be rolled (cold worked). For example, an article such as a valve cannot be hardened economically by cold working. This disadvantage was overcome by the development of an age-hardening stainless steel designated as Stainless W.^(1a)

Stainless W is essentially a 17 per cent chromium, 7 per cent nickel steel containing titanium and aluminum. As would be expected, this is an "unbalanced" composition. At high temperature, its structure consists of austenite and delta ferrite. On cooling, the austenite transforms at about 200° F (93° C) to low-carbon martensite, so that the room-temperature structure of this alloy consists of martensite plus delta ferrite and some retained austenite. The age hardening occurs in the martensite and in the ferrite but not in the austenite. Aging is accomplished by heating in the temperature range 900° to 1050° F. Table 47—IV shows the typical properties of Stainless W in the annealed and aged conditions.

Elevated-Temperature Properties—In Figure 47—13 is shown the creep strength of many of the commercial stainless steels, compared with that of carbon steel. Note that the austenitic stainless steels have higher strength than the ferritic stainless steels and that additions of chromium and silicon have little effect on the creep strength. On the other hand, the effect of molybdenum, columbium (niobium), and tungsten are marked. This phase of the subject is discussed in detail in Chapter 48.

APPLICATIONS

Martensitic Grades—The most widely used of the martensitic grades is Type 410 containing under 0.15 per cent carbon and 11.50 to 13.50 per cent chromium. In the annealed condition, this grade may be drawn or formed. As it is an air-hardening steel, a wide range of mechanical properties may be obtained by heat treatment. In sheet or strip form, Type 410 is used extensively in the oil industry for bubble trays, bubble caps and liners.

Table 47—IV. Typical Ranges of Mechanical Properties of Stainless W, as Solution Annealed and as Aged at the Temperatures Indicated

Item No.	Treatment	Yield Strength 0.2% Offset (1000 Lb. per Sq. In.)	Tensile Strength (1000 Lb. per Sq. In.)	* Elongation (% in 2 In.)		Rockwell Hardness "C" Scale
				½ In. and Less	Over ½ In.	
1	Solution Annealed at 1850 to 1950° F, Air Cooled.	75-115	120-150	8-14	10-15	22-28
2	No. 1 plus Aged at 950° F, ½ hour, Air Cooled.	180-210	195-225	8-14	10-15	39-47
3	No. 1 plus Aged at 1000° F, ½ hour, Air Cooled.	170-210	190-220	8-14	10-15	38-46
4	No. 1 plus Aged at 1050° F, ½ hour, Air Cooled.	150-185	170-210	8-15	10-16	35-43

* For thicknesses of plates and bars indicated.

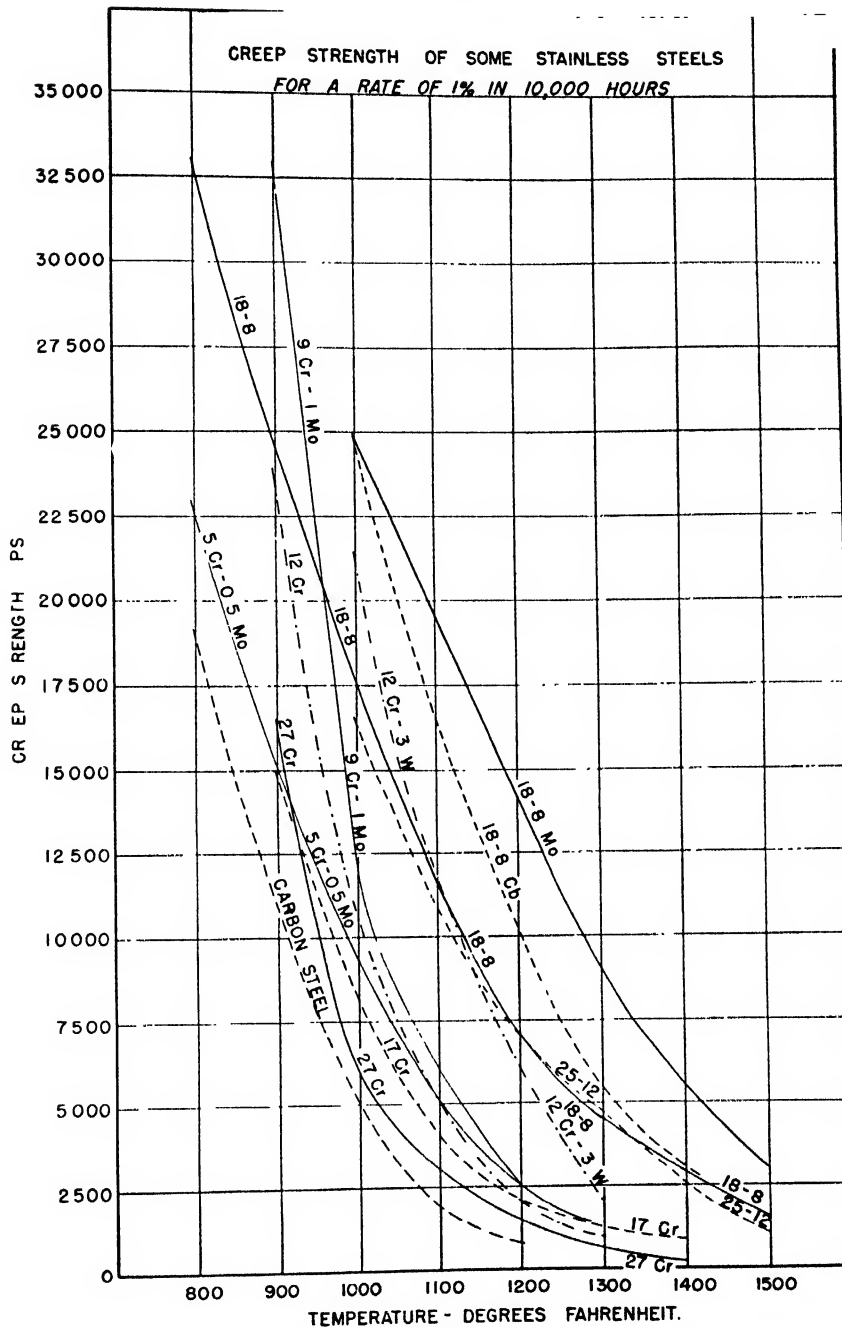


FIG. 47-13. The creep strength of some stainless steels.

It is also used for furnace parts where the operating temperature is not over 1200° F and for blades and buckets in steam turbines. Type 420, with about 0.35 per cent carbon and resultant greater hardness, is used for cutlery. In bar form it is used for valves, valve stems, valve seats, and shafting where resistance to corrosion and wear are needed. Type 440 may be employed for such applications as surgical instruments, especially those requiring a durable cutting edge. The necessary hardness for different applications can be obtained by selecting Grade A, B, or C, which have increasingly greater carbon contents, in that order.

Types 410 and 420 are usually not used for welding applications because of their hardening characteristics. However, as mentioned previously, the addition of 0.1 to 0.3 per cent aluminum to Type 410 decreases its hardening tendency and makes it easier to weld.

Other martensitic grades are Types 501 and 502, the former having carbon over 0.10 per cent and the latter having under 0.10 per cent carbon, but both containing 4 to 6 per cent chromium. These grades are also air-hardening, but do not have the corrosion-resisting properties of the 12 per cent chromium grades. These grades have wide application in the oil industry for such uses

as hot-oil lines, bubble towers, valves, plates, and so on.

Ferritic Grades—The most common and widely used of the ferritic grades is Type 430 containing 0.12 per cent carbon or under and 14 to 18 per cent chromium. The higher chromium content of Type 430 makes its corrosion resistance superior to that of the martensitic grades and, in addition, this material may be drawn, formed and, with proper techniques, welded. It is widely used for automotive and architectural trim. Its resistance to nitric acid makes it useful in the manufacture and handling of this acid. Type 430 does not have a high creep strength, but it is suitable for some types of service up to 1500° F and thus finds application for such parts as combustion chambers for domestic heating furnaces.

The high chromium content of Type 446 (23 to 27 per cent chromium) gives this grade excellent heat-resistant characteristics although its elevated-temperature strength is only slightly better than that of carbon steel. Type 446 is used in sheet or strip form at temperatures up to 2100° F. This grade does not have the good drawing characteristics of Type 430, but it may be formed. Accordingly, it is used widely for furnace parts such as muffles, burner sleeves and annealing baskets. Its resistance to nitric and sulphuric acids makes it suitable for much chemical-processing equipment, and tubular products of Type 446 were widely used during World War II by the synthetic-rubber industry for the processing of butane.

Austenitic Grades—The basic and most widely used grade of the austenitic types is "18-8"—Type 302, with 0.15 per cent maximum carbon, 8 to 10 per cent nickel and 17 to 19 per cent chromium. It has excellent corrosion resistance and, because of its austenitic structure, possesses very good ductility. It may be deep drawn and can be very severely formed. It can be welded readily, but, as described in the general section on iron-chromium-nickel alloys, the heat of welding may cause carbides to precipitate in the weld and in the metal adjacent to the weld if a sufficiently rapid rate of cooling is not obtained, thus rendering these zones susceptible to intergranular corrosion. This may be corrected by annealing the welded part above 1900° F to redissolve the carbides, followed by rapid cooling to retain them in solution. Where such a treatment is not feasible, Types 321, 347, or 18-8 with 0.03 per cent maximum carbon may be used.

The applications of Type 302 are wide and varied, including kitchen equipment and utensils; dairy installations; transportation equipment; oil-, chemical-, paper- and food-processing machinery.

Type 301 contains a maximum of 0.15 per cent carbon, 6 to 8 per cent nickel and 16 to 18 per cent chromium. Its lower nickel content causes it to work harden more rapidly than Type 302 (18-8) because of reduced austenite stability. Accordingly, it does not have quite as good drawing properties as Type 302, but for the same reason, this grade (Type 301) can be cold rolled to very high strength levels for use in applications where a high strength-to-weight ratio is desired.

Type 304 is a companion grade to Types 301 and 302. Its carbon content is somewhat lower, 0.08 per cent maximum, and it has a slightly higher chromium content, 18 to 20 per cent. Because of its lower carbon content it is not so prone to give trouble after welding due to carbide precipitation and resultant corrosion. In addition, the somewhat higher chromium content makes it slightly more resistant to general corrosion. It is well suited for those applications which require resistance to severe forms of corrosion such as are encountered in the paper and chemical industries. The austenitic grades have good elevated-temperature strength and these

grades are widely used for elevated-temperature service.

Types 321 and 347, with carbide-forming additions of titanium and columbium (niobium), respectively, are widely used in those applications involving welding and where high-temperature service under corrosive conditions is required. Type 304L, the 0.03 per cent maximum carbon grade, may be used as an alternative for Types 321 and 347 in those applications involving welding and stress relieving for service below about 800° F.

The addition of 2 to 4 per cent molybdenum to the basic 18-8 composition produces Types 316 and 317 which have improved corrosion resistance. These grades are used in applications in the textile, paper and chemical industries where strong sulphates, chlorides and phosphates and such reducing acids as sulphuric, sulphurous, acetic and hydrochloric acids are used in such concentrations as to make the use of a more highly corrosion-resistant alloy mandatory. Types 316 and 317 have the highest creep and rupture strengths of the commercial stainless steels.

The austenitic stainless grades most resistant to oxidation are Types 309 (22 to 24 per cent chromium, 12 to 15 per cent nickel) and Type 310 (24 to 26 per cent chromium, 19 to 22 per cent nickel). Because of their high nickel and chromium contents these steels resist scaling at temperatures up to 2000 and 2100° F and, consequently, are ideal for furnace parts and heat exchangers. They are somewhat harder and not as ductile as the 18-8 types, but they may be drawn and formed. They can be readily welded and are finding increasing use in the manufacture of jet-propulsion motors and industrial-furnace equipment.

For applications requiring good machinability, Type 303 containing sulphur or selenium may be used.

SUMMARY

The stainless steels are alloys of iron and chromium or iron, chromium, and nickel, although occasionally small amounts of certain other elements are added in order to enhance corrosion resistance and mechanical properties or to immunize the steels to the action of certain harmful impurities. The inherently slow reaction rates of the stainless steels have hampered the establishment of precise equilibrium diagrams; however, the diagrams now in existence permit at least qualitative conclusions to be drawn regarding the structure of these steels. In regard to corrosion resistance, the chromium content seems to be the controlling variable and the effect of chromium may be enhanced by additions of molybdenum, nickel, and other elements. The mechanical properties of the stainless steels, like those of the plain carbon and lower alloy steels, are functions of the structure and composition of the material. Thus, the austenitic steels possess the best impact properties at low temperatures and the best strength at elevated temperatures while the martensitic steels possess the highest hardness at room temperature. Therefore, the stainless steels, by being available in a variety of structures, exhibit a range of mechanical properties which, combined with their excellent corrosion resistance, makes these steels highly versatile from the standpoint of design.

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Chapter 48

STEELS FOR ELEVATED-TEMPERATURE SERVICE

SECTION 1

CLASSES OF STEEL

The designation "elevated-temperature service" is an inclusive one, involving many operations and industries. Examples of equipment operated at high temperatures are steam and mercury boilers and turbines, gas turbines, stills for cracking petroleum, tar stills, vessels for hydrogenating oils, heat-treating furnaces, and fittings for Diesel and internal-combustion engines. Jet-propelled planes and guided missiles powered by gas turbines or rocket motors are introducing new requirements for metals with even greater strength at elevated temperature, and are increasing the complexity of problems in high-temperature design, both in the motors and in the fuselage. Valve steels, hot-working die steels, and high-speed tool steels are also used at elevated temperature, but these form special categories and are discussed elsewhere in this book.

The steels under consideration in this section are those in commercial use in large tonnages for construction of equipment to operate under stress at elevated temperature, where creep is involved. (Creep and creep testing are discussed in detail in the section on "Creep and

Rupture Testing" in Chapter 49.) In spite of the fact that plain carbon steel has the lowest strength, it is widely used for elevated-temperature applications up to 1000° F, where scaling commences and a chromium-bearing steel must be employed. Low-alloy steels, containing small amounts of chromium and molybdenum, have higher creep strength than carbon steel and are employed where material with higher strength is needed. Above 1000° F, the amount of chromium required for scaling resistance increases rapidly. The 5 per cent chromium steels, with added molybdenum, are useful up to 1150° F, and steels containing 10 to 14 per cent chromium may be employed up to 1250° F. Above this temperature, the austenitic 18 per cent chromium, 8 per cent nickel stainless steels are used customarily, and their scaling resistance is considered adequate up to 1600° F. For service in the temperature range between 1600° and 2000° F, steels containing 25 per cent chromium and 12 per cent nickel, 25 per cent chromium and 20 per cent nickel, or 27 per cent chromium are used.

SECTION 2

FACTORS AFFECTING HIGH-TEMPERATURE PROPERTIES

Composition and Temperature—The creep strength of plain carbon steel can be greatly improved by the addition of alloying elements. It has been found that creep strength may be increased by use of those elements such as molybdenum which increase the recrystallization temperature and form stable carbides or intermetallic compounds. The effect of molybdenum is shown in Figure 48—1 and the effect of chromium is shown in Figure 48—2. Small additions of chromium do not appear to improve the creep strength of plain molybdenum steels and higher additions actually decrease the strength. However, above 1000° F, chromium is needed for surface protection.

Carbon itself is beneficial in amounts to about 0.15 or 0.20 per cent; above this amount, increase of the carbon content results in decrease of creep strength. Tungsten and vanadium act in a manner similar to molybdenum and are quite useful in improving high-temperature strength although they are not as widely used in this country. Titanium and columbium (niobium) are moderately beneficial and a slight increase in phosphorus has been found desirable. The influence of manganese, nickel, copper and silicon is mild, and aluminum decreases creep strength. It has been found that the face-centered cubic structure of austenite is more creep resistant than the body-centered cubic structure of ferrite, and the austenitic chromium-nickel steels have excellent creep strength; i.e., when used as a means of changing

the lattice structure, nickel is very beneficial, and nitrogen may act in a similar manner.

The effect of the alloying additions on creep strength is summarized in Figure 48—3, which shows the decrease of strength with increase of temperature. For the sake of simplicity and comparison, this chart is based on average properties, but actually the strength of any particular steel at a given temperature comprises a range rather than a single value. The spread of the data is due partly to the fact that all steels are purchased under specifications which permit some latitude in composition, and partly to the other causes such as variation of heat treatment and manufacturing practice to be discussed in detail below.

Heat Treatment (Microstructure)—The strength of steel at elevated temperatures is profoundly affected by its microstructure, and hence by the heat treatment employed for the material. Carbon and molybdenum steels are used in the as-rolled, normalized, or annealed condition, while the air-hardening chromium-molybdenum steels are normalized and tempered or annealed. Steels for bolts are quenched and tempered. Stainless steels are annealed or annealed and stabilized. Steels are purchased to specifications having room-temperature mechanical strength and hardness requirements. Since these properties may be produced by several of the above heat treatments, few of the various types of steel are used in only one condition of heat treatment, ac-

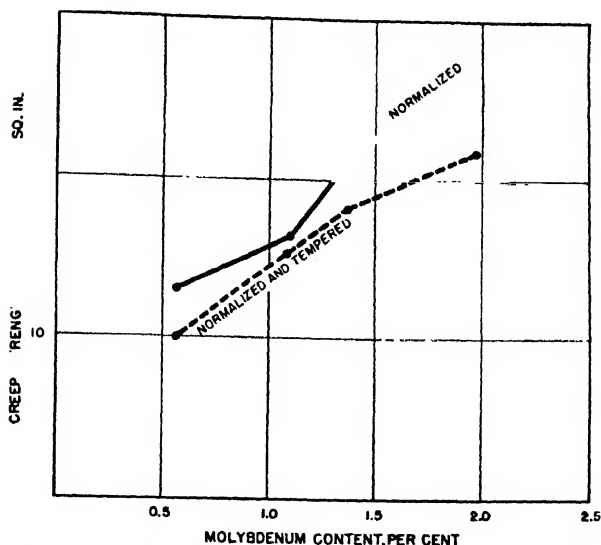


FIG. 48-1. Effect of molybdenum content on creep strength (stress per creep rate of 1 per cent per 10,000 hours) of molybdenum steel at 1000° F.

counting for much of the spread in the high-temperature data. It should be borne in mind that specifications do not contain requirements concerning the properties of the steels at elevated temperature.

The effect of heat treatment on the creep strength of molybdenum steel was clearly shown by Weaver. In creep tests at 1000° F, he found that the stress to produce a creep rate of 1 per cent per 10,000 hours varied from 20,000 to 4,500 lb. per sq. in., depending upon whether the structure was pearlitic or spheroidized.

The temperature at which the steel is used in service may serve as a heat-treating temperature and bring about internal changes in the material which will cause a change of its properties. Sometimes the strength and hardness are found to increase, at least temporarily, but more often the lack of structural stability causes a progressive loss of strength. There are several forms in which structural instability makes itself manifest, such as spheroidization, graphitization, precipitation hardening or aging embrittlement, and carbide precipitation. These will be discussed in the section entitled "Behavior of Steels at Elevated Temperature."

Grain Size—It has been found that the creep strength is increased somewhat by a coarsening of the grain size. It is not possible to coarsen the grains unduly without injuring other properties such as impact strength, but a grain size of 2-5 instead of the customary 6-8 has been found to be beneficial.

Cold Working—The long-time creep strength of cold-worked material will generally be lower than that of annealed material free from strain hardening. However, for short time periods of a few hours up to several hundred hours at moderately elevated temperatures, the strengthening effect of cold working may persist long enough to be useful in certain short-time applications.

Manufacturing Practice—The creep behavior of a material is a very sensitive index of its strength and is strongly affected by differences in chemical composition and microstructure. However, even for materials of similar composition and microstructure, the creep strength is not always the same, but is affected by the method by which the steel is manufactured. For example, pearlitic carbon steel under a stress of 15,000 lb. per sq. in. at 850° F shows a thousandfold difference in creep rate de-

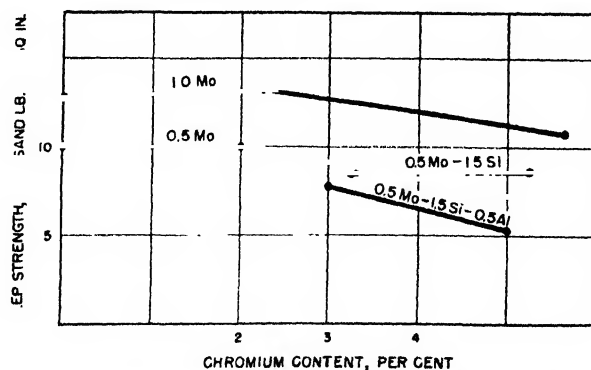


FIG. 48-2. Effect of chromium content on creep strength (stress for creep rate of 1 per cent per 10,000 hours) in chromium-molybdenum steel at 1000° F.

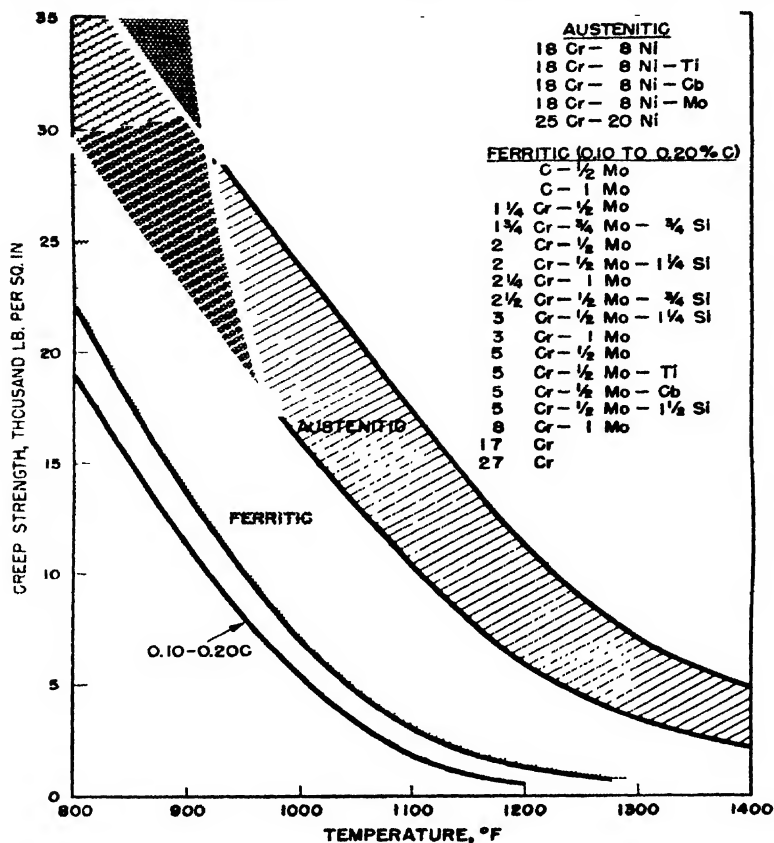


FIG. 48-3. Creep strength (stress for creep rate of 1 per cent per 10,000 hours) of various steels at temperatures between 800° and 1400° F.

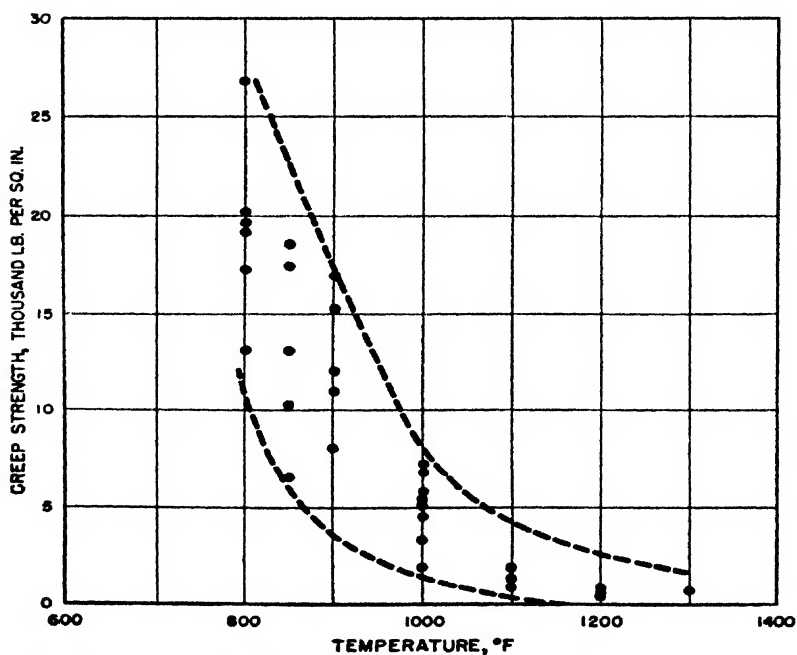


FIG. 48-4. Creep strength (stress for creep rate of 1 per cent per 10,000 hours) of carbon steel at temperatures between 800° and 1300° F.

pending upon the manner in which it was manufactured.

The variation of the available data on the creep strength of carbon steel is shown in Figure 48—4. At 1000° F, for example, the creep strength varies from 2,000 to 7,000 lb. per sq. in., and at 800° F from about 12,000 to 27,000 lb. per sq. in. Part of this variation in strength is, of course, due to variation in grain size and microstructure. However, the method of deoxidation has been found to have an important effect on the creep strength. For example, molybdenum steels deoxidized with ½ lb. and 1½ lbs. of aluminum per ton were tested

for creep strength at 850° and 1000° F. The steels were heat treated to produce identical grain size and microstructures, eliminating these variables. It was found that at 1000° F, the low-aluminum steel was more creep resistant than the high-aluminum steel, indicating that the aluminum content itself affected the behavior of these materials. The same behavior has been noted in carbon steels.

Electric-furnace steels have been reported to have a higher creep strength than open-hearth steels of the same type.

SECTION 3

BEHAVIOR OF STEELS AT ELEVATED TEMPERATURE

A. INTERNAL STABILITY

Carbide Instability—Steels in service at elevated temperature do not usually show any significant change of grain size or shape, but both visible and invisible changes may occur within the grains. Visible changes within the grains may be classified as formation of new phases, spheroidization and migration of carbides and nitrides, and graphitization.

In carbon and molybdenum steels, spheroidization occurs at temperatures above 1000° F; as shown in Figure 48—5, graphitization, particularly in fine-grained aluminum-killed steel, may also occur. The addition of chromium tends to stabilize the carbides, thus lessening or eliminating graphitization and delaying spheroidization. The relative stability of 24 steels at 900°, 1000°, and 1100° F, is shown in Table 48—I.

A form of carbide instability occurs in the austenitic chromium-nickel steels in the temperature range between 800° and 1600° F. In this temperature range, chromium carbides form and precipitate at the grain boundaries, depleting these regions in chromium and hence lowering the resistance of the grain boundaries to corrosion. This form of carbide precipitation has not been found to lower the tensile, creep or impact strength of the material, which has been successfully used for oil-cracking still tubes in this temperature range. Elimination of the carbide precipitation is accomplished by lowering the carbon content below 0.02 per cent or

alloying with such carbide stabilizers as titanium or columbium (niobium).

Aging—Aging or precipitation hardening as a heat-treating process may be utilized to increase the strength of metals at elevated temperature providing that the material is used below the temperature at which over-aging would occur during life of the part. For example, in age-hardened Stainless W, an alloy containing 17 per cent chromium, 7 per cent nickel with added titanium, relaxation measurements show that, while some softening occurs in about a week at 800° F, very little softening occurs in the same length of time at 600° F. Below this temperature, therefore, it should be possible to utilize the strengthening produced by the precipitation hardening.

Aging can, of course, occur during service and may result in embrittlement of the material. In carbon steels, it has been found that aging occurring at temperatures between 400° and 600° F is more pronounced in Bessemer steels than in those made in the open hearth, and that it is greater the less thorough the deoxidation or killing of the material. The aging tendency is lessened by addition of titanium and aluminum. If this aging or precipitation hardening resulted only in an increase of strength, no harm would result. However, as the strength increases, the ductility decreases, particularly as measured by the notched-bar impact test after straining. The sensitivity to strain-aging is lessened by normalizing and by thorough deoxidation of the steel.



FIG. 48—5. (Left) Spheroidized annealed molybdenum steel. (Right) Same steel exhibiting graphitization after heating at 1100° F for 1000 hours. Magnification (both photomicrographs): 1000X.

Table 48—1. Relative Change of Microstructure* of Alloys During 3000 Hours at 900°, 1000°, and 1100° F.

Nominal Composition	Heat Treatment**	Change in Structure† in 3000 Hours at		
		900° F	1000° F	1100° F
C-0.5 Mo	1650 AC	I	III	III
C-0.5 Mo	1650 AC, 1300 AC	0	II	III
C-1 Mo	1650 AC	0	III	—
C-1 Mo	1650 AC, 1300 AC	0	II	—
C-1.5 Mo	1650 AC	0	III	—
C-1.5 Mo	1650 AC, 1300 AC	0	II	—
C-2 Mo	1650 AC	0	III	—
C-2 Mo	1650 AC, 1300 AC	—	II	—
C-Mo-Mn	1650 AC, 1300 AC	0	II	—
2 Cr-0.5 Mo	1650 AC, 1380 FC	—	I	—
2.25 Cr-1 Mo	1650 AC, 1380 FC	—	0	II
2.25 Cr-1 Mo	1650 FC	—	I	—
1.2 Cr-0.5 Mo-0.7 Si	1650 AC, 1380 FC	—	0	II
1.7 Cr-0.7 Mo-0.7 Si	1750 AC, 1375 FC	—	I	—
3 Cr-0.5 Mo-1.5 Si	1600 FC	—	0	0
3 Cr-Mo-Si-Al	1550 AC	—	0	0
5 Cr-0.5 Mo	1600 FC	—	0	0
5 Cr-0.5 Mo	1600 AC, 1380 AC	—	0	I
5 Cr-1 Mo	1650 AC, 1380 AC	—	0	I
5 Cr-1 Mo	1650 FC	—	0	—
5 Cr-0.5 Mo-1.5 Si	1600 FC	—	0	0
5 Cr-Mo-Si-Al	1550 AC	—	0	0
5 Cr-Mo-Ti	1380 AC	—	I	—
5 Cr-Mo-Ti	1850 AC	—	0	—
5 Cr-Mo-Cb	1380 AC	—	I	—
5 Cr-Mo-Cb	1850 AC	—	0	—
5 Cr-Mo-Si-Cb	1750 AC, 1380 FC	—	0	—
8 Cr-1 Mo	1700 FC	—	0	0
8 Cr-1 Mo-Cb	1700 FC	—	0	0
9 Cr-1 Mo	1700 FC	—	0	0
18-8	1900 WQ	—	—	II
18-8 Ti	1900 AC	—	—	I
18-8 Ti	1900 AC, 1600 AC	—	—	0
18-8 Cb	1900 AC	—	—	I
18-8 Cb	1900 AC, 1600 AC	—	—	I

*It may be assumed that steels showing a change of microstructure at 1000° F would do so at a higher temperature, and that steels which remained unchanged at 1000 or 1100° F would do likewise at a lower temperature.

**AC = Air Cool FC = Furnace Cool WQ = Water Quench
†0 —no change. II —moderate change.
I —slight change. III—marked change.

Temper Brittleness—Steels with even moderate amounts of chromium or manganese, and such steels plus nickel, are likely to become brittle if cooled slowly from the tempering operation after quenching—hence the name, “temper brittleness.” Steels with 1 to 2 per cent manganese, and even plain 3 to 5 per cent nickel steels, may show it. The addition of 0.30 to 0.50 per cent Mo lessens or prevents the embrittlement. The higher the nickel and chromium content, the greater the amount of molybdenum required to increase freedom from embrittlement, and an increase in the nickel content decreases the resistance to embrittlement of steels containing chromium and molybdenum. The brittleness resulting from heating at about 840° F can be eliminated by heating to between 930° and 1110° F. Embrittlement can occur under no load, but stress has been found to accelerate the embrittlement.

Temper brittleness is found in bolts, studs, and other quenched and tempered parts used in high-temperature steam lines at temperatures in the vicinity of 800° to 900° F. The embrittled material may retain its full original toughness at the service temperature, and yet be brittle to a notch test at room temperature. The fundamental cause of the embrittlement is not known, but is thought to be due to some form of precipitation hardening. There is no measurable difference between the tough and the embrittled forms of the steel observable in tensile, bend or fatigue tests, provided that the test pieces are not notched. Vanadium has also been found to minimize the effect of this embrittlement, and chromium-molybdenum or chromium-manganese-vanadium steels have been widely adopted for heat-treated fastener applications.

Embrittlement of Ferritic Chromium Steels—Plain chromium steels containing more than 12 to 15 per cent chromium have been found to become embrittled during prolonged heating in the vicinity of 900° F. This embrittlement is due to the formation or precipitation of a body-centered-cubic iron-chromium compound containing 70 to 80 per cent chromium. The embrittlement during heating at higher temperatures (1120° to 1470° F) is due to the precipitation of the sigma phase, an iron-chromium compound, FeCr. The ductility and impact properties of alloys containing over 20 per cent chromium may be seriously impaired even by slow cooling in the range 1100° to 700° F.

B. EXTERNAL OR SURFACE STABILITY

Scaling and Corrosion Resistance—The property of surface stability or corrosion resistance is considered to be of primary importance, since the metal must not deteriorate excessively during service at elevated temperature. One of the simplest forms of corrosion, and one frequently encountered, is oxidation or scaling of the metal. Scaling occurs by a process of diffusion of oxygen inward and of alloying elements outward through the scale layer already formed. In plain carbon steel, the amount of scaling in air is negligible below 1000° F. Above this temperature, the rate of scaling of carbon steel increases rapidly. For a given period of exposure, the amount of scaling varies exponentially with respect to the reciprocal of the absolute temperatures.

The most important element for increasing the scaling resistance of carbon steel above 1000° F is chromium. This element appears to oxidize preferentially to iron, and to form a tightly adherent layer of chromium-rich iron oxide on the surface of the metal, retarding the inward diffusion of oxygen and inhibiting further oxidation. Other elements such as silicon and aluminum also increase the scaling resistance, particularly when added to a steel containing chromium. These elements have

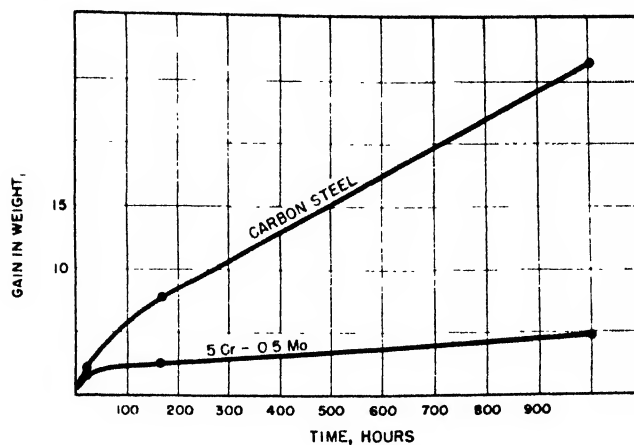


FIG. 48-6. Oxidation of plain carbon and 5 per cent chromium—0.5 per cent molybdenum steel at 1100° F.

a greater affinity for oxygen than does iron, and are also preferentially oxidized.

The rate of scaling decreases as the scale becomes thicker and additional protective layers are formed. The nature of the progress of scaling with time at 1100° F is shown in Figure 48-6 for carbon steel and 5 per cent chromium steel containing molybdenum. At the start, the rate of scaling of the alloy steel is as rapid as that of the carbon steel or perhaps more so, but the rate of scaling of the alloy steel soon decreases while that of the carbon steel continues at a rapid rate. The results of oxidation tests are shown in Figure 48-7. In these tests, the amount of scaling (oxidation) was measured by the gain in weight in 1000 hours. Several interesting points immediately become apparent. As the temperature is increased above 1000° F, the amount of scaling

in the plain carbon and molybdenum steels increases rapidly, and increase of chromium to 2.25 per cent or of silicon to 0.75 per cent does not improve the scaling resistance significantly. The 5 per cent chromium steels are somewhat better, but their scaling resistance decreases rapidly above 1200° F. Additions of 9 per cent or 12 per cent chromium considerably improves the scaling resistance, these materials showing little scaling in 1000 hours below 1400° F. The increase in scaling resistance in going from 5 per cent to 9 per cent chromium is noticeable. The very large increase in scaling resistance produced in 3 per cent chromium or 5 per cent chromium steel by addition of 1.5 per cent silicon is also striking.

A survey of the available data indicates that the chromium content for freedom from scaling is roughly

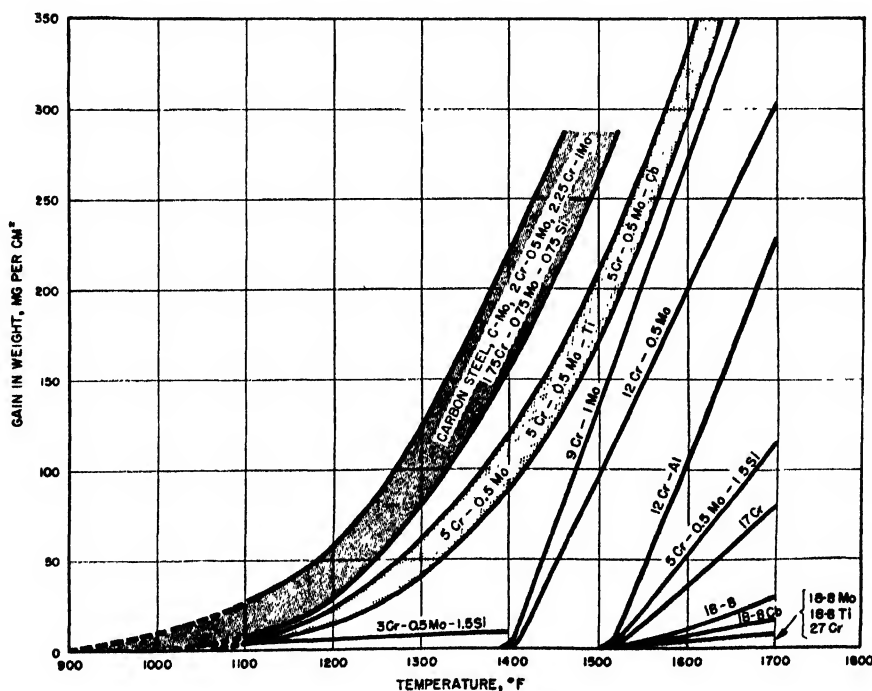


FIG. 48-7. Amount of oxidation (scaling) of carbon, low-alloy, and stainless steels in 1000 hours in air at temperatures from 1100° to 1700° F.

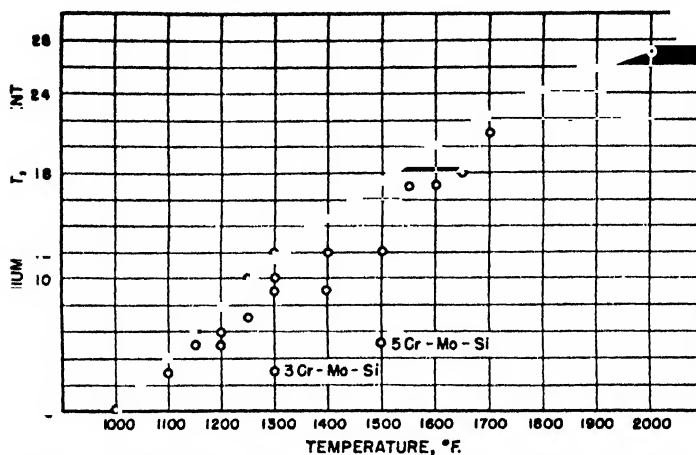


Fig. 48-8. Maximum amount of chromium necessary for freedom from scaling at temperatures from 1000° to 2000°F.

proportional to the temperature, as shown in Figure 48-8. While this diagram indicates the maximum chromium content for freedom from scaling, it should be noted that addition of 1.25 to 1.50 per cent silicon greatly improves scaling resistance, decreasing the amount of chromium needed for protection at a given temperature.

Effect of Various Atmospheres—The corrosion of steels at elevated temperatures in air is not necessarily indicative of their performance in other atmospheres. However, it is generally true that the corrosion resistance of steel increases with its chromium content. The precise behavior must, however, be established under the conditions in which the material will be used in service. Based on service experience in various atmospheres, as well as on laboratory tests, the American Society for Testing Materials has listed the maximum temperature which various stainless steels can withstand without excessive scaling (Table 48-II).

In 3000 and 7000 hour tests in steam at 1100° F, Van Duzer and McCutchan found that the silicon content of

the chromium-molybdenum steels did not increase their resistance to steam corrosion, as had been expected from laboratory scaling tests in air. They also found that the corrosion of the plain carbon steels proceeded at a constant rate, but that as little as 1 to 2 per cent chromium lessened materially the amount of scaling. In the 12 per cent chromium stainless steels, the high-sulphur free-machining steel was as good as the standard grades, but addition of 2 per cent manganese lowered the corrosion resistance.

Rohrig, Van Duzer and Fellows tested various carbon and alloy steels in steam at 925° and 1100° F. In periods up to 15,000 hours at 925° F, the steels formed a thin, dense, tightly adherent layer of scale. The amount of scaling was very light and there was little difference between the various types of steels. At 1100° F, scaling of the carbon steels was continuous, but that of the alloy steels decreased after a period of time. In general, the same trends are shown by these tests in steam as by the scaling tests in air.

Scaling tests in carbon dioxide show about the same results as scaling tests in air.

In flue gases, Oertel and Landt found that the amount of scaling in 10 per cent chromium steels at temperatures between 1300° and 2000° F was three times as rapid as in air. Gases that are chemically reducing in nature increase the scaling rate over that obtained in air.

Chromium is particularly effective in increasing the resistance of steel to corrosion by sulphur compounds at elevated temperature, a condition frequently encountered in oil refining. Service experience with oil-cracking still tubes and evaporators has shown that under these conditions small additions of chromium are effective; 2 per cent chromium was found to reduce the amount of corrosion to $\frac{1}{3}$ that encountered in plain carbon steel.

In ammonia synthesis, Schiffer and Berlecken showed that the conditions at high temperatures and pressures lead to reactions between the hydrogen of the gas and the carbon of the steel to form methane (CH_4) and other hydrocarbons. At 390° and 750° F, at 50 and 75 atmospheres pressure, respectively, carbon steel and 1 per cent chromium steel with molybdenum showed loss of impact strength after service, while 5 per cent chromium steel with molybdenum was unaffected. The improvement of the resistance of steels to hydrogen must be accomplished by using lower carbon content, minimizing inclusions, and alloying with chromium, prefer-

Table 48-II. Maximum Temperature Without Excessive Scaling

Alloy Nominal Composition (%)	Steel Type No.	Maximum Temperature Withstood Without Excessive Scaling (° F)
4-6 Cr-Mo	502	1150
8-10 Cr	—	1200
10-14 Cr	410	1250
12-14 Cr	420	1200
14-18 Cr	440	1400
14-18 Cr	430	1550
23-30 Cr	446	2000
18-8	302	1650
18-8	303	1600
18-8	304	1650
25-12	309	2000
25-20	310	2000
18-8 Mo	316	1650
18-8 Ti	321	1650
18-8 Cb	347	1650

From: "Tables of Data on Chemical Compositions, Physical and Mechanical Properties of Wrought Corrosion-Resisting and Heat-Resisting Chromium-Nickel Steels," American Society for Testing Materials, Dec., 1942.

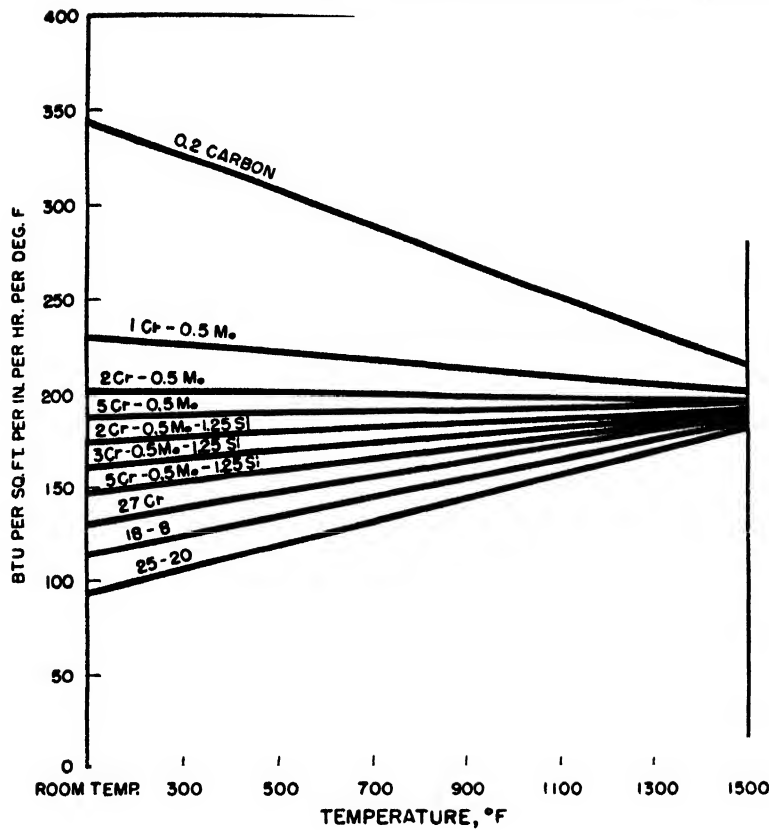
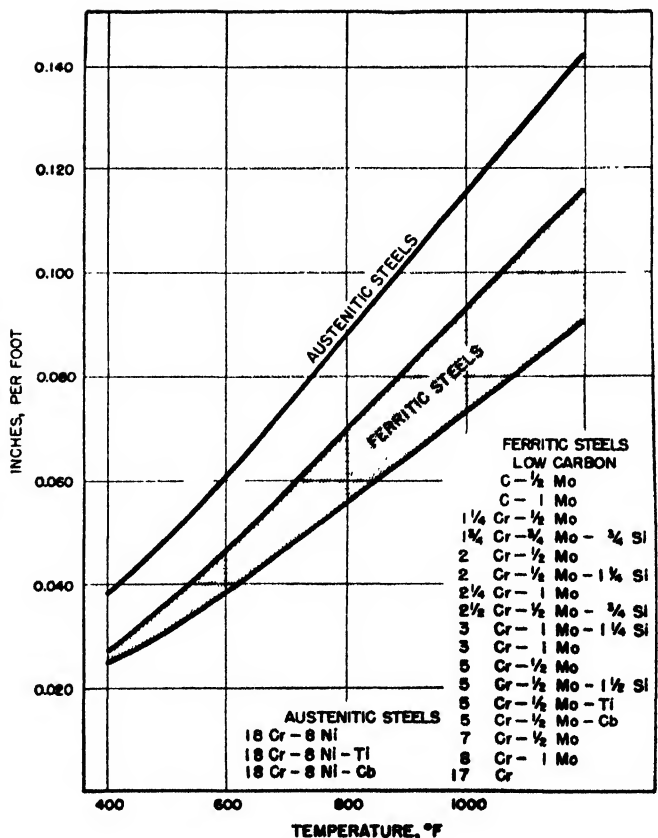


FIG. 48-9. Thermal conductivity of various steels at temperatures between room temperature and 1500° F.

FIG. 48-10. Influence of temperature on linear thermal expansion. The diagram shows the change of length, in inches per foot, occurring in carbon, low-alloy, and stainless steels as they are heated from room temperature to any temperature between 400° and 1200° F. For the ferritic steels, the thermal expansion decreases with increasing chromium content, following the order of the steels as listed on the diagram. In the austenitic steels the thermal expansion is larger than in the carbon steel.



ably above 3 per cent, in connection with such carbide stabilizers as tungsten, molybdenum, titanium, vanadium, tantalum, and columbium. The elements that resist hydrogen are good nitride formers, so failure can occur by formation of a brittle nitride layer. All high-chromium and chromium-nickel steels show excellent resistance to hydrogen. Chromium steels are resistant to hydrogen sulphide (H_2S) and sulphur dioxide (SO_2) gases, while nickel steels are sensitive to these gases. A chromium-silicon-aluminum steel was found to be useful where gases containing H_2S had to be handled at $1830^\circ F$.

Caustic Embrittlement of Boiler Plates—Caustic embrittlement or "caustic cracking" of boiler plates sometimes occurs when alkalis are present in the boiler feedwater. The caustic cracking is intergranular and results from a combination of stress, exposure to alkaline solutions at temperatures of $212^\circ F$ or above, and the opportunity for concentration of the solution in capillary spaces. Both hydrogen (H_2) and the deposit of iron oxide (FeO) formed by the reaction enter into the process, but the full explanation of the process is lacking. Common methods of protection when using alkaline water are to maintain a ratio of sulphate to alkali above a certain value which depends on the working pressure, or to distill the water.

Thermal Conductivity—Another property of importance at elevated temperature is the thermal conductivity of the material, shown in Figure 48—9. It will be noted that the addition of alloys decreases the thermal conductivity of carbon steel, and that the difference between the thermal conductivities of the various steels decreases with increasing temperature. The thermal conductivities of the austenitic 18 per cent chromium, 8 per cent nickel and of the 25 per cent chromium, 20 per cent nickel steels are the two lowest on the chart.

Thermal Expansion—In designing apparatus for use at elevated temperature, allowance must be made for the thermal coefficient of expansion of the component materials.

The linear thermal expansion (increase in length), in inches per foot in going from room temperature to any elevated temperature up to $1200^\circ F$, is shown in Figure 48—10.

The steels are listed in the order in which they occur on these bands. It is seen that the austenitic stainless steels have a higher coefficient of expansion than the ferritic steels, in which the thermal expansion is decreased by the addition of alloying elements.

Modulus of Elasticity—Elastic moduli are often needed for design for service at elevated temperatures. The temperature dependence of Young's modulus (tensile modulus, E) for a number of ferrous materials that might be used in service at elevated temperatures is shown in Figure 48—11. The relatively low spread in values which encompasses all the ferritic or austenitic steels should be noted, since it indicates that this prop-

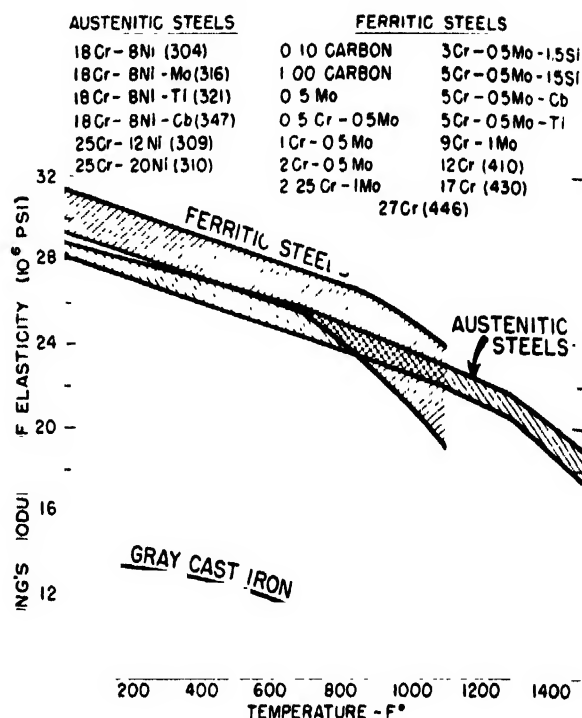


FIG. 48—11. Young's modulus, E , of various ferrous materials, as affected by temperature.

erty is relatively independent of composition and micro-structure.

Chapter 49

MECHANICAL TESTING

SECTION 1

INTRODUCTION

The manner in which metals react to applied forces is important in almost all practical applications. Steels, for example, are used in a multitude of applications involving their ability to withstand service loadings without permanently deforming or rupturing. Of equal importance, on the other hand, is the ability of steel to undergo large permanent deformations, thus permitting the formation of useful shapes under the application of the proper forces. Even though the final service application of a metal part may not in any way involve load-carrying ability, it is almost certain that at some stage in fabrication the manner of reaction of the metal to applied forces has come into play. Mechanical properties are the characteristic response of a material to applied forces; the methods of measurement of these properties and of evaluating their significance form the subject matter of this chapter.

It is both logical and useful to think of mechanical properties as falling into two broad categories: strength and ductility. Strength properties are related to the ability of the material to resist applied forces, while ductility is a measure of the ability to undergo permanent changes of shape without rupturing. Many mechanical characteristics or behaviors depend on both strength and ductility, frequently in a complex manner, as illustrated so well by that characteristic of materials known as toughness, or the ability to absorb energy. Even such complex behavior can be better understood, however, by considering it in terms of the separate contributions of strength and ductility.

The mechanical reactions of metals to applied forces are extremely diversified, depending upon the exact nature of the forces and the conditions under which they are applied. It is essential that cognizance be taken of this fact in attempting to devise or select tests which will permit a prediction of service performance. The final answer to the question of suitability for service can, in general, only be found in an actual service test. In most cases, however, actual service tests are impractical and simpler tests must be found, especially where frequent inspection of material is essential. In devising simpler tests, it is important that the type of forces involved in the particular service application of interest be known, i.e., whether the loading is tension, compression, bending, twisting, shear, etc. It is also important to know whether the loading is static or dynamic, and if dynamic, it is important to know the nature of the rate of application and variation of the loading. The temperature at which the loading is applied also may be of critical importance. If these features of the service conditions are known, one can be guided in the selection of tests most likely to provide a reliable evaluation of the suitability of a material for a particular application. One criterion of test validity with respect to service would be,

then, the extent to which the test simulates the actual application. For example, suppose that a large structural member is required to carry a certain compressive load at a temperature of 500° F. A test of the actual member is out of question because of the large size. An alternate course of action is to select from the material in question a small specimen suited to available testing equipment and to make a compression test at 500° F. The type of loading and the service temperature are thereby duplicated, and it is likely that the test results will be a reliable guide in selecting a material suitable for the application.

Another criterion by which a test may be selected depends on whether or not the test measures the same ultimate fundamental property on which successful performance in service depends. For example, in bending, the minimum bend radius obtainable with a particular material depends on the ability of the material to elongate over very short gage lengths on the tension fibers of the bend. Since the reduction of area in the tension test may be interpreted also as a measure of ability to elongate over very short gage lengths, it has been possible in some instances to relate minimum bend radii to reduction of area. The test is thus measuring a property which is directly responsible for the behavior in the application.

There exist a great many applications of metals, however, which are characterized by such complex conditions of loading that a directly applicable test may not be readily found. Because of this circumstance, the selection of mechanical tests for a particular instance is based primarily on experience. It may be known that, over a period of years, many lots of a certain grade of steel having properties falling within a certain range have performed satisfactorily in service. It can be expected, then, that new lots of this grade of steel having the same mechanical properties will perform satisfactorily in the same application.

A good example of a test selected on the basis of experience is offered by the extensive use of hardness testing in the inspection of deep-drawing sheets. It is known from experience that, in order for a sheet to be formed satisfactorily into a particular part, its hardness must lie within a certain range. Although hardness in itself can hardly be considered the property which controls the performance, it reflects the critical properties to a degree which makes it extremely useful as a control test, particularly in view of the ease with which hardness tests can be made. Testing on this basis, however, should only be considered as a means of increasing the probability of selecting satisfactory material. Unless the actual characteristics which determine the performance are known and are actually being tested for, an arbitrary test, even though backed by considerable experience, may at some

time fail to discriminate between good and poor material.

In the field of the mechanical behavior of metals, there is a strong trend away from arbitrary tests and toward a more exact evaluation of the fundamental properties which are actually called upon in particular service applications. The new methods of experimental stress analysis, particularly the wire-resistance strain-gage techniques, have permitted great advances to be made in more thorough evaluation of service requirements. Once the mechanics of an application have been analyzed and the critical fundamental material characteristics have been ascertained, the task of selecting a suitable test is greatly simplified.

Many steel products are sold to mechanical-property specifications and much of the testing carried out by the steel producer is for the purpose of ascertaining whether or not a given specification is being met. If the product is a widely used one, a standard specification may be set up by a body such as the American Society for Testing Materials in order to assist the user in specifying material for a certain application. Such specifications may provide fundamental design figures for the engineer, or in many cases may be based on previous experience which indicates that, if a material has certain properties, it will serve a particular purpose satisfactorily.

As already indicated, mechanical testing may be carried out as a means of quality control, even though the product may not be sold to a mechanical-property specification. In such instances, the experience of the producer dictates the test most suitable for the purpose.

The particular test chosen may depend to a great extent upon the ease with which it can be made if frequent inspection or extensive sampling is needed.

Another important function of mechanical testing is to determine the causes for failure in service. If the material is shown to be at fault, a substitution of another material or the elimination of the deficiency of the first material can be guided by judicious mechanical tests. If no fault can be found with the material, a change in design may be indicated.

Still another function of mechanical testing results from its great usefulness in the development of new and improved products. Even in applications so complex in nature that a final test in service may be required, it is usually possible, by judicious mechanical tests, to establish trends and select the materials most likely to perform satisfactorily.

In the subsequent sections of this chapter, the mechanical tests most generally applied to steel will be discussed. Attention will be focused primarily upon the tension test, the hardness test, the notched-bar impact test, the fatigue test, and the creep and rupture tests. In addition, certain miscellaneous tests sometimes made on steel will be briefly described. Emphasis will be placed on the significance of the various mechanical properties to be discussed, as well as on the precautions necessary to obtain reliable test results. Test procedures approved and recommended by the American Society for Testing Materials will be indicated for those tests for which standards or tentative standards have been established.

SECTION 2

THE TENSION TEST

Because of the large amount of information which can be derived from it, the tension test is undoubtedly the most generally useful of all the mechanical tests applied to steel. The versatility of the tension test lies in the fact that it permits both strength and ductility properties to be measured. The strength properties measured in the tension test are directly useful in design, whereas the ductility properties provide some indication of the extent to which changes in shape can be brought about by plastic forming, or an indication as to whether sufficient ductility is present for the intended service. Since the tension test is used quite extensively in the steel industry, a rather complete description of the test and of the properties measured will be presented.

Testing Machines—The machines employed in tension testing consist essentially of a load-producing mechanism and a load-measuring mechanism. The most elementary method of producing a tensile load simply involves the suspension of dead weights from the specimen to be loaded. This procedure is in general not practicable, however, because of the inconvenience of handling the large weights which would be required. Dead-weight loading, sometimes with the force multiplied by a lever, is used in certain instances, however, where it is desired to subject a specimen to a fixed load for a period of considerable duration. Creep and rupture testing, which is discussed in a later section, is perhaps the best example of the use of dead-weight loading for testing purposes.

The more commonly used testing machines employ either a mechanical system of loading actuated by screws or a hydraulic system in which the load is applied through a hydraulic ram. In both types of machines, there is a fixed crosshead and a moving crosshead through which the tensile force is applied. The moving crosshead in the screw machine is motivated either by

threaded columns (screws) rotating in stationary nuts or by rotating nuts, depending upon the design of the machine. Various speeds of crosshead movement are available through changes of the gear combination in the drive. In the hydraulic testing machine, the moving crosshead is powered by a hydraulic ram. The hydraulic machine has the advantage of a continuous variation in rate of crosshead movement which is controlled by a simple valve system. A complete range of rates of loading from very slow up to the limit set by the capacity of the testing machine pump can be obtained. The popularity of the hydraulic machine has been increasing in recent years.

The mechanical or screw type of testing machine is characterized by the load-measuring mechanism generally employed. In this mechanism, the load is transmitted through a system of levers acting on fulcrums of hardened steel to a beam carrying a movable weight or poise. The lever system is arranged so that the force exerted by the poise on the beam has a great mechanical advantage and can balance the force being applied to the tensile specimen. The poise can be moved outward along the beam, thereby increasing its lever arm and providing by its position a means of load indication. The movement of the poise is effected by manually rotating a screw during the test so that the beam is kept in balance. Because of the characteristic load-measuring mechanism, this type of machine is generally referred to as a "beam-and-poise" or "lever" machine. A typical machine is shown in Figure 49-1. One of the greatest objections to the lever machine lies in the manual operation of the weighing mechanism and the attendant human factor introduced into the test. An attempt has been made in one development to circumvent this objection by permitting the end of the beam to actuate a pendulum, the

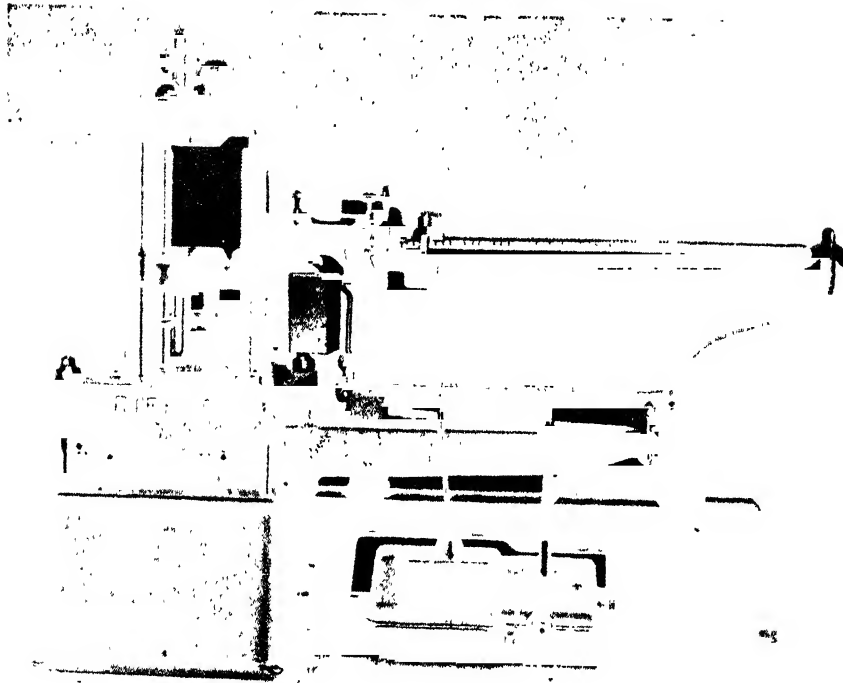


FIG. 49—1. Beam-and-poise testing machine. (Courtesy, Riehle Testing Machines Division, American Machine and Metals, Inc.)

movement of which provides a continuous and automatic indication of load.

Most of the recent developments in the field of tensile testing machines have centered around the hydraulic machine. Hydraulic testing machines are of two main types: the lapped-ram type and the packed-ram type. In the lapped-ram type, the piston and cylinder are lapped to an extremely close fit so that an almost frictionless movement is possible. Because of the extremely low friction, the oil pressure in the cylinder can be used directly as a measure of load on the machine. One load-measuring device used rather widely is the pendulum weighing mechanism. A small piston is acted upon by the same hydraulic pressure acting on the main ram, thus producing a force proportional to that acting on the main ram. The small piston is arranged in such a way as to displace a pendulum, the magnitude of the displacement providing an indication of the test load. Another method employs electronic indication methods for the direct measurement of the oil pressure.

In the packed-ram type of hydraulic testing machine, the piston and cylinder are not fitted as closely as in the lapped-ram type, and a packing is necessary to prevent leakage of oil from the system. Since friction between the packing and the ram is variable, depending upon the applied hydraulic pressure, the possibility of a direct pressure measurement being used to provide an accurate indication of load is precluded. One device used to circumvent this difficulty is the so-called Tate-Emery load indicator, which utilizes a hydraulic capsule. The load on the machine acts on the capsule which is part of a closed hydraulic system. The capsule is a very rigid assembly containing an oil layer of only about 0.030-inch thickness. As the load is applied, a pressure is developed in the capsule and transmitted to a Bourdon tube. The actual load indication is provided by a measurement of the force which is necessary to hold the tip of the

Bourdon tube in its equilibrium position. This force is measured by the displacement of the free end of a helical spring attached to the tip of the Bourdon tube. The spring is made from an alloy of the Elinvar type, so that the spring constant is unaffected by minor temperature fluctuations. Since the tip of the Bourdon tube is not required to move more than a very small amount, the usual objections to a Bourdon tube, such as hysteresis effects, are eliminated, and the weighing system is practically free of inertia. A modern hydraulic testing machine provided with a weighing system of the type just described is shown in Figure 49—2.

Generally, a number of load-measuring ranges are provided so that the most sensitive and accurate range can be chosen for a particular test, that is, the range having its top limit just above the maximum load expected in the test. It is important to know the accuracy of the load-measuring mechanism, and frequent calibration should be carried out to insure that the accuracy remains within the desired limits. With modern load-measuring mechanisms, accuracies of one-half of one per cent of the indicated load over all but the lowest portion of the range may be guaranteed. For most purposes, one per cent accuracy is considered satisfactory, but even at this level of accuracy, frequent calibration is desirable. If the errors are found to exceed the desired limits, the load-measuring mechanism should be adjusted until further calibration indicates the accuracy to be within the desired range.

Electronic devices are employed for load measurement in some machines of recent design. Wire resistance strain gages are used to measure the changes in dimensions of an elastic member through which the load is applied. The system is calibrated so that the changes in resistance of the strain gages provide an indication of the load. Different load ranges can be obtained by appropriate selection of the dimensions of the elastic member.

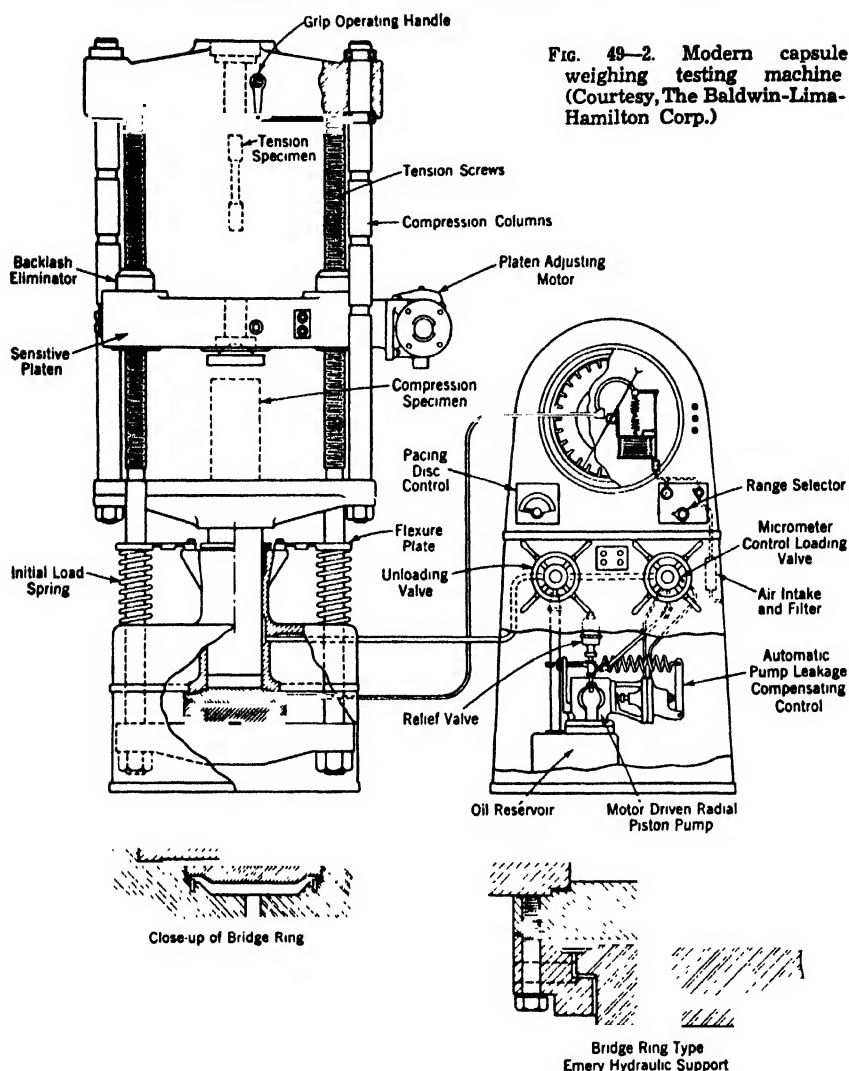


FIG. 49-2. Modern capsule weighing testing machine (Courtesy, The Baldwin-Lima-Hamilton Corp.)

In the ASTM Standards, three methods for the verification of testing machines are described. These include: (a) verification by standard weights (only applicable in the case of machines weighing a downward force), (b) verification by standardized proving levers, and (c) verification by elastic calibration devices. The first two of these methods make use of standard weights, in one case applying the weights directly to the machine, and in the other case making use of levers to multiply the forces that can be applied by the standard weights. Although both methods are quite accurate, they are obviously very inconvenient, and the feasible range of load capacities which can be covered does not encompass the larger machines.

The most widely used method of calibration makes use of elastic calibration devices. The most frequently employed device of this type is the so-called proving ring, shown in Figure 49-3. Such a ring is subjected to a series of loads in the machine to be calibrated, and the deflection of the ring is measured by a micrometer which is an integral part of the proving ring. Proving rings are available in a variety of ranges up to 300,000 lb. and are calibrated and certified by the National Bureau of Standards.

Extensometers—Certain aspects of tensile testing require the use of some device for the measurement of the

extension of the specimen; such a device is called an extensometer. There are a very large number of types of extensometers available and no attempt will be made here to describe the features and specific applications of all the various types. The characteristics of a few of the more widely used types will be discussed briefly in order to illustrate the general nature of commercially available extensometers.

Extensometers can be considered to fall into two main groups, depending on the range of extensions which can be covered. On the one hand, there are the very accurate, high-sensitivity extensometers used for the measurement of minute extensions. These extensometers are characterized by very small ranges; i.e., the total extension which can be measured is quite small. On the other hand, there are the long-range extensometers designed to measure extensions up to the instant of rupture of the specimen. It is evident that, if long range is desired, sensitivity must be sacrificed; while it is equally true that sensitivity can be gained only at the expense of range.

The optical types of extensometers provide a high degree of sensitivity and accuracy for extremely small extensions. Such extensometers are used in elastic-strain measurements and for very precise indications of the beginning of plastic yielding. Perhaps the most con-



FIG. 49-3. Morehouse proving ring, compression type. (Courtesy, Morehouse Machine Company.)

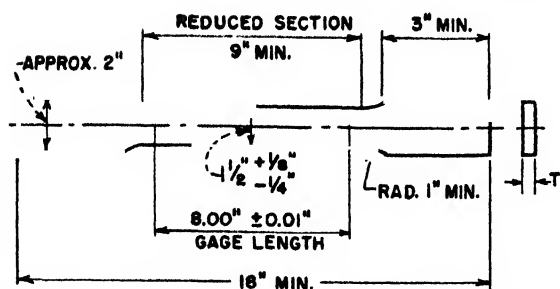
venient and frequently used optical type is the Tucker-man extensometer. This extensometer is attached to the specimen so that a fixed knife edge and a movable knife edge are in contact with the specimen. The movable knife edge consists of a lozenge, one face of which is polished to a mirror finish. A light beam is focused on the lozenge, and as the lozenge rotates as a result of extension of the specimen, the reflected beam is displaced relative to the incident beam. A specially calibrated collimator provides the light source and measures the deflection of the reflected beam. Extensions as small as two millionths (0.000002) of an inch per inch can be measured on a gage length of two inches.

Generally, the great sensitivity of the optical extensometers is not required and other more convenient types of low-range extensometers are employed. Extensometers can be either of the direct-reading (indicating) or recording type. The indicating extensometers make use of a dial gage to measure the movement of some element of the extensometer. A great many combinations of range and sensitivity can be obtained in dial-gage extensometers, the choice depending on the strain range over which observations are to be made.

Bonded wire resistance strain gages are sometimes

used in the determination of stress-strain curves, although their main application is in experimental stress analysis. Essentially, this type of gage consists of a grid of fine wire which is suitably bonded to the specimen, usually by cementing. As the specimen is strained, the wires of the grid undergo similar strains which produce a change in the cross-section of the wire. This change in cross-section results in a change in electrical resistance that is proportional to the amount of strain and can be readily measured. The range of strain of most wire resistance gages is about one per cent, but special gages having considerably higher ranges have recently become available. When suitable techniques are employed, the sensitivity and accuracy of wire resistance gages is at least as good as the best optical strain gages.

Recording extensometers are now in wide use and offer the advantage of an automatically plotted stress-strain curve from which determinations of various material characteristics can be made, as well as the advantage of a permanent record of the test. High-sensitivity short-range recording extensometers can be obtained with an accuracy of 0.00005 inch or one-half of one per cent of indicated strain, whichever is greater, and with a range of from 0.02 inch to 0.04 inch. Sensitivities up to



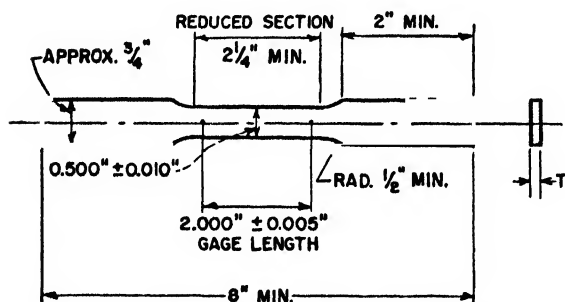
NOTE 1.—When necessary, it is permissible to use a narrower specimen but in such a case the reduced portion shall be not less than 1 in. in width.

NOTE 2.—The dimension "T" is the thickness of the test specimen as provided for in the applicable material specifications.

NOTE 3.—The reduced section shall be parallel within 0.010 in. and may have a gradual taper in width from the ends toward the center with the ends not more than 0.010 in. wider than the center.

NOTE 4.—The ends of the specimen shall be symmetrical with the center line of the reduced section within 0.10 in.

A.—STANDARD RECTANGULAR TENSION TEST SPECIMEN WITH 8-IN. GAGE LENGTH.

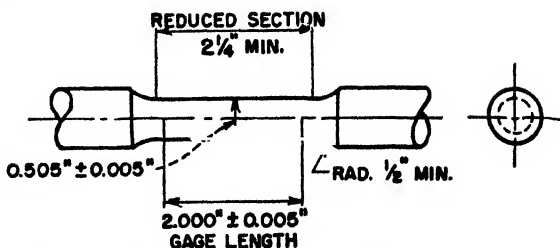


NOTE 1.—The dimension "T" is the thickness of the test specimen as provided for in the applicable material specifications.

NOTE 2.—The reduced section shall be parallel within 0.005 in. and may have a gradual taper in width from the ends toward the center, with the ends not more than 0.005 in. wider than the center.

NOTE 3.—The ends of the specimen shall be symmetrical with the center line of the reduced section within 0.05 in.

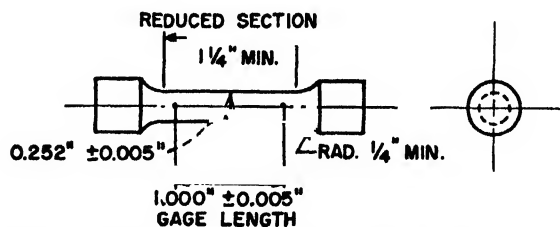
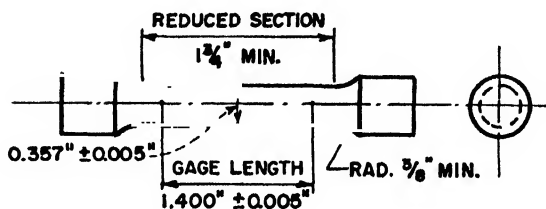
B.—STANDARD RECTANGULAR TENSION TEST SPECIMEN WITH 2-IN. GAGE LENGTH.



NOTE 1.—The gage length and fillets shall be as shown, but the ends may be of any shape to fit the holders of the testing machine in such a way that the load shall be axial.

NOTE 2.—The reduced section may have a gradual taper from the ends toward the center, with the ends not more than 0.005 in. larger in diameter than the center.

C.—STANDARD ROUND TENSION TEST SPECIMEN WITH 2-IN. GAGE LENGTH.



NOTE 1.—If desired, the length of the reduced section may be increased to accommodate an extensometer.

NOTE 2.—The gage length and fillets shall be as shown, but the ends may be of any shape to fit the holders of the testing machine in such a way that the load shall be axial.

NOTE 3.—The reduced section may have a gradual taper from the ends toward the center, with the ends not more than 0.003 in. larger in diameter than the center.

D.—EXAMPLE OF SMALL SIZE SPECIMENS PROPORTIONAL TO STANDARD 2-IN. GAGE SPECIMEN.

FIG. 49—4. Sketches of ASTM standard tension specimens. (From "ASTM Standards, 1955")

0.00002 inch can be obtained. This type of extensometer operates on a micrometer-screw principle. A small electric motor, known as a Selsyn motor, rotates a screw which makes electrical contact with an element of the extensometer actuated by the extension of the specimen. The movement of the screw results in a corresponding movement in the stress-strain recorder which can be calibrated to indicate the extension directly. By a different arrangement of the lever system in this type of recording extensometer, the range can be greatly extended, and such an extensometer, known as a total-elongation extensometer, is commercially available. The

magnification ranges of this extensometer vary from 5 to 20 as compared to 250 to 1000 for the more sensitive, short-range extensometer.

Another type of long-range recording extensometer which has seen considerable application is the wedge type. In this extensometer, a wedge of known taper is pulled through a slot in the extensometer which opens as the specimen elongates. The movement of the wedge is used to rotate the recorder drum and since the taper of the wedge is known, the strain can be calculated. The magnification obtainable with this extensometer depends upon the degree of the taper in the wedge.

Another type of recording extensometer operates on a magnetic principle. In this extensometer, the lever system is arranged in such a way that extension of the specimen results in a movement of an iron core in a small magnetic coil, thus changing the inductance of the coil. A similar magnetic coil is contained in the recorder, and is automatically kept in balance with the extensometer coil by a servomotor which also actuates the recorder. This extensometer provides a smoother motion of the recorder than the micrometer screw or Selsyn motor type of extensometer already described. The ranges and sensitivities available are comparable to those available in the micrometer-screw type.

For strain measurements in and just beyond the elastic range, it is desirable to use extensometers which incorporate an averaging mechanism. Unless extreme precautions are taken, axi-ality of loading will not be obtained and a non-uniform strain distribution will result. By averaging the extension along two opposite fibers of the specimen, a closer approximation to the extension which would have occurred had the loading been axial can be obtained.

Specimens—Certain standard tension specimens have been adopted and are recommended by the American Society for Testing Materials. The shapes and dimensions of the most frequently used tensile specimens are shown in Figure 49-4. Figure 49-4A shows the rectangular cross-section specimen with 8-inch gage length used for tests of plates and structural sections. The rectangular cross-section specimen with 2-inch gage length generally used in sheet-metal testing is shown in Figure 49-4B. The standard circular cross-section specimen with 2-inch gage length is shown in Figure 49-4C. The details of the ends of this type of specimen will vary widely, depending on the types of grips employed. If the section from which the specimen is to be taken is too small to permit the procurement of a full-size specimen, smaller specimens can be used if the dimensions are kept in geometric proportion. It is especially important that the gage length for measuring elongation be proportioned to the nominal diameter of the specimen. One-inch and 1.4-inch gage-length specimens are frequently used and have diameters of 0.252 inch and 0.357 inch respectively. The nominal diameters of 0.252, 0.357 and 0.505 inch for the specimens having gage lengths of 1 inch, 1.4 inches, and 2 inches, respectively, were selected to simplify calculation of loads in pounds per square inch from actual loads. These diameters provide specimens having respective cross-sectional areas of very nearly 0.05, 0.1, and 0.2 sq. in. Other specimens for special products can be found in the ASTM Standards. For certain types of products, such as small bars, tests may be made on the full section, in which case the only preparation necessary is the cutting of the specimen to length.

A number of general precautions should be observed in the procurement and preparation of tension-test specimens. It is important that heating and cold working of the specimen be kept to a minimum during procurement and preparation if reliable results are to be obtained. When specimen blanks are flame cut from plates, for example, allowance should be made for machining off all metal affected by the heat introduced during flame cutting. Cold shearing or punching of specimen blanks should be performed with care, since specimens improperly prepared in this manner may become cold worked and thus not be representative of the material being sampled. Care should also be taken to insure that the test specimen is straight and flat. An initial bow or curvature in a tension specimen will result in distortion of the elastic loading line and may affect the stress level at which initial plastic yielding occurs. In machining

tension-test specimens, precautions should be taken to insure that the various portions of the specimen are symmetrical with respect to the loading axis of the specimen. If such precautions are not observed, the specimen will be loaded eccentrically, and bending stresses will be set up. Such eccentric loading will affect the elastic loading line and may influence the initial yielding behavior.

Grips used in tension testing vary considerably, depending upon the type of specimen being used. For many tests, the so-called wedge grips, which are serrated and simply grip the specimen by a transverse pressure which builds up as the specimen is loaded, are satisfactory. However, if the specimen is rather short, eccentric loading may become serious, necessitating the use of machined specimen ends and of grips with spherical bearings, an arrangement which to some extent provides self-alignment. Eccentric loading of plate or sheet-type specimens may be caused by the method used in preparing the test specimen; for example, shear or punch drag, or beads of metal deposited on the edges of the grip ends when burning out the specimen. A sketch of typical spherical seated grips is shown in Figure 49-5.

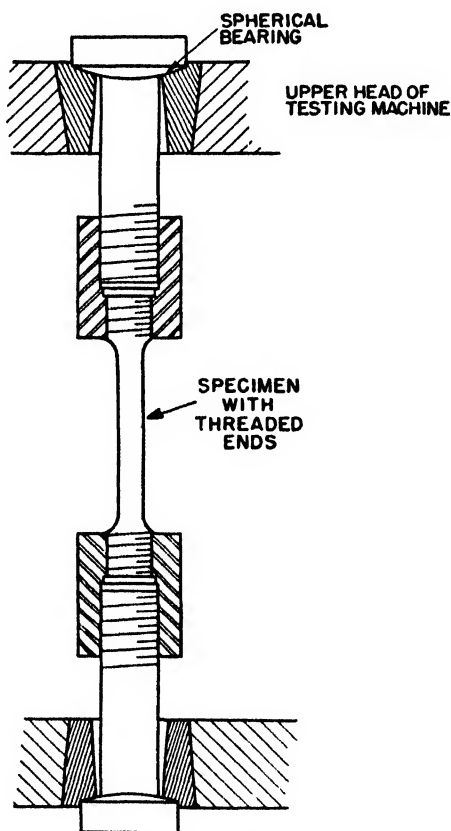


FIG. 49-5. Sketch of self-aligning tension grips. (From "ASTM Standards, 1955.")

For extremely accurate work, adjustable grips may be used in which the load is transmitted through a small hardened-steel ball and which permit a shift of the specimen with respect to the load application points in order to obtain a very high degree of axi-ality.

The Tension Test and the Properties which are Determined—In conducting a tension test, the specimen is introduced into the machine in such a manner that a load can be applied along the specimen axis, and the applied

load is then gradually increased until the specimen breaks. An important factor which is generally controlled within certain predetermined limits in tension testing is the speed of testing. Some of the tensile properties are markedly affected by rate of straining, and it is important to specify and control the speed of testing within certain definite limits. These limits should be chosen in such a way as to prevent more than a certain percentage variation in the property being measured from arising as a result of variations in the speed of testing. Speed of testing may be specified in terms of rate of stressing the specimen, rate of crosshead movement, or rate of straining of the specimen. The most reliable of the three methods of specifying speed of testing is the rate of straining of the specimen, since any variations in properties as a result of variations of speed of testing arise directly from variations of strain rate in the specimen itself. In some instances the rate of crosshead movement may be closely enough related to rate of strain in the specimen that it can be used satisfactorily for control purposes.

Before considering the determination of specific properties in the tension test, it is necessary to define stress and strain, since these concepts are used in the definitions of the various tensile properties. The definitions of stress and strain proposed by the American Society for Testing Materials are as follows:

Stress—The intensity at a point in a body of the internal forces or components of force which act on a given plane through the point. Stress is measured in force per unit area (pounds per square inch, kilograms per square millimeter, etc.).

Strain—A measure of the change in size or shape of a body, due to force, referred to its original size or shape. Strain is a non-dimensional quantity, but it is sometimes expressed in inches per inch, etc.

In discussing the various properties which are determined in tension tests, these properties will be classified according to whether some aspect of strength or ductility is being measured. Further details of testing procedure will be mentioned as they apply to the measurement of specific properties.

A. STRENGTH PROPERTIES

Modulus of Elasticity (Young's Modulus)—When a metal is subjected to load, there is an initial range of loading in which no permanent deformation of the specimen occurs; i.e., if the load is removed at any value within this range, the specimen will return completely to its original dimensions. Furthermore, within this range of loading, which is designated as the **elastic range**, the strain produced is directly proportional to the applied stress; i.e., the strain produced by 20,000 lb. per sq. in. stress will be twice that produced by 10,000 lb. per sq. in. The law of proportionality between stress and strain in the elastic range is known as **Hooke's Law**. The modulus of elasticity is simply the proportionality constant between stress and strain and can be obtained from the slope of a plot of stress against strain within the elastic range. The modulus for ordinary steels is usually taken as about 30,000,000 lb. per sq. in. in design work. Since the modulus does not vary much from steel to steel, it is not determined except in special instances where a more accurate value may be needed, or for example, where the effect of temperature on the modulus must be ascertained. Very great accuracy of load measurement and strain measurement is necessary for reliable modulus determinations. Furthermore, special precautions must be exercised to procure specimens free from residual stress and to insure very accurate specimen alignment.

Elastic Limit—In practice, the elastic limit is determined by subjecting a specimen carrying a strain-measuring device (extensometer) to a series of loading steps in which the maximum load applied is gradually increased, the load being released completely at each step. A load will finally be reached upon release of which the specimen will fail to return to its original length: this load is the elastic limit. The size of the load increments used and the sensitivity of the extensometer used will, of course, affect the value obtained, and, consequently, this property is not frequently determined.

Proportional Limit—The proportional limit represents an aspect of elastic behavior similar to the elastic limit, the principal difference lying in the method of determination. The straight-line proportionality between stress and strain in the elastic range has already been discussed. It is the upper limit of the range of proportionality that defines the proportional limit. In other words, the proportional limit is the greatest stress which a material is capable of developing without a deviation from the law of proportionality (Hooke's Law).

In practice, the proportional limit is determined from a plot of stress against strain, being taken as the stress at the first visible departure from the straight line drawn through the points in the elastic range. Since the departure from linearity is in general quite gradual, the determined value will depend on the accuracy and sensitivity of the strain-measuring device employed in the test. The experimentally determined value for a given material will be found to decrease as the sensitivity of extensometer used is increased, that is, as the ability to detect smaller and smaller strain increments is increased. Because of these uncertainties, proportional limit is very seldom employed in specifications.

Yield Strength—As the tensile load on a specimen is increased through the elastic range, a stress will be reached at which the specimen will begin to deform in a plastic manner; i.e., it will undergo a permanent set which is not recoverable upon release of load. In the design of structural members to be subjected to static loads, it is generally necessary to design so that the service loads do not cause large deformations, since the usefulness of the structure would thereby be destroyed. It is for this reason that the portion of the tension test concerned with the onset of plastic yielding is of extreme importance. It has already been shown that in many materials the very first stages of plastic yielding are very difficult to detect, and that the stresses corresponding to the apparent beginning of yielding depend on the sensitivity of the strain measuring instrument used. It has become customary, therefore, to refer to the stress at which a material exhibits a specified limiting permanent set as the yield strength. The choice of the limiting amount of offset is to some extent arbitrary, but insofar as possible should be based on that amount of plastic yielding that would be considered damaging in a statically loaded member of a structure. Generally, yield strength is based on a 0.2 per cent permanent set.

Assuming that a 0.2 per cent permanent set has been chosen, the following example will illustrate the so-called offset method which is commonly used in determinations of yield strength. In Figure 49-6, the early portion of a stress-strain diagram extending up to about one per cent elongation is shown. Let the origin be designated as O and the elastic line and its extension as OA. Now, on the strain axis, lay off OB = 0.2 per cent, and construct BC parallel to OA. The stress corresponding to the point at which BC intersects the stress-strain curve represents the 0.2 per cent offset yield strength. The offset method is based on the observed fact that if the load is released at X, the specimen will recover

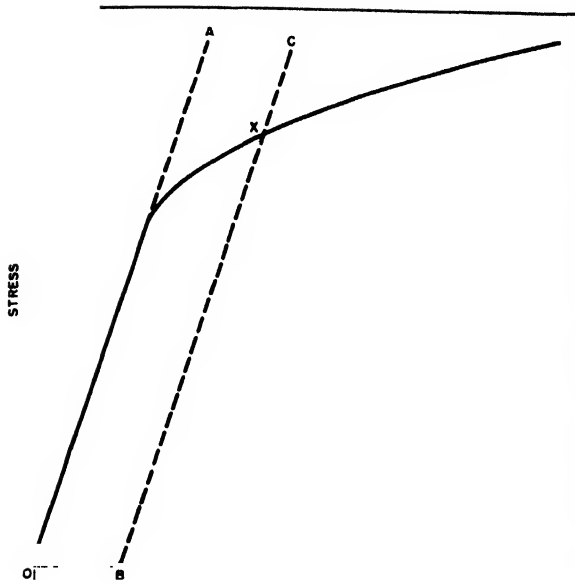


FIG. 49-6. Diagram illustrating offset method for determining yield strength.

along BX until at zero load, the permanent set OB remains. The procedure for any other amount of offset is identical with the exception that the offset OB will correspond to the new amount of permanent set chosen as the basis of the yield strength. The amount of offset used should always be reported with yield-strength values.

If a large number of tests, as for example in production control testing, are to be made on a material for which the stress-strain characteristics are known from experience, a shortened procedure for determination of yield strength may be used, which is known as the extension-under-load method. This method is based on the fact that the total extension, i.e., the sum of the elastic and plastic extensions, corresponding to the amount of permanent set chosen as a basis for the yield strength determination will be known within certain fairly narrow limits. Since for a given material the stress corresponding to the yield strength will not ordinarily vary more than a certain percentage, say 10 per cent, the elastic strain present at the offset on which the yield strength is based also will not vary more than 10 per cent. This 10 per cent variation in elastic strain will be a small fraction of the total strain at 0.2 per cent offset. For this reason, the yield strength can satisfactorily be determined at a total extension which is based on the permanent set chosen plus a mean expected value of elastic strain. An extensometer reading to 0.0001 in. per in. of gage length should be used for the total-strain method.

It should be pointed out that the elastic limit and the proportional limit can be considered as special cases of yield strength, the permanent set or offset corresponding to the least permanent strain detectable with the instruments used.

Yield Point—It is only for those materials that with increasing stress show a gradual departure from elastic behavior and thus exhibit a stress-strain curve such as shown in Figure 49-7 that it is necessary to define the onset of plastic yielding in terms of yield strength. Many steels exhibit a rather abrupt yielding and may show an initial increase of strain without any appreciable in-

crease of stress when yielding occurs. Such materials are said to exhibit a yield point, the yield point being defined as "the stress in a material at which there occurs a marked increase in strain without an increase in stress." This definition of yield point is that presented in the ASTM Standards and forms the basis for yield-point specifications. As will be pointed out, the definition does not provide a complete description of the yield-point behavior.

Since the yield-point phenomenon is complex, an understanding of its general features is essential for the proper planning and execution of yield-point determinations. The degree to which the yield point is manifested varies widely, depending upon the grade of steel being tested and upon the thermal and mechanical history of the steel. In some steels, the yield point may appear as little more than a "jog" in the stress-strain curve, while in low-carbon deep-drawing steel whose final treatment has been box annealing, the yield-point behavior may extend over elongations of several per cent.

The yield-point behavior can most easily be pictured by a description of the sequence of events during the yielding of a steel which shows a very pronounced yield point. If such a steel is stressed in tension, the load rises as the specimen is strained in the elastic region, and suddenly falls when the first yielding occurs. After this initial drop, with continuing elongation, the load fluctuates about some fairly constant value for a time and then begins to rise again. The maximum stress before the sudden drop is known as the upper yield point, and the average value of the relatively constant stress level that follows is known as the lower yield point. The amount of extension which occurs before the load begins to rise steadily again is called the yield-point elongation. The yielding process just described is very heterogeneous, different portions of the specimen successively undergoing yielding. After the first drop in load, locally depressed areas can be seen to form and to grow over the entire specimen as straining proceeds. These locally deformed areas are usually visible in the form of surface irregularities or strain markings which are referred to by a variety of terms including Lüders' lines, Hartmann lines, stretcher strains, "worms," and others. The formation of such strain markings has also been referred to as the Piobert effect, since the Frenchman Piobert was one of the first to observe the phenomenon. As the

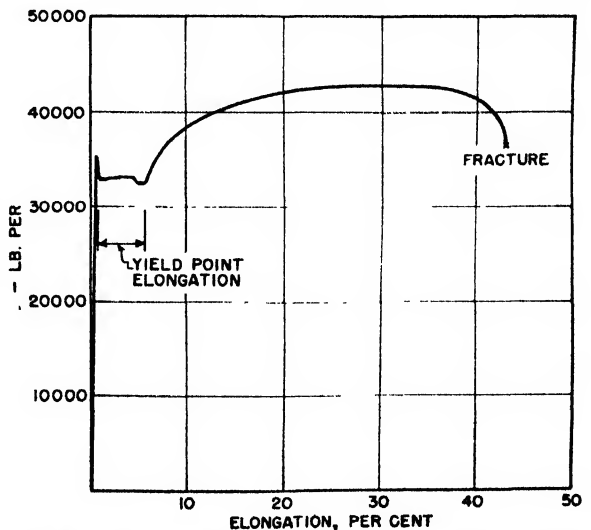


FIG. 49-7. Stress-strain curve for box-annealed rimmed deep-drawing steel.

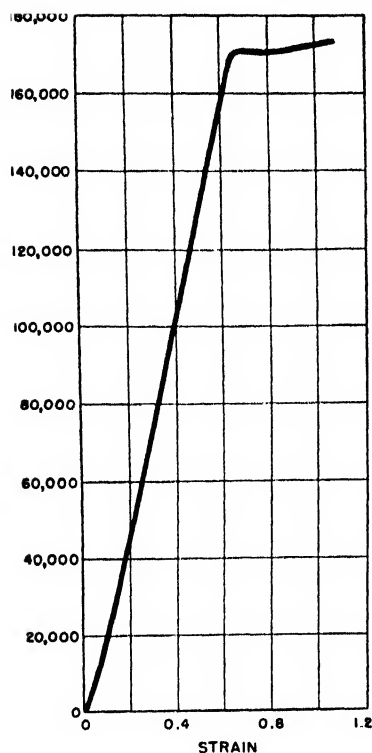


FIG. 49-8. Stress-strain curve for AISI 4340 steel, quenched and tempered at 900°F.

specimen is strained through the yield-point-elongation range, which may amount to about 10 per cent, the strain markings continue to grow and gradually merge until the entire specimen has yielded. From this point on, the specimen deforms in a homogeneous manner, as opposed to the highly localized mode of deformation occurring during the yield-point elongation. Aggregate features of the yielding behavior just described are well illustrated by the stress-strain curve shown in Figure 49-7 for an annealed-last, rimmed, deep-drawing steel. Generally, the heterogeneous yielding process will be much less pronounced as, for example, is illustrated by the stress-strain curve shown in Figure 49-8 for a quenched and tempered bar of AISI 4340 steel.

Heterogeneous yielding is associated with the presence of carbon and nitrogen in solid solution in ferrite. The carbon and nitrogen atoms are so situated that they exert an anchoring effect against the onset of plastic flow. When a sufficiently high stress is reached, the anchoring effect is suddenly overcome, and a yield point is observed.

One of the outstanding features of the yield-point behavior is the extreme sensitivity of the upper yield point to specimen preparation and testing conditions. Initial yielding can be greatly affected by nonaxial loading, sharp fillets in the specimen, cold working of the specimen during procurement or preparation, or the presence of residual stresses in the specimen. Any of these factors may be sufficiently effective to obscure the upper yield point entirely, and if the material being tested does not exhibit appreciable yield-point elongation, the influence of the aforementioned factors may eliminate all evidence of the yield point.

A number of methods of determining the yield point are recognized by the American Society for Testing Materials. These include the drop-of-beam or halt-in-gage

method, the dividers method, the total-strain method using an extensometer, and the autographic-diagram method. These methods are generally used only for materials exhibiting "sharp-kneed" curves for their stress-strain characteristics; for other materials a value equivalent to the yield point in its practical significance may be determined by methods outlined in "ASTM Standards, 1955," under Designation A370-54T (pages 359 and 360).

a. Drop-of-the-Beam Method—This method applies particularly to tests conducted on the beam and poise type of testing machine described in a preceding section of this chapter. As the specimen is loaded, the beam is kept in balance by running the poise outward along the beam. To quote the ASTM Standard A370-54T: "When the yield point of the material is reached, the increase of load stops, but the operator runs the poise a trifle beyond the balance position and the beam of the machine drops for a brief but appreciable interval of time." The load at this drop of the beam is recorded, and the corresponding stress is taken as the yield point. In view of the previous general discussion of the yield-point behavior, it seems evident that, if an especially pronounced yield point is present, an actual drop in the load carried by the specimen may occur at the upper yield point, which will result in a true drop of the beam not depending upon the operator. The main criticism of the drop-of-the-beam method lies in the fact that a drop of the beam can be obtained when a true yield point is not present in the specimen. When yielding occurs, even if no yield point is present, the rate of change of stress with strain undergoes a marked decrease. If the operator is running the poise out along the beam at a fast rate such as used in ordinary testing, he is apt to overshoot the load at which yielding occurs and observe a drop of the beam. Thus, it is possible to obtain a drop of the beam even though the stress-strain curve exhibits no region in which the strain increases with no increase in stress. For this reason, considerable skill and experience are required in the operation of the beam-and-poise machine, some indication of the presence of a true yield point being provided by the sharpness of the drop of the beam. It must be accepted, however, that the human factor is a serious variable in the drop-of-the-beam method of determining the yield point.

In testing machines provided with automatic load-indicating devices, a sudden halt or a drop of the load-indicating pointer will occur at the yield point. The load at the "halt in the gage" is recorded and the corresponding stress taken as the yield point.

b. Total-Strain Method Using Dividers, or Dividers Method—In this method, an observer with a pair of dividers set to correspond to the distance between two gage marks on the specimen places the divider points in the gage marks and watches for visible elongation as the specimen is loaded. When visible stretch has been observed, the load is noted, and the corresponding stress is taken as the yield point.

c. Total-Strain Method Using Extensometer—This method is similar to the dividers in principle but is more sensitive. An extensometer reading to 0.0001 inch per inch of gage length is attached to the specimen, which is loaded at a uniform rate. The yield point is based on the load at which the specimen is observed by the extensometer to exhibit a sudden increase of rate of elongation. Because of the greater sensitivity, this procedure may result in lower yield-

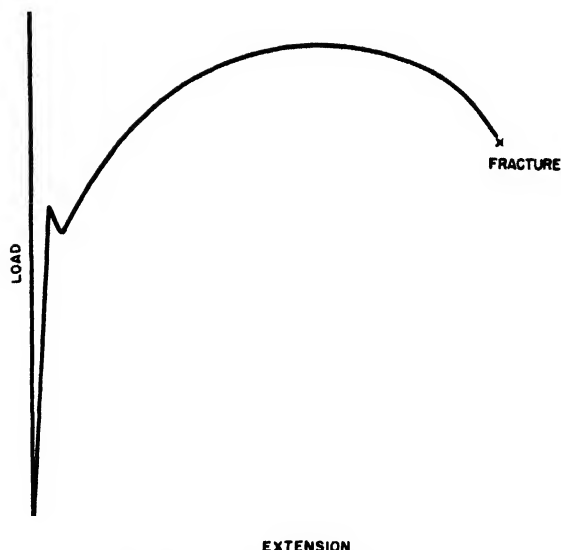


Fig. 49-9. Typical load-extension diagram.

point values than those obtained by the dividers method.

d. **Autographic-Diagram Method**—When a load-extension diagram has been obtained by an autographic recording device, the stress corresponding to the top of the knee or the point at which the curve drops is taken as the yield point. When a pronounced yield-point behavior is exhibited, both upper and lower yield points, as well as the amount of yield-point elongation are often reported. In this manner a more complete description of the behavior of the material is provided.

Tensile Strength—As the specimen is strained past the yield point, the load rises. For ductile materials, the load passes through a maximum and fracture eventually occurs, as shown in the schematic load-extension diagram of Figure 49-9. Some less-ductile materials may fracture while the load is still increasing, that is, without passing through a maximum. The tensile strength, according to the ASTM, "shall be calculated by dividing the maximum load carried by the specimen during a tension test by the original cross-sectional area of the specimen."

As already mentioned, speed of testing can affect the tension properties of a material. Considerable research has been conducted on the effects of speed of testing, and it has been generally concluded that the yield point is affected to a much greater extent than the tensile strength. It has been shown that a tenfold increase in rate of pulling increased the yield point of a 0.12 per cent carbon steel by about 7200 lb. per sq. in. in a two-inch gage length specimen. The effect on tensile strength in the range of speeds investigated was considered to be negligible. These observations have the practical significance that, in general, the control of the speed of testing is more critical in the region of the yield point than in the later stages of the test.

B. DUCTILITY PROPERTIES

Elongation—One aspect of the ductility of a material which is generally determined in the tension test is the elongation which the material is capable of undergoing before the occurrence of fracture. The elongation is

measured over some arbitrarily chosen gage length which is laid out on the specimen prior to the test. The gage length chosen depends upon the specimen being tested, but is usually two inches or eight inches. After the specimen is broken, the two fractured portions are fitted together and the new distance measured, usually to the nearest 0.01 inch. The percentage elongation is then calculated in the following manner:

$$\text{Per Cent Elongation} = \frac{\text{New length} - \text{Original length}}{\text{Original length}} \times 100$$

The original length refers to the initial gage length chosen and the new length refers to the length to which the initial gage length has been extended during the test.

The elongation at fracture of a ductile metal is not distributed uniformly along the length of the specimen. This nonuniform distribution is a consequence of necking down, the local elongation being greatest in the necked-down region of the specimen. This behavior is illustrated in Figure 49-10, which shows the distribu-

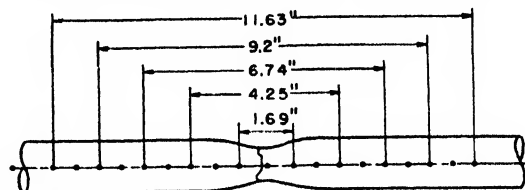


Fig. 49-10. Distribution of elongation along fractured tension specimen. (Original spacing between gage marks, ½ inch.) (From "The Mechanical Testing of Metals and Alloys," by P. Field Foster, page 76. Published by Sir Isaac Pitman & Sons, Ltd., London, 1948.)

tion of elongation over one-inch gage lengths along a fractured specimen. Up to the maximum load, the specimen elongates in a uniform manner; at maximum load, however, the strain becomes localized and the specimen necks down. The portions of the specimen sufficiently removed from the necking zone then cease to elongate. It can be seen, therefore, that the total elongation measured at fracture over some arbitrary gage length is a sum of two components: the elongation up to maximum

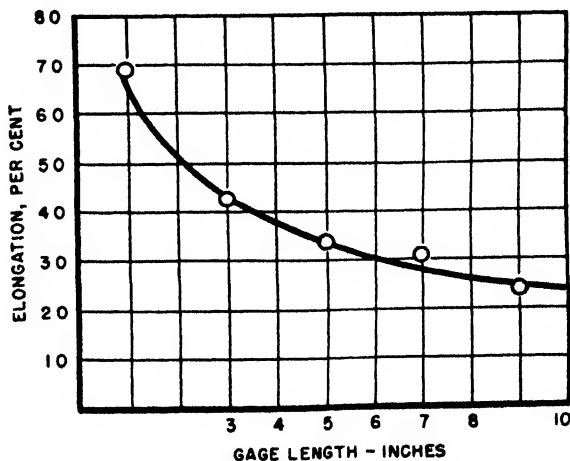


Fig. 49-11. Per cent elongation as function of gage length for fractured tension specimen. (From "The Mechanical Testing of Metals and Alloys," by P. Field Foster, page 77. Published by Sir Isaac Pitman & Sons, Ltd., London, 1948.)

load (uniform elongation) and the elongation after maximum load (local elongation). The elongation will obviously vary, therefore, with the gage length over which it is measured, being greater the smaller the gage length. This variation of elongation with initial gage length is illustrated in Figure 49-11 for the same specimen represented in Figure 49-10.

When comparing elongations obtained from different sizes of specimens, it is essential that geometric similarity be observed in the comparison. In other words, the ratio of gage length to cross-sectional dimensions must be held constant if comparable results are to be obtained. Barba's Law of Similarity states that for tension tests on different sizes of specimens from the same material, the same elongation values are to be expected only if the gage lengths are maintained in proportion to the square root of the cross-sectional area of the specimens.

Reduction of Area—The reduction of area provides a measure of the ultimate local ductility of a material up to the instant of rupture. In determining the reduction of area, the fractured tensile specimen is fitted together, and the dimensions at the minimum cross-section are measured. From the original and final areas, the percentage reduction of area is calculated in the following manner:

$$\text{Per Cent Reduction of Area} = \frac{\text{Original Area} - \text{Final Area}}{\text{Original Area}} \times 100$$

C. SIGNIFICANCE OF THE TENSION TEST

In addition to providing engineering design data, the tension test is used to a very great extent as an empirical test for evaluating the suitability of materials for particular mechanical applications. The test is interpreted in such instances on the basis of a wealth of practical experience, which permits an engineering opinion of the probable performance of a material to be drawn according to the tensile properties of materials known to have performed satisfactorily in the past. Many tensile-property specifications are drawn up in this manner and may not be used directly for actual design purposes.

Traditionally tensile strength has been used widely as a basis for design. At present, there is a marked trend to base the engineering design of structures to be subjected to static loads upon yield point or yield strength, with the application of safety factors based on engineering judgment. The use of the yield point or yield strength as a basis for design is, of course, based on the premise that any appreciable over-all yielding of the structure will destroy its usefulness. Tensile ductility specifications are almost universally based on practical experience as to what amount of ductility in the tension test has accompanied adequate ductility in service.

With regard to the use of the tensile yield strength or yield point in design, the question may well be raised as to how such a property, determined in a simple unidirectional tensile loading, can be used in the design of structures which are to be subjected to complex loads and complex states of stress. Any state of combined stresses can always be broken up into three so-called principal normal stresses which act in three mutually perpendicular directions. These three principal stresses can be designated as S_1 , S_2 , and S_3 , the subscripts indicating the order of algebraic magnitude; i.e., S_1 is the algebraically greatest principal stress. Some examples of simple stress combination are listed below and are illustrated in Figure 49-12. Tensile stresses are ordinarily designated as positive, while compressive stresses are designated as negative.

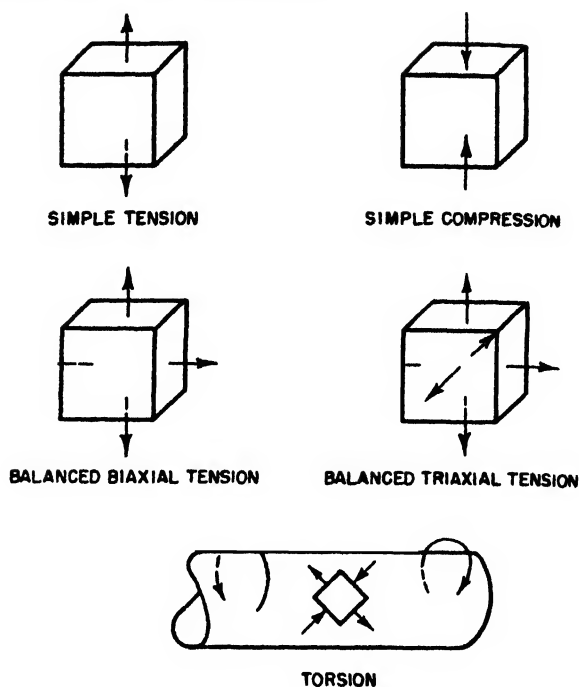


FIG. 49-12. Illustration of simple states of stress.

Simple tension:	S_1 positive; $S_2 = S_3 = 0$
Simple compression:	$S_1 = S_2 = 0$; S_3 negative
Balanced biaxial tension:	$S_1 = S_2$, both being positive; $S_3 = 0$
Balanced triaxial tension:	$S_1 = S_2 = S_3$, all positive
Torsion or twisting:	$S_1 = -S_3$, S_1 positive and S_3 negative; $S_2 = 0$

Two principal theories of initial yielding are used in predicting the strength under combined stresses: the critical shear stress theory and the critical shear strain energy theory. Shear stresses are those stresses which tend to cause one part of a body to slip over another part, as opposed to normal stresses which act in such a way as to tend to separate the body along a plane normal to the stress direction. It is recognized that shear stresses are the important stresses in controlling plastic flow. The maximum shear stress theory of yielding states that plastic action will begin when the maximum shear stress reaches some critical value characteristic of the material, the maximum shear stress being given by half the difference between the greatest and the least principal normal stresses. If the yield point in the tension test is designated as S_0 , the maximum shear stress yielding criterion can be stated as:

$$\text{Critical shear stress} = \frac{S_1 - S_3}{2} = \frac{S_0}{2}$$

As an example of the use of this relationship, the value of the greatest principal stress at yielding will be calculated for torsion, where the state of stress is: $S_1 = -S_3$; $S_2 = 0$. For purposes of illustration, it is assumed that the yield point in simple tension, S_0 , is 100,000 lb. per sq. in. It is then found that:

$$S_1 - S_3 = 100,000$$

$$S_1 - (-S_1) = 100,000$$

$$S_1 = 50,000$$

This solution indicates that a bar which is twisted will yield at half the value of the greatest principal stress, S_1 , at which a bar pulled in tension will yield.

The other important theory of yielding is the shear strain energy theory. As a body is loaded in the elastic range, elastic strain energy is stored up, just as when a spring is stretched. The total elastic energy stored in a body consists of two parts: that resulting from a change in volume of the body and that resulting from a change in shape of the body. The latter portion is known as the distortion energy or the shear strain energy. According to the shear strain energy criterion for initial yielding, yielding will occur when the shear strain energy reaches a critical value, which is dependent upon the material. This criterion can be expressed in terms of the three principal stresses as follows:

$$(S_1 - S_2)^2 + (S_2 - S_3)^2 + (S_3 - S_1)^2 = 2S_y^2$$

where S_y is again the yield point in simple tension. If the problem of the torsion of a bar of 100,000 lb. per sq. in. yield point material is again considered, it will be found that the value of S_1 at yielding is 57,700 lb. per sq. in. as compared to the value of 50,000 lb. per sq. in. predicted by the maximum shear stress theory, a difference amounting to 15.4 per cent. In actual experiments conducted to test the two theories, the predictions of the shear strain energy theory are generally found to be in better agreement with the experimental results. The maximum shear stress theory is sufficiently accurate for many purposes, the difference between the predictions of the two theories never amounting to more than the 15.4 per cent cited for the example of torsion.

Another direction in which the significance of the tension test has been extended is the development of the concept of the so-called true stress-strain curve. Ludwik, a German investigator who was one of the outstanding figures in the development of better understanding of the mechanical behavior of materials, pointed out around 1910 that the definitions of stress and strain, as commonly used in materials testing and in defining engineering properties, are in some respects lacking in fundamental significance. Only in relatively recent years, however, have the concepts of Ludwik come to be widely used in this country. The principal criticism of the customary treatment of tensile data lies in the definitions of stress and strain, these quantities generally being referred to the initial dimensions of the specimen throughout a test. Such a procedure is obviously in error and has no real physical significance except at the very beginning of the test.

As a tensile specimen is stretched, its cross-sectional area diminishes progressively. Only at the beginning of the test, therefore, can stress be based on the original area without introducing appreciable, and, indeed, serious error. True stress, then, is defined as the instantaneous stress acting on the specimen and is computed by dividing the instantaneous load by the actual cross-sectional area at the instant that particular load is acting.

Since the area will have changed a negligible amount at initial yielding, the usual definitions of yield point and yield strength are satisfactory. Since the change in cross-sectional area at the maximum load is considerable, however, it must be concluded that the tensile strength is not a real stress and has no fundamental physical significance. Its principal use is for relative comparison purposes and in testing for uniformity of product.

The usual definition of strain, in which strain is referred to initial dimensions, is likewise in error. The inaccuracy of referring elongation increments to an original gage length can be visualized in the following

manner. If a bar ten inches long is stretched one inch, the elongation expressed as a percentage is 10 per cent. Now suppose that the bar, initially ten inches long, has been stretched to a length of 100 inches and is then stretched one inch further. The last inch of stretch referred to the original length would represent an elongation of 10 per cent. If, however, we base the elongation produced by the last one inch increment of stretch on the length of the specimen just prior to that increment, the elongation would be considered to be 1 per cent. Obviously, the latter procedure provides a truer physical description of the stretching process than referring the strains to the initial dimensions. "True strain" has been defined, therefore, in the following manner:

$$\text{True strain} = \int_{l_0}^l \frac{dl}{l} \\ = \log_e \frac{l}{l_0}$$

where l_0 is the original length between two gage marks, l is the instantaneous length between these marks as the specimen is stretched, and e is the base of Napierian or natural logarithms. \log_e is usually expressed simply as \ln , so that the above expression may be written: True strain = $\ln \frac{l}{l_0}$. According to this definition, true strain

is simply a summation of strain increments in which each increment is referred to the instantaneous specimen dimensions. A comparison of true strain with ordinary or nominal strain can be seen in Figure 49-13.

A stress-strain curve making use of true stresses and true strains provides a more fundamental description of the plastic behavior of a material than the ordinary nominal stress-strain curve and can, therefore, be expected to be of greater general significance. In practice, the method of determining true stress depends to some extent upon the shape of the specimen. For a cylindrical specimen, a convenient method is based on simultaneous measurements of load and minimum specimen diameter. The true strain can be calculated from area changes as well as from length changes, since the volume of the specimen does not change by a significant amount during plastic deformation. Therefore,

$$A_0 l_0 = A l \quad \text{and} \\ \frac{A_0}{A} = \frac{l}{l_0}$$

where A_0 and l_0 are the original cross-sectional area and length, A and l are the instantaneous area and length, respectively. Since true strain was defined as $\log_e \frac{l}{l_0}$,

it follows that $\log_e \frac{A_0}{A}$ also represents true strain. Pointed

micrometers should be used in the diameter measurements for the determination of true strain, especially after maximum load when the local constriction produced by necking down has become appreciable.

With thin sheet specimens, where area measurements are difficult to make, an extensometer can be used up to maximum load; i.e., during the period of uniform elongation, for obtaining length measurements to be used in calculating true strain. In order to obtain instantaneous area values for the calculation of true stress, the constancy of volume condition is used, thus permitting a computation of area changes from length changes. This procedure must not be used after maxi-

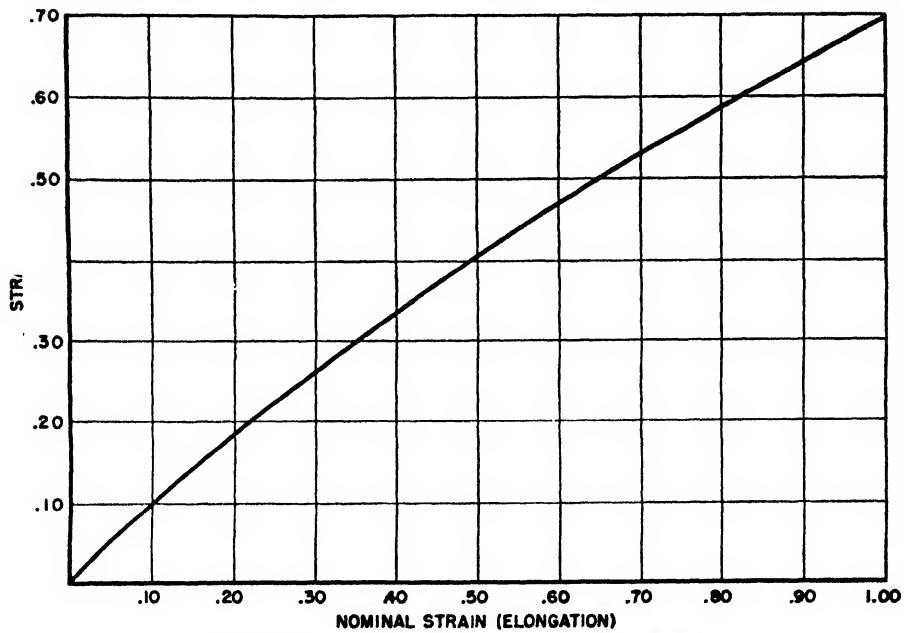


FIG. 49-13. Comparison of nominal and true strains.

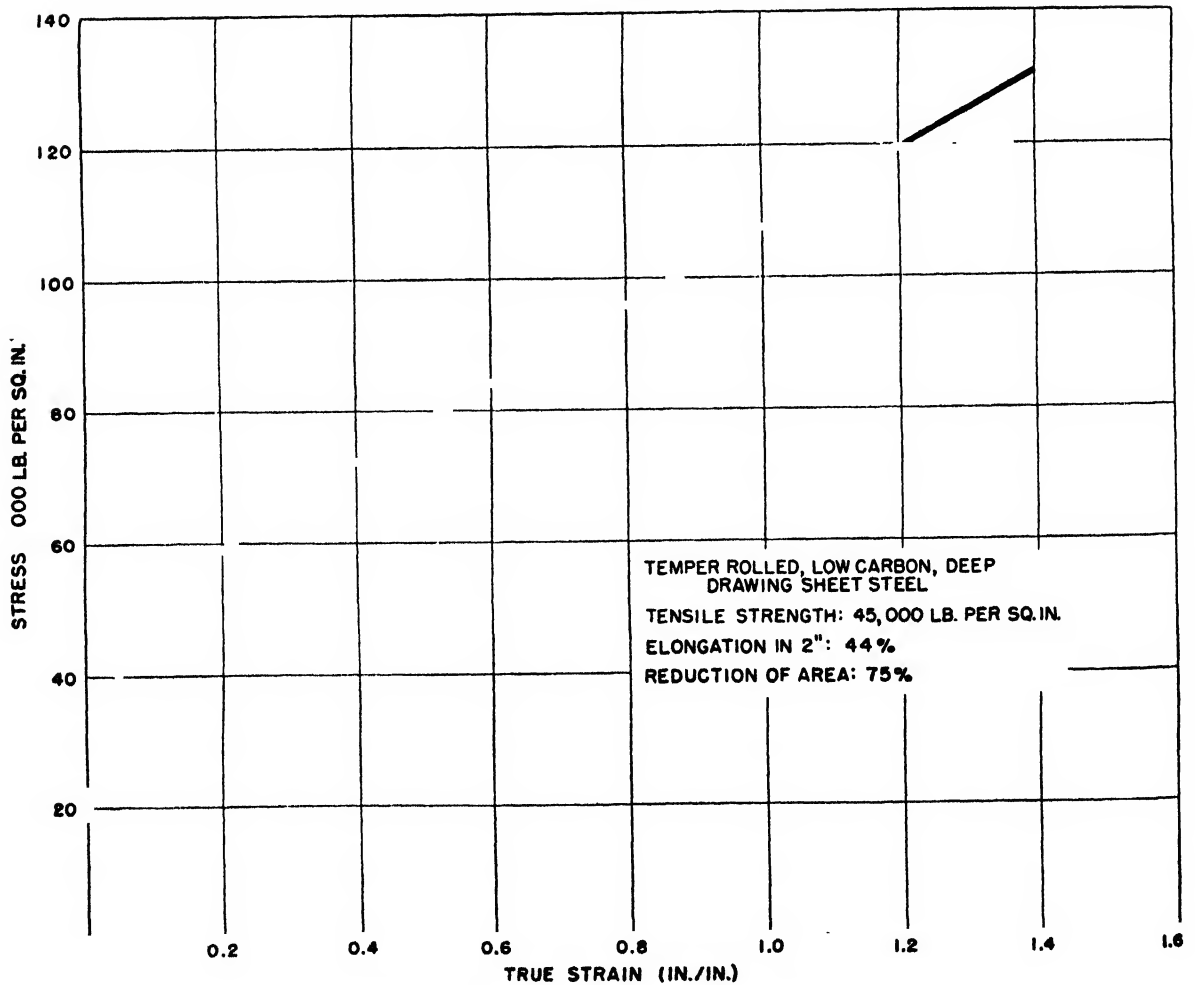


FIG. 49-14. Typical true stress-strain curve.

mum load because of the nonuniformity of the elongation along the gage length which arises as a result of necking down.

An example of a true stress-strain curve is shown in Figure 49-14. It can be seen that the true stress rises continuously with increasing true strain. This is a consequence of strain hardening which is active during the entire test up to the instant of fracture. It is apparent that the ordinary load-extension diagram does not provide a clear-cut description of strain hardening. The strain-hardening characteristics of a material are of especial importance in sheet metal forming operations where the ability of the sheet to work harden and thereby pass the deformation along from element to element is critical. This is only one example of many problems where the true stress-strain curve, which is based on actual physically existent stresses and strains rather than fictitious ones, provides a more rational approach to the understanding of material behavior under loading.

The significance of the ductility properties determined in the tension test with respect to service is much more difficult to interpret. Gillette has presented a very capable discussion of this question in the American Society for Testing Materials "Symposium on The Significance of the Tension Test of Metals in Relation to Design." Tensile ductility values are rarely used in design, primarily because it is indeed seldom that the amount of ductility needed for a certain service application is known. Furthermore, even if such a value were known, it is doubtful if it could be translated into terms of ductility measured in a tension test. As already mentioned, ductility specifications are almost universally based on engineering judgment and experience. Because of the many uncertainties in interpretation with our present state of knowledge, however, it must be recognized that the significance of tensile ductility is somewhat limited.

Gillette mentions a number of categories of service which require that the materials used possess a certain amount of ductility. The first example is that of plastic forming, especially where operations such as bending or deep drawing are involved, and the deformations may be sufficiently large to substantially exhaust the capacity of the material to deform. In attempting to apply tension-test ductility data to analyses of formability, it is important that care be taken to interpret the ductility data in the proper manner. Many attempts have been made to correlate the percentage elongation in 2 inches in the tension test with formability, and the general lack of success is well known. The total elongation in the tension test can only be considered as a rough indication of relative ductility to be expected in actual drawing operations. One obvious reason for the failure of the total elongation in the tension test to correlate with drawability lies in the fact that, as already pointed out, the total elongation value includes both the uniform and the local or necking elongation. In a stretching type of sheet metal forming operation, it is more logical to expect a correlation of uniform elongation with performance. Once a pronounced necking down or localization of deformation occurs in a

forming operation, the useful limit of elongation has been reached, since the material at positions removed from the neck can no longer contribute to the overall deformation necessary to successfully form the part.

The problem of bending provides another example of the lack of fundamental significance of total elongation in the tension test. In bending, the local elongation at fracture is of critical importance, and there is some evidence that the reduction of area in the tension test, which is a measure of the ability to deform locally, correlates with the ability of a material to be bent. This correlation is based on the fact that the peak elongations at fracture in the tension fibers of a bend approach the values of the local elongation in the tension test for the same material, provided the comparison is made on a true strain basis.

The second example of service cited by Gillette, which involves ductility, is that in which the normal service calls for plastic extension. In many structural applications, a small amount of local plastic extension may be very important in relieving local stress concentrations, thereby possibly preventing rupture. Readjustment of local stresses by local plastic flow may also occur in cyclic loading applications. Although the availability of sufficient ductility to permit these readjustments is very important, it is extremely difficult to predict just what amount of ductility is adequate. Again, the engineer must call upon his past experience and best judgment.

Ductility is sometimes demanded as an "insurance factor," that is, extra protection in the event of accidents or overloads. The ability to deform and absorb such overloads without rupture may be desired, although again the necessary amount of ductility may be unpredictable. Actually, what is required in this class of applications is the ability to absorb energy, or toughness, an attribute which depends not only on ductility but on strength as well. Toughness will be discussed in more detail in a subsequent section dealing with impact testing and notch toughness.

In considering the significance of ductility, it is important to recognize that ductility in the tension test, especially as measured by reduction of area, is strongly dependent upon microstructure and is considered, therefore, to be a "structure sensitive" property. The tensile and yield strength, on the other hand, are less structure-sensitive. Tensile ductility, therefore, may provide a useful means of detecting the presence of undesired microstructures, particularly in heat-treated steels, which could result in inferior mechanical performance.

In summary, it can be said that the tension test, if properly conducted and interpreted, is a very informative and versatile test, providing information on both the strength and ductility properties of materials. In addition to the direct application of some of the tensile properties in design, the wealth of practical experience built up around the tension test makes it extremely useful in specifying materials for particular applications as well as in the control of the uniformity of material supplied for those applications. This practical experience also makes the tension test an extremely useful tool in the development of new and improved materials for applications on which previous experience is available.

SECTION 3

HARDNESS TESTING

Hardness is a material characteristic which can perhaps best be defined in terms of resistance to deformation. The degree of hardness of a material can be

manifested in a number of different ways depending upon the conditions to which the material is subjected. In metals, the most commonly used measure of hardness

depends upon the resistance to penetration by a much harder body. Hardness may also be manifested as a resistance to abrasion or wear, as a resistance to cutting, as a resistance to crushing, as a resistance to deformation as in tension or compression, as a manifestation of resilience, i.e., rebound hardness, and others. In this discussion, attention will be focused on the types of hardness tests which measure the resistance to penetration under certain specified conditions, since these are by far the most widely used.

The extremely wide use of hardness tests, especially in conjunction with the making, shaping, and treating of steel, can be attributed not only to the extreme simplicity of sample preparation and test procedure, but also to the close relationship between hardness and other mechanical properties. The best example of the correlation between hardness and other mechanical properties is provided by quenched and tempered steels, where a hardness measurement permits a good estimate of most of the other mechanical properties. Hardness tests are especially well adapted to checks of uniformity of product, because of the great ease with which they can be made. If a process or treatment passes out of control, the departure from uniformity can frequently be detected in hardness changes in the product. It is primarily because of this usefulness in control of uniformity that hardness testing is used so extensively in the steel industry.

The two hardness tests used most widely are the Brinell test and the Rockwell test, each of these tests having been standardized by the American Society for Testing Materials. Because of their universal usage, these two tests will be described in considerable detail. Other types of hardness tests which have certain specialized applications will also be discussed briefly.

THE BRINELL HARDNESS TEST

In the Brinell hardness test, which was proposed by Dr. J. A. Brinell of Sweden around 1900, a spherical ball, usually made of hardened steel, is forced into the specimen under a definite static load. The size of the resulting indentation provides a measurement of hardness as it is manifested under the particular conditions of the Brinell test.

A Brinell hardness tester consists of a device for applying a predetermined static load to the specimen through the indenter. One of the most commonly used types of Brinell machines is shown in Figure 49—15. In this machine, the load is applied hydraulically, and a weighted yoke is provided to prevent the maximum load desired from being exceeded. The yoke, which carries weights proportional to the desired load, acts on a small piston in the hydraulic system. As the pressure is increased by pumping, the load on the indenter and on the yoke piston will gradually increase until the yoke and weights float, indicating that the desired load has been attained. As long as the weights float, the load will remain constant. A Bourdon tube is usually provided for the purpose of giving an indication of the rate of loading and the approach to the desired testing load. If the parts of the hydraulic system are well made and the pistons accurately lapped, this type of machine will provide a very accurate load application and is much to be preferred to the use of a Bourdon gage alone.

The standard indenter for the Brinell test used in this country is a 10-millimeter spherical ball which is usually made of hardened steel. For tests on extremely hard materials, cemented carbide balls may be employed. According to the ASTM Standard for the

Brinell test (A370-54T), a ball must not exhibit a permanent change in diameter greater than 0.01 mm. (0.0004 in.) when pressed with a force of 3000 kg. against the test specimen.

Once an indentation has been made, either its depth or diameter must be measured in order to obtain the Brinell hardness number. Measurement of depth of indentation, however, has never been approved as a standard method. The diameter is usually determined by a special measuring microscope which is fitted with a glass scale graduated in tenths of a millimeter. The depth of the indentation can be measured by a special device attached to the indenter. Although measurements of indentation depth can be made very rapidly, it has been found that, in general, hardness values determined on the basis of depth measurements are less reliable than those determined from diameter measurements. Unless great speed is desired, therefore, the diameter of the indentation should be used in determining the Brinell hardness number.

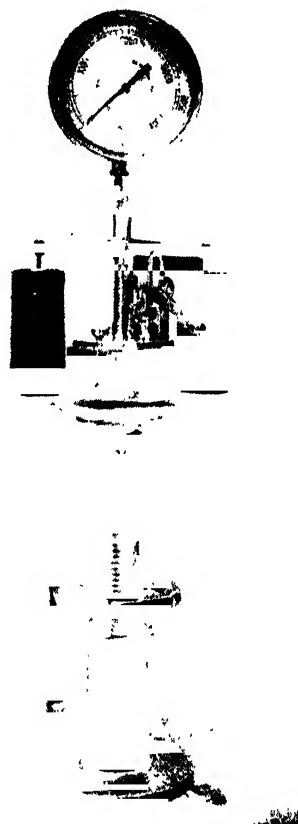


FIG. 49—15. Brinell hardness testing machine. (Courtesy, Tinus Olsen Testing Machine Company.)

Brinell-Testing Technique—In selecting and preparing the specimen for use in the Brinell test, a number of precautions should be observed. First of all, the specimen must be thick enough that no anvil effect is encountered. After a test, the side of the specimen opposite the impression must show no effect from the loading such as a local bulging. In order to avoid such effects, which may lead to fictitious hardness values, the thickness of the specimen should never be less than ten times the depth of the impression. Care should also be

taken that an indentation is not made too near the edge of the specimen. A distance from the specimen edge of not less than 2.5 times the diameter of the indentation should prove sufficient to eliminate edge effects. The specimen should be flat, and its surface should be sufficiently smooth that the periphery of the indentation appears sharply defined under the measuring microscope. Another precaution which should be observed is concerned with spacing of multiple indentations. A spacing of at least two indentation diameters should be used in order to avoid testing metal which has been disturbed by a previous indentation.

In the United States, the load for the Brinell test on iron and steel has been standardized as 3000 kilograms; for softer materials, a load of 500 kilograms may be used. In conducting the test, the prepared specimen is placed on the anvil, which is raised until the specimen is in contact with the penetrator ball, and the load applied as smoothly as possible. The load is held for at least 10 seconds in the case of ferrous materials and for at least 30 seconds for softer metals. In order to be acceptable by ASTM standards, the load-measuring device should indicate actual loads within 3 per cent tolerance.

In measuring the diameter of the indentation, most satisfactory results may be obtained by measuring in at least two directions and using the average in determining the hardness number. If the indentation is not circular, a directional variation in hardness may be present in the material being tested. On the other hand, out of roundness of the indentation may also indicate deformation of the indenting ball. The ASTM Standard requires that if a ball is used in testing a specimen of greater than 500 Brinell hardness number, it should be checked for permanent set after each indentation.

The Brinell hardness number is given by the quotient of the applied load and the surface area of the indentation, i.e.,

$$\text{Brinell Hardness Number (BHN)} = \frac{P}{A},$$

where P is the applied load in kilograms, and A is the area of the surface of the indentation expressed in square millimeters. It is important to note that the area referred to is the actual surface area and not the projected area of the indentation. If " D " is the diameter of the ball indenter, and " d " is the diameter of the indentation as measured with the Brinell microscope, the Brinell hardness number will be given by the relationship

$$\text{BHN} = \frac{\pi D}{2} \left(D - \sqrt{D^2 - d^2} \right)$$

where " P " is expressed in kilograms and " D " and " d " are expressed in millimeters. Tables have been calculated from this equation for the standard test conditions of 500-kilogram and 3000-kilogram loads and the 10-millimeter ball. Hardness numbers are tabulated for a wide range of impression diameters, so that it is merely necessary to locate in the table the diameter measured in the test and to read the corresponding number.

It may sometimes be desirable to obtain Brinell hardness values on very small specimens or thin specimens in which the standard loads and indenter would be too large to obtain a satisfactory test from the point of view of anvil or edge effects. In such instances, it is possible to obtain an approximate Brinell hardness number by reducing the size of the ball indenter, and at the same time reducing the applied load in proportion to the

square of the reduction in diameter of the ball. In other words, comparable test results should be obtained with different sizes of indenting balls, provided the applied loads are in the same ratios as the squares of the diameters of the indenting balls. For iron and steel, where the standard conditions call for a 3000-kilogram load and a 10-millimeter ball, the load " P " which should be used for a ball diameter " D " will be given by the equation:

$$\frac{P}{3000} = \frac{D^2}{10^2}$$

or

$$P = 30 D^2$$

A test carried out under other than the standard conditions is not considered as standard by the ASTM, but is merely recommended as an alternate if the specimen size prohibits a standard test. From Table 49—I, however, it can be seen that it is possible to obtain consistent results under a wide range of test conditions.

Table 49—I. Results of Brinell Hardness Determinations Using Various Loads and Sizes of Indenters

Diameter of Ball, mm.	Load, kg.	Diameter of Impression, mm.	Brinell Hardness Number
10.00	3,000.0	6.300	85
7.00	1,470.0	4.400	85
5.00	750.0	3.130	87
1.19	42.5	0.748	86

In considering the Brinell test, it should be remembered that the deformation of the indenter under load, in addition to a certain amount of recovery of the indented metal, prevents a perfectly spherical surface from being formed. When the indenting ball is pressed into the specimen, it tends to flatten to an extent which depends on the magnitude of the applied load, and thus creates a larger diameter of indentation than would have resulted had no deformation of the indenter occurred. This circumstance, of course, causes some error in the hardness tables, particularly at higher hardness levels. One of the principal reasons for standardizing on the size and characteristics of the indenting ball and on the magnitude of the applied load has been to circumvent this difficulty; in reality, an attempt has been made to standardize the error. In the "Hardness Conversion Table for Steel" presented in the ASTM Standards for Ferrous Metals, the effect of varying amounts of indenter deformation can be seen. Brinell hardness numbers are shown for standard steel balls, Hultgren balls (a cold-worked steel ball), and carbide balls for a 3000-kilogram load. Up to 433 Brinell hardness, the values agree for all three balls. Above 433 BHN, the carbide ball deforms least and indicates the greatest hardness of any of the indenters. When the indicated hardness with the carbide ball is 517, it is only 505 with a standard steel ball. Above 505 BHN no hardness values are given for the standard steel ball since in tests on harder materials it no longer conforms to the requirements for permanent set. However, a comparison of the Hultgren ball with the carbide ball is given at a considerably higher hardness level. When the carbide ball indicates 656 BHN, the Hultgren ball results in a value of only 615. Undoubtedly, if a material such as diamond were used as an indenter, a still higher hardness would have been indicated. In attempting to compare results of tests above 433 BHN, therefore, it is important to take into account the characteristics of the balls used in making the indentations. The type of ball used in tests at

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high hardness levels should always be designated in reporting test results.

THE ROCKWELL HARDNESS TEST

The Rockwell hardness test, like the Brinell test, measures that aspect of hardness which manifests itself as a resistance to penetration. Because of its simplicity, accuracy, and extreme versatility, the Rockwell test is more widely used today than any other type of hardness test. A wide variety of testing conditions is available, which permits testing over a wide range of hardnesses and also permits testing of very thin materials. In the Rockwell test, in contrast to the Brinell test, the hardness numbers do not bear a mathematical relation to diameter of indentation but are dial divisions, which indicate the depth of impression. Much of the inaccuracy associated with a measurement of total indentation depth is eliminated by the use of a differential depth measurement. The indenter is first seated by a minor load, after which a standardized major load is applied. It is the increment in indentation depth produced by the major load over that produced by the minor load which provides the basis of the Rockwell hardness. In this manner, the effects of small surface irregularities and surface disturbances caused by the indentation itself are eliminated and a very reproducible measurement is made possible.

The Rockwell hardness machine which is shown in Figure 49—16 is a precision-built apparatus which per-



FIG. 49—16. Rockwell hardness testing machine. (Courtesy, Wilson Mechanical Instrument Div., American Chain and Cable Co., Inc.)

mits the application of accurate, predetermined loads to standardized indenters, as well as a device for measuring the depth indentation produced. The load is applied through a system of weights and levers, and the rate of loading is controlled by a dashpot mechanism which provides a smooth, steady application of load. A dial gage which indicates depth directly during the test is provided for measurement of indentation depth, thus eliminating a separate operation such as required in the Brinell test. On the normal Rockwell machine, one dial division is equivalent to 0.002 millimeter penetration.

The penetrators which are most frequently used are

the spheroconical diamond penetrator and the $\frac{1}{16}$ -inch spherical steel ball, which are designated as the C-scale and B-scale penetrators respectively. The C-scale penetrator consists of a conical portion with a spherical tip lapped tangent to the cone. The angle of the cone is 120 degrees, and the radius of the spherical tip is 0.200 millimeter.

Specimen preparation for the Rockwell test is somewhat more critical than for the Brinell test, since the size of the indentation is much smaller. The surface of the specimen should be smooth, clean, and dry, and if a high degree of accuracy is desired, polishing through 2/0 or 3/0 metallographic paper is advisable. The surface to be tested should be free of scale and other foreign particles. The bottom surface should be reasonably flat, parallel to the test surface, and should also be free of scale or other matter which would tend to crush under the applied load. Pitted surfaces should be avoided. The presence of oil on the test surface will also tend to cause a low reading. The thickness of the specimen should be sufficient to prevent any anvil effect. It is desirable that no effect of the indentation be evident on the back side of the specimen, although if any such effect is not too pronounced, the hardness reading may not be greatly affected. It has been found in tests on specimens of commercially pure iron of Rockwell hardness B-35 that the readings were not affected by thickness down to a thickness of 0.040 inch, although the impressions showed through the specimen at 0.060-inch thickness. Although no definite procedure is specified by the ASTM, it is recommended that caution be exercised with respect to "anvil effect."

When the specimen has been prepared, it is placed on the anvil of the machine ready for testing. The anvil is raised slowly until the indenter is contacted, and continued to be raised until the minor load is applied. The small pointer on the machine dial shown in Figure 49—17 indicates the application of the minor load. When

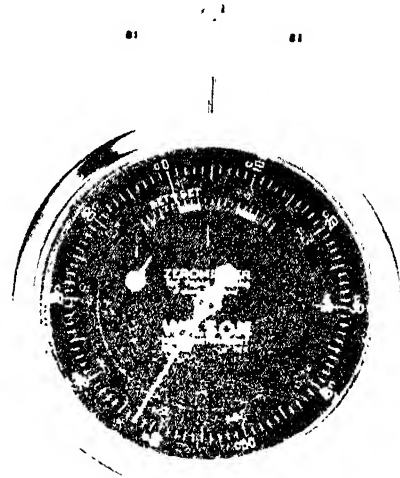


FIG. 49—17. Indicating dial on Rockwell hardness testing machine. (Courtesy, Wilson Mechanical Instrument Div., American Chain and Cable Co., Inc.)

the minor load has been applied, the hardness dial is set to zero on the scale being used, and the major load is applied. The rate of application of major load, which can be controlled by adjustment of the dashpot mecha-

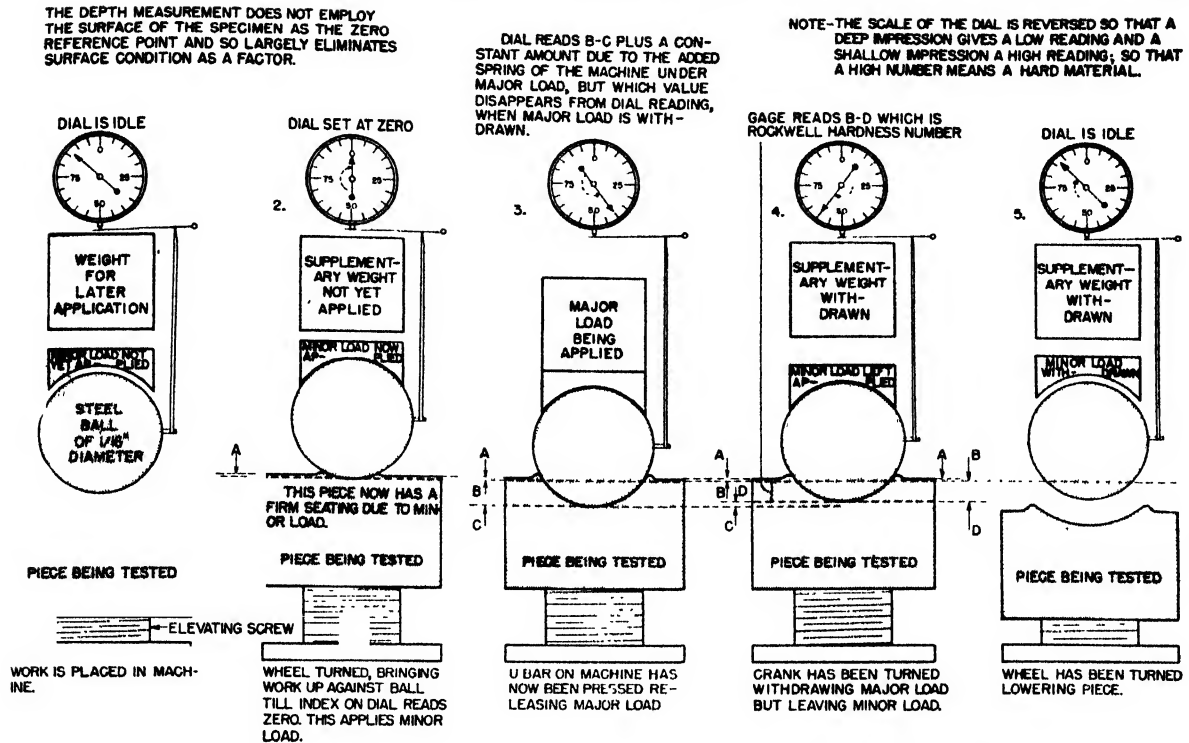


FIG. 49—18. Sketches illustrating principle of operation of Rockwell hardness testing machine. (Courtesy, Wilson Mechanical Instrument Div., American Chain and Cable Co., Inc.)

nism, is very important and should be checked periodically. The ASTM recommends that the travel of the operating handle of the machine be completed in four to five seconds, with a major load of 100 kilograms and with no specimen in the machine. The duration of major load application is also important, and should be controlled according to the procedures described in ASTM Specification E 18-55T.

When the major load is removed, and with the minor load still on, the hardness is read directly from the dial which is an integral part of the Rockwell machine. The scale of this dial is calibrated to read Rockwell hardness number directly, as already indicated. The dial is divided into 100 equal divisions on each of two scales, the "B" and "C" scales. In order to avoid confusion, the B-scale is printed in red, while the C-scale is printed in black. The B-scale also differs in that the zero is shifted counterclockwise 30 hardness points so that the marking for hardness of B-30 corresponds to that of C-0. In using other Rockwell scales on the standard machine, the red scale is always used with a ball penetrator and the black scale with the C-scale penetrator. The scale of the dial is numbered in such a way that low numbers correspond to deep impressions and relatively soft material, whereas a high hardness is indicated for a shallow indentation in harder material. The dial scale can be rotated manually and can always be set exactly on zero just prior to application of the major load. This arrangement permits a direct reading of the increment of penetration produced by the application of the major load over that produced by the minor load.

The principle of operation of the Rockwell testing machine can be seen from the series of sketches shown in Figure 49—18, which indicates the sequence of steps in a test in which the B-scale is employed. The operation is identical for other scales, with the exception of variations in major load and indenter.

Table 49—II. Rockwell Hardness Scales

Scale Symbol	Penetrator	Major Load, kg.	Dial Figures
Group One			
B	$\frac{1}{16}$ -in. ball	100	Red
C	Sphero-conical diamond	150	Black
Group Two			
A	Sphero-conical diamond	60	Black
D	Sphero-conical diamond	100	Black
E	$\frac{1}{8}$ -in. ball	100	Red
F	$\frac{1}{16}$ -in. ball	60	Red
G	$\frac{1}{16}$ -in. ball	150	Red
H	$\frac{1}{8}$ -in. ball	60	Red
K	$\frac{1}{16}$ -in. ball	150	Red
Group Three			
L	$\frac{1}{4}$ -in. ball	60	Red
M	$\frac{1}{4}$ -in. ball	100	Red
P	$\frac{1}{4}$ -in. ball	150	Red
R	$\frac{1}{2}$ -in. ball	60	Red
S	$\frac{1}{2}$ -in. ball	100	Red
V	$\frac{1}{2}$ -in. ball	150	Red

In Table 49—II, the various standard Rockwell scales are listed. As already indicated, the "B" and "C" scales are most widely used. The B-scale is used in what might be considered a medium hardness range, and is especially useful in testing low-carbon and medium-carbon steels in the annealed condition. The $\frac{1}{16}$ -inch diameter hardened steel ball, used as the penetrator in the B-scale test, is carried in a special chuck which permits rapid change of balls. The ball must not differ by over ± 0.0001 inch from its nominal diameter, and must not show a variation of over ± 0.00002 inch in diameter within itself.

The C-scale is the scale used most frequently for hardness above C-20. There is some overlapping in a number of the Rockwell scales, and for the sake of accuracy, it is desirable to select a scale such that the test value will fall in the middle of the scale range. In making B-scale tests, the steel ball tends to flatten at hardnesses of 100 and above, while at very low hardness near B-0, the impression is so deep that the cap holding the ball may contact the specimen surface and affect the reading. Likewise, in using the C-scale indenter at hardnesses below C-20, the impression is so deep that inaccuracies which may exist in the upper portion of the cone may affect the readings.

Rockwell Superficial Hardness Tests—It is frequently necessary to obtain a hardness value under conditions which prohibit the use of other than an extremely shallow indentation. The Rockwell superficial hardness tester has been developed for such applications. This test operates on the same principle as the regular Rockwell test, but utilizes much lighter loads and a more sensitive dial gage. The superficial test is particularly useful in hardness determinations on very thin strip, on nitrided or lightly carburized surfaces, and on very small parts or parts shaped in such a manner that they would collapse under the heavy load used in the standard test. The small indentation is also frequently useful in obtaining hardness readings very close to the edge of an object where an edge effect would affect the reading in a regular test.

Two types of indenters are used in superficial hardness testing. One indenter is the same $\frac{1}{16}$ -inch diameter ball used in the standard test, and is used in superficial tests on the softer metals, such as brasses, bronzes, and unhardened steel. The superficial hardness scales using this indenter are designated by "T." In superficial tests on harder materials, a diamond penetrator having the same configuration as the standard C-scale penetrator is used, the only difference being a more accurate finishing to final dimensions. This penetrator is designated as the "N diamond," and the corresponding hardness scales are referred to as "N" scales. Table 49—III shows the various superficial hardness scales, with the indenters and major loads used in each. It will be noted that major loads of 15, 30, and 45 kilograms are used. The minor load in every instance is 3 kilograms. Each scale division

on the hardness dial represents 0.001 millimeter penetration, as compared to 0.002 mm. on the regular dial.

Specimen preparation for the superficial test is very critical, and polished surfaces are advisable. Specimens should be flat and free from dirt and foreign matter on both upper and lower surfaces. Test blocks are available for standardization as for the regular Rockwell test.

Superficial hardness numbers are prefixed by a number indicating the major load used and a letter designating the indenter. For example, if a reading of 42 were obtained in a test using the N-diamond and a 30-kilogram major load, the hardness would be designated by Rockwell 30N-42. If the $\frac{1}{16}$ -inch ball had been used, on the other hand, the hardness would be indicated as Rockwell 30T-42.

THE VICKERS OR DIAMOND PYRAMID HARDNESS TEST

The Vickers hardness test is another of the class of tests which measures resistance to penetration. It is similar in principle to the Brinell test, but utilizes a different indenter and different magnitudes of loads. The indenter used in the Vickers test is a square-based diamond pyramid, and the hardness value obtained when using this penetrator is frequently referred to as the diamond pyramid hardness. The angle between opposite faces of the pyramid is 136 degrees, which was chosen so that the Vickers hardness scale would correspond approximately to the Brinell scale. This choice is based on the fact that it is recommended that loads be used in the Brinell test which result in indentations having diameters in the range of 0.25 to 0.50 times the ball diameter. The average of this range is 0.375 times the ball diameter, and if tangents are constructed to an impression of this size at the specimen surface, it will be found that the angle between the tangents is 136 degrees.

In making the Vickers test, the indenter is forced into the specimen and the diagonals of the square impression measured and averaged. From the known geometry of the indenter, the surface area of the indentation can be calculated once the diagonals have been measured. The diagonals are measured rather than the sides of the impression in order to obtain greater accuracy. The diamond pyramid hardness number is then calculated as the ratio of the applied load to the surface area of the impression. For the 136-degree square-based pyramid, the hardness can be calculated from the formula:

$$\text{Diamond pyramid hardness} = 1.854 \frac{D}{P}$$

where "P" is the load in kilograms applied in making the indentation, and "D" is the average of the measured diagonals of the indentation expressed as millimeters.

The principal advantage of the diamond pyramid type of test is that geometrically similar indentations are always obtained regardless of the load applied. This useful characteristic of the impression geometry is not obtained with the spherical indenter, since the ratio of impression diameter to depth varies with the actual depth of the impression. In the discussion of the Brinell test, it was shown that, in order to make tests on thin material, it is not possible merely to decrease the load, but that it is also necessary to decrease the diameter of the indenter in proportion to the square root of the change in load in order to obtain a comparable hardness value. Since geometrically similar indentations are always obtained with the pyramidal indenter, decrease in load permits a satisfactory test on thin material, thus permitting the test to be applied over a wide range of thicknesses and over a wide range of hardnesses.

Table 49—III. Rockwell Superficial Hardness Scales

Major Load, kg.	Prefix Symbols				
	N or Diamond Scale	T Scale, $\frac{1}{16}$ -in. Ball	W Scale, $\frac{1}{16}$ -in. Ball	X Scale, $\frac{1}{16}$ -in. Ball	Y Scale, $\frac{1}{16}$ -in. Ball
15	15 N	15 T	15 W	15 X	15 Y
30	30 N	30 T	30 W	30 X	30 Y
45	45 N	45 T	45 W	45 X	45 Y

Specimen surface preparation is very important for the Vickers test, and for very light loads should approach a metallographic polish. It is also recommended that the thickness of the specimen be at least 1.5 times the diagonal of the indentation. The Vickers test is especially useful at high hardness levels because the diamond indenter deforms very little as compared to the balls used in the Brinell test. Up to about 300 Brinell hardness, Vickers numbers and Brinell numbers are practically identical. Above 300, the Vickers number becomes appreciably higher, partially because of the deformation of the Brinell ball and partially because of the shallowness of the Brinell impressions, which causes the 136-degree conical impression to be no longer comparable to the spherical impression.

For certain types of research work where especially reliable hardness values are desired, the Vickers test has seen considerable use. For general utility, as in rapid production testing, however, the Rockwell test is more commonly used.

MISCELLANEOUS HARDNESS TESTS

When a property is difficult to define and arbitrary tests must be devised to measure it, almost certainly a multitude of tests will be proposed. Hardness is no exception, and a great deal has been written about the various hardness tests which have been devised. Each of these tests attempts to measure some particular aspect of hardness which the proponent of the test considered to be of importance. A few of the more important miscellaneous methods of measuring hardness will be mentioned briefly primarily for purposes of identification.

a. **The Shore Scleroscope Test**—This test, which operates on the rebound principle, was introduced commercially about 1907, shortly after the advent of the Brinell test. It was used to a considerable extent for many years as a supplement to the Brinell test. The advantages claimed for the scleroscope are its portability and the small size of indentation made. The portability feature permitted the testing of massive objects which otherwise could not be readily tested. A small smooth spot was prepared, the scleroscope placed over it and a reading taken. The small impression made by the tester also made it suitable for testing finished articles on which a large indentation would have been undesirable.

The scleroscope itself consists of a small diamond-tipped hammer enclosed in a glass tube which is provided with a suction bulb whereby the hammer may be raised to the top of the tube and dropped from a fixed and predetermined height. When the hammer is dropped on the object being tested, it rebounds to a height which is considered as a measure of hardness. If the impact were perfectly elastic, the hammer would rebound to its original height. A slight amount of energy will be dissipated in deforming the specimen, however, and this energy is not available for the rebound. A scale is provided on the instrument for measuring the height of the rebound, the units of the scale being obtained by dividing the average rebound from quenched high carbon steel into one hundred equal parts. In one model the height of rebound is determined by careful observation, while another model is equipped with an indicating dial.

In making the test, precautions must be taken that the specimen is solidly supported, the sound of the impact providing some indication of the solidity. The specimen surface must, of course, be smooth and flat. The tube of the scleroscope must be vertical and the surface being tested must be horizontal so that true vertical impact and rebound can be obtained.

The scleroscope is not used as much as in the past

because the advantages once held over the Brinell test do not hold over newer types of tests. Furthermore, the nature of rebound hardness is not as well understood as indentation hardness. Perhaps the principal use of the scleroscope test has been in testing the surface hardness of rolls for rolling mills. Most other testing machines have the disadvantage that the throat is not sufficiently large to accommodate a large roll, whereas the scleroscope can be placed directly on the roll surface.

It is of interest to note that the rebound principle has also been utilized in establishing a go-no go gage for the hardness of steel balls. Balls are allowed to fall from an incline onto a hardened steel anvil. The balls rebound into two bins, one placed above the other, so that hard balls rebound into the top bin and softer balls rebound into the lower bin.

b. **The Penetrascope**—This instrument is a portable hardness tester which is similar in principle to the Vickers test in that a square-based 136-degree pyramidal diamond is utilized as an indenter. The operation of the Penetrascope consists of application of a load on the specimen through the indenter and the measurement of the diagonal of the resulting square indentation. The load is applied by a hydraulic thrust unit and can be varied through a range up to 40 kilograms. The measurement of the diagonal is made using a microscope with an adjustable opening consisting of movable knife edges. The adjustment of the knife edge opening operates a micrometer counter so that the diagonal can be measured to an accuracy of 0.001 mm. The diamond pyramid hardness number is obtained from the length of the diagonal by reference to a table.

The indenter is mounted on a turret along with the microscope for alignment of either component over the test area. The diamond indenter may be interchanged with one and two mm. diameter ball-type indenters.

The instrument is attached to the work with a chain clamp, "C" clamp or an electromagnetic base. With these clamping devices the instrument can be adapted for testing various dimensions and shapes.

The usefulness of the Penetrascope lies in the fact that with it a tool is available in the shop having the reliability and simplicity of the Vickers test plus the adaptability of securing diamond pyramid hardness results directly from the product. These features are particularly advantageous in inspection of those products having special requirements of relatively high surface hardness, such as the working surface of cold-reduction rolls or teeth in hardened gears. Since the instrument provides a means of measuring hardness directly on the surface of the product it also furnishes a means of not only maintaining process control but also of improving the effectiveness of the control of variables that influence surface hardness.

c. **File Hardness**—Hardness testing with a file is an old and crude method of measuring relative hardness, but one which can be useful in the hands of an experienced operator. A standard file is rubbed against the surface to be tested. If the file does not bite, the piece is designated as file hard. The test is, of course, greatly dependent upon the human factor and can only be considered useful where relatively large differences in hardness are of interest.

d. **The Monotron Hardness Test**—In this test, the load required to produce an indentation of fixed depth is measured. The standard penetrator is a hemispherical diamond of 0.75-millimeter diameter which is forced into the specimen to a depth of 0.045 millimeter. The hardness is read in Brinell units directly from a dial on the machine while the load is on the specimen. The test can be made on a surface which has had no special prepara-

tion by using a preloading arrangement which seats the indenter. The basic principle of the Monotron test is that the same size of indentation is always obtained so that the strain distribution in the specimen is always the same. The contribution of work hardening to the measured hardness is thereby controlled to some degree.

e. The Cloudburst Tester—The principal use of the cloudburst tester is in determining the uniformity of hardness over a relatively large area of surface. Essentially, the cloudburst process is simply a shot-peening process in which the intensity of peening can be adjusted. The surface of the object being tested is subjected to a rain of small balls $\frac{1}{8}$ inch in diameter. The force of impact is adjusted to a point so the impact just fails to indent material of the desired hardness. Softer areas will then be delineated by peening marks. The original cloudburst machine was also used as a shot peener for the purpose of surface hardening.

f. The Telebrineller (or Brinell Meter)—This instrument is a portable hardness tester consisting of a rubber anvil containing a 10-mm. Brinell ball that protrudes through the base of the anvil, and a $\frac{1}{2}$ -inch square steel bar, of known hardness, that is inserted into the anvil to back up the Brinell ball. To obtain a hardness number the tester is held in such a way that the ball is between the bar of known hardness and the specimen. The anvil is struck a sharp blow with a hammer and the diameters of the indentations made in the bar and the specimen are measured.

The Brinell hardness number of the specimen is then determined by multiplying the ratio of the diameter of impression in the test bar to the diameter of the impression in the specimen by the Brinell hardness number of the test bar. This calculation is done on a hardness calculator furnished with the instrument.

The instrument is used in determining the hardness of large castings, railroad rails, pipe, etc.

g. Microhardness Tests—Although the applications of microhardness testing are highly specialized, more and more applications are being found, and many special problems can be studied with this type of test. One of the primary uses of a microhardness tester is in fundamental studies of the hardness of various phases in the microstructure of a metal. By the use of very light loads, extremely small indentations can be placed in different phases in the microstructure and their differences in hardness determined. Extremely small scale variations in hardness such as variations across the diameter of very fine wire or across the thickness of very thin sheet can also be measured.

Several microhardness testers are on the market, but perhaps the most widely used of these is the Tukon tester. The indenter provided with this machine is

known as the Knoop indenter, a pyramidal type of indenter which was developed by the National Bureau of Standards. The indentation produced is a long, narrow, diamond-shaped impression. It is claimed that the advantage of this indenter lies in the fact that elastic recovery along the long axis of the indentation is very small, thus reducing variation from this source, which could be especially troublesome at very low loads. Hardness numbers are based on the long dimension of the indentation and are calculated as the ratio of the indenting load to the projected area of the indentation. An application of the Knoop indenter to a study of the variation of hardness in the various layers of a galvanized steel sheet is shown in Figure 49—19. The Tukon tester can also be fitted with the Vickers type of diamond pyramid, so that ordinary diamond pyramid hardness numbers can be obtained.

Another principle of microhardness measurement is utilized in the **microcharacter**. The microcharacter is a small, cubically pointed diamond mounted under a very light load. A scratch is produced on a polished specimen by drawing the specimen under the diamond, and the width of the scratch is measured under a microscope. Variations in scratch width indicate variations in hardness, the narrower the scratch the higher the hardness.

Hardness Conversion Tables—It is evident from the descriptions of the various methods of hardness testing, that each test possesses certain inherent advantages. The choice of test for a particular hardness determination will depend on a number of factors, including the size of specimen and the hardness level. Frequently it may be desirable to convert a hardness reading obtained on one scale, say Rockwell "C," to some other scale, say Brinell hardness, for purposes of comparison with other data. Hardness conversion tables have been prepared for this purpose. The Society of Automotive Engineers, American Society for Metals, and American Society for Testing Materials have jointly prepared a set of conversion data for steels harder than 220 BHN, which is presented in Table 49—IV. Considerable discretion must be exercised in making hardness conversions, and particular care must be taken with regard to testing details if reliable conversions are to be made. Hardness conversion tables for other metals, including austenitic stainless steels, can be found in the ASM Handbook.

Significance and General Utility of Hardness Tests—It has been seen that the most widely used hardness tests measure resistance to penetration under certain arbitrarily chosen conditions. In forcing a penetrator into a metal specimen, the metal in the vicinity of the penetration is plastically deformed, in order that the penetrator can be accommodated. The factors controlling the amount and distribution of this plastic deformation are very complex and no exact quantitative analysis of the penetration of a metal by any shape of indenter has been developed. Qualitatively, however, it can be seen that resistance to penetration is a measure of resistance to plastic flow and should be related in some manner to the stress-strain curve of the material being tested, i.e., to the strength and strain-hardening properties of the material. In general for steels, a good correlation has been found to exist between hardness and tensile strength. A particularly close correlation exists in quenched and tempered steels where, in addition, yield strength and ductility can also be predicted reasonably well from hardness measurements. For this reason, the hardness test is of particular value in the field of heat-treatable steels.

In the making, shaping, and treating of steel products, the most extensive use of the hardness test is for inspection and control of uniformity. A good example of



FIG. 49—19. Microhardness tests of various layers in galvanized sheet steel.

Table 49—IV. Hardness Conversion Numbers for Steel (see Notes below).

Rockwell C Hard- ness Number	Diamond Pyramid Hardness Number	Brinell Hardness Number			Rockwell Hardness Number		Rockwell Superficial Hardness Number				Sclero- scope Hardness Number	Approx. Tensile Strength, 1000 psi	Rockwell C Hard- ness Number
		10-mm Standard Ball, 3000-kg Load	10-mm Hultgren Ball, 3000-kg Load	10-mm Carbide Ball, 3000-kg Load	A Scale, 60-kg Load, Brinell Penetrator	D Scale, 100-kg Load, Brinell Penetrator	15-N Scale, 15-kg Load, Superficial Brinell Penetrator	30-N Scale, 30-kg Load, Superficial Brinell Penetrator	45-N Scale, 45-kg Load, Superficial Brinell Penetrator				
68	940	•	•	•	85.6	76.9	93.2	84.4	75.4	97	•	68	
67	900	•	•	•	85.0	76.1	92.9	83.6	74.2	95	•	67	
66	865	•	•	•	84.5	75.4	92.5	82.8	73.3	92	•	66	
65	832	•	•	739	83.9	74.5	92.2	81.9	72.0	91	•	65	
64	800	•	•	722	83.4	73.8	91.8	81.1	71.0	88	•	64	
63	772	•	•	705	82.8	73.0	91.4	80.1	69.9	87	•	63	
62	746	•	•	688	82.3	72.2	91.1	79.3	68.8	85	•	62	
61	720	•	•	670	81.8	71.5	90.7	78.4	67.7	83	•	61	
60	697	•	613	654	81.2	70.7	90.2	77.5	66.6	81	•	60	
59	674	•	599	634	80.7	69.9	89.3	76.6	65.5	80	•	59	
58	653	•	587	615	80.1	69.2	88.3	75.7	64.3	78	•	58	
57	633	•	575	595	79.6	68.5	88.3	74.8	63.2	76	•	57	
56	613	•	561	577	79.0	67.7	87.9	73.9	62.0	75	•	56	
55	595	•	546	560	78.5	66.9	87.4	73.0	60.9	74	•	55	
54	577	•	534	543	78.0	66.1	87.4	72.0	59.8	72	•	54	
53	560	•	519	525	77.4	65.4	86.9	71.2	58.6	71	•	53	
52	544	500	508	512	76.8	64.6	86.4	70.2	57.4	69	•	52	
51	528	487	494	496	76.3	63.8	85.9	69.4	56.1	68	•	51	
50	513	475	481	481	75.9	63.1	85.5	68.5	55.0	67	•	50	
49	498	464	469	469	75.2	62.1	85.0	67.6	53.8	66	•	49	
48	484	451	455	455	74.7	61.4	84.5	66.7	52.5	64	•	48	
47	471	442	443	443	74.1	60.8	83.9	65.8	51.4	63	•	47	
46	458	432	432	432	73.6	60.0	83.5	64.8	50.3	62	•	46	
45	446	421	421	421	73.1	59.2	83.0	64.0	49.0	60	•	45	
44	434	409	409	409	72.5	58.5	82.5	63.1	47.8	58	•	44	
43	423	400	400	400	72.0	57.7	82.0	62.2	46.7	57	•	43	
42	412	390	390	390	71.5	56.9	81.5	61.3	45.5	56	•	42	
41	402	381	381	381	70.9	56.2	80.9	60.4	44.3	55	•	41	
40	392	371	371	371	70.4	55.4	80.4	59.5	43.1	54	•	40	
39	382	362	362	362	69.9	54.6	79.9	58.6	41.9	52	•	39	
38	372	353	353	353	69.4	53.8	79.4	57.7	40.8	51	•	38	
37	363	344	344	344	68.9	53.1	78.8	56.8	39.6	50	•	37	
36	354	336	336	336	68.4	52.3	78.3	55.9	38.4	49	•	36	
35	345	327	327	327	67.9	51.5	77.7	55.0	37.2	48	•	35	
34	336	319	319	319	67.4	50.8	77.2	54.2	36.1	47	•	34	
33	327	311	311	311	66.8	50.0	76.6	53.3	34.9	46	•	33	
32	318	301	301	301	66.3	49.2	76.1	52.1	33.7	44	•	32	
31	310	294	294	294	65.8	48.4	75.6	51.3	32.5	43	•	31	
30	302	286	286	286	65.3	47.7	75.0	50.4	31.3	42	•	30	
29	294	279	279	279	64.7	47.0	74.5	49.5	30.1	41	•	29	
28	286	271	271	271	64.3	46.1	73.9	48.6	28.9	40	•	28	
27	279	264	264	264	63.8	45.2	73.3	47.7	27.8	39	•	27	
26	272	258	258	258	63.3	44.6	72.8	46.8	26.7	38	•	26	
25	266	253	253	253	62.8	43.8	72.2	45.9	25.5	37	•	25	
24	260	247	247	247	62.4	43.1	71.6	45.0	24.3	36	•	24	
23	254	243	243	243	62.0	42.1	71.0	44.0	23.1	35	•	23	
22	248	237	237	237	61.5	41.6	70.5	43.2	22.0	34	•	22	
21	243	231	231	231	61.0	40.9	69.9	42.3	20.7	33	•	21	
20	238	226	226	226	60.5	40.1	69.4	41.5	19.6	32	•	20	

*For the 10-mm standard steel ball, no values of Brinell hardness are given higher than 500 Bhn. This is in conformance with the limitations established by the Standard Method of Test for Brinell Hardness of Metallic Materials (ASTM Designation E 10).

"Brinell" is the trade name for the C-scale spheroidal diamond penetrator. The Superficial Brinell penetrator has the same configuration but is finished to closer tolerances. The values of scleroscope hardness number and approximate tensile strength are from the 1955 SAE Handbook. All other values are from ASTM Standard E 48-47.

such use is provided by the extensive use of hardness tests in the production of deep-drawing sheets. It is usually known from experience that the hardness must be within a certain range if the sheet is to successfully form a certain part. Because of the great ease of making

hardness tests on sheet products, very frequent tests can be made to assure that all the product is within the desired hardness range. Most uses of hardness tests are of a similar nature, depending upon a combination of ease of testing and correlation with other properties.

SECTION 4

NOTCHED-BAR IMPACT TESTS

The notched-bar impact test is, as the name implies, a test in which a bar containing a notch is supported as a beam and subjected to the impact from a moving mass having sufficient kinetic energy to break the bar. The energy absorbed by the specimen is measured by the loss in kinetic energy of the moving mass. Notched-bar impact tests were originally conceived with the idea of providing an indication of the ability of a material to absorb punishment under very severe service conditions. Notches or stress concentrations of one kind or another are present in most structures and, generally, are the origin of structural failures. It was suggested that the capacity of a material to absorb energy in the notched-bar impact test should indicate the ability of the material to absorb energy in the event of overloading in a structural member and the ability to resist failure at local stress concentrations. In recent years, the notched-bar impact test has taken on a more general significance, in that attention has come to be focused on the variation of energy absorption with temperature, rather than on the absolute level of energy absorption at a single testing

temperature. Many steels exhibit a rapid drop-off in the level of energy absorption in some critical temperature interval as the testing temperature is lowered. As will be discussed in more detail in a subsequent section, the temperature range in which this abrupt decrease in energy absorption occurs provides much more information about the notch-toughness characteristics of a steel than the energy absorption at a single temperature.

Impact-Testing Methods—The details of impact testing procedures are described in a tentative ASTM specification designated as E 23-47T. The most generally used impact testing machines are of the pendulum type such as shown in Figure 49-20. In this type of machine, the impact energy is furnished by the swing of a weighted pendulum. The available energy depends, therefore, on the effective mass of the pendulum and its velocity at impact. The point of impact is usually situated at the bottom of the swing, at which point the pendulum has attained its maximum velocity. The available kinetic energy at this point is equivalent to the potential energy just before the start of the swing. If the pendulum were allowed to swing through its arc freely without striking a specimen, it would rise to the height from which it had been released, neglecting the effects of friction and air resistance. If a specimen is struck, the pendulum loses the amount of energy required to break the specimen and will not swing to the full height from which released. The amount by which the pendulum fails to reach its original height provides an easy method for the measurement of the energy absorbed by the specimen.

The two principal types of notched-bar or notched-beam impact tests are the simple beam test and the cantilever beam test, the former being generally referred to as the Charpy type and the latter as the Izod type. The methods of supporting and striking the specimen in each type of test are shown in Figure 49-21. Actually, the simple beam, or Charpy type of test, is used more extensively because of its greater convenience for other than room-temperature tests. The Izod type of test is impractical for controlled-temperature tests. In the Charpy type of test, it is a simple matter to transfer the specimen from a controlled-temperature bath to the anvil support and to break it in a sufficiently short time that no appreciable temperature change occurs. The Izod test has the added disadvantage that even in room-temperature tests, each individual specimen must be clamped and unclamped in the support, causing a considerable loss of time.

The types of specimens which are used in the simple beam type of notched-bar impact test are shown in Figure 49-22. The keyhole-notch specimen is a very widely used impact specimen and that upon which most impact specifications have been based in the past. The V-notch specimen is coming into wider usage, however, and its use is now required in many impact specifications. The details of impact-test specimen preparation are of considerable importance in that, in some temperature regions of testing, the degree of scatter of test results may be increased by improper attention to specimen preparation. The most critical part of the impact

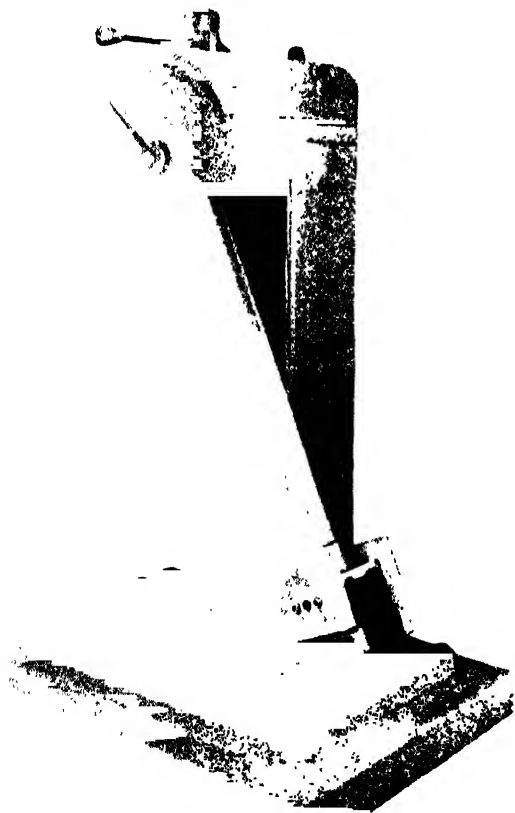
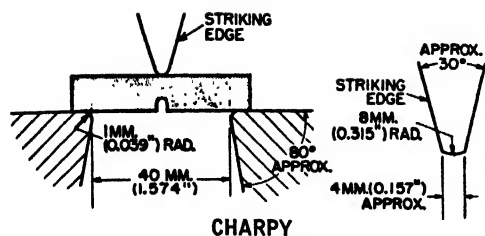
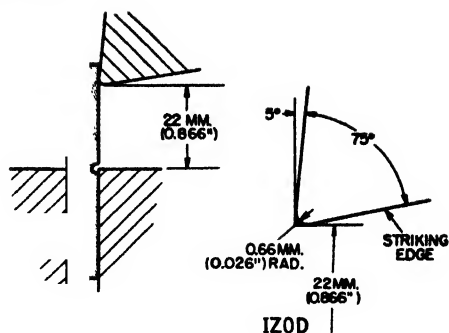


FIG. 49-20. Sonntag pendulum impact testing machine. (Courtesy, The Baldwin-Lima-Hamilton Corp.)



CHARPY



IZOD

FIG. 49—21. Method of striking and supporting Charpy and

specimen is the root of the notch, and it is especially important that the finish of the notch surface be free of pronounced machining marks and excessive cold work from the machining operation. The base of the notch should be accurately parallel to the back face of the specimen, and the specimen should be square, so that fracture will not start at a corner of the notch surface. In notching the keyhole specimen, a hole is drilled at the proper location and a slot sawed or milled to the hole. Special care should be exercised that the saw or cutter does not accidentally bite the bottom of the notch, since the notch contour will be drastically altered. In the V and U notches shown in Figure 49—22, special attention should also be given to machining marks, since such marks will be disposed parallel to the notch axis and thus are more effective as stress raisers.

In an actual impact test, the pendulum is raised to its release position, the specimen placed on the anvil, the pendulum released, and the energy absorption noted. Details of machine calibration and precautions to be exercised in the use of pendulum impact machines can be found in the ASTM Specification E 23-47T already mentioned.

Since a great deal of impact testing is conducted at low temperatures, it is important to consider some of the methods employed in obtaining controlled testing temperatures. As already mentioned, the simple beam type of specimen is best adapted to low-temperature tests, since it can be cooled in a bath of the desired tempera-

ture and readily transferred to the impact machine and broken in a very short time. Temperatures between 212° F and 32° F can be obtained by water baths properly heated or cooled. The volume of the bath should be relatively large compared to the volume of specimens introduced, in order to make the bath temperature more stable. The bath temperature is also stabilized by efficient insulation of the bath container. Stirring of the bath is recommended in order to minimize temperature gradients.

Temperatures from 32° F down to about -110° F can be obtained conveniently by the use of dry ice as a refrigerant. Acetone is frequently used as a bath with dry ice. The dry ice can be introduced directly into the acetone bath, the resulting bubbling action providing rapid temperature equalization. Below -110° F, it is customary to use liquid nitrogen as a refrigerant. The container carrying the specimen bath may be immersed in liquid nitrogen until the bath temperature is lowered to the desired point; an alternative procedure involves passing liquid nitrogen through a copper coil immersed in the bath to be cooled. If testing is being carried out in a well ventilated room, inflammable liquids may be used for the bath, provided proper fire precautions are taken. Iso-pentane is frequently used as a cooling bath, permitting temperatures down to about -255° F to be reached. It is important to point out that the use of liquid air in conjunction with inflammable liquids is extremely dangerous, and very severe explosions have occurred under these conditions. For optimum safety conditions when using either liquid air or liquid nitrogen, it is recommended that some non-inflammable bath be used for the lower temperature ranges of testing.

A series of commercial refrigerants marketed under the trade name of Freon possesses a number of excellent characteristics for low-temperature testing. Freon is non-inflammable, non-toxic, and non-corrosive. The particular Freon compound (CCl_2F_2) designated as Freon 11 has a freezing point of -168° F and a boiling point of about 75° F. Freon 12 (CCl_2F_2) has a freezing point of -252° F and a boiling point of -22° F. The principal disadvantage of this material is its low boiling point, which necessitates considerable special handling equipment. Freon 13 (CCl_2F_2) freezes at -296° F, thus permitting testing in the interval between the freezing point of Freon 12 and the boiling point of liquid nitrogen. Boiling liquid nitrogen is frequently used as a bath giving a testing temperature in the neighborhood of -315° F, and in a few instances boiling liquid hydrogen has been used for tests at -423° F.

For low-temperature tests, the bath should be located near the testing machine and arranged so as to permit the most rapid transfer of specimens from bath to machine. It is desirable to use tongs that are of the bath temperature to avoid conducting heat into the specimen.

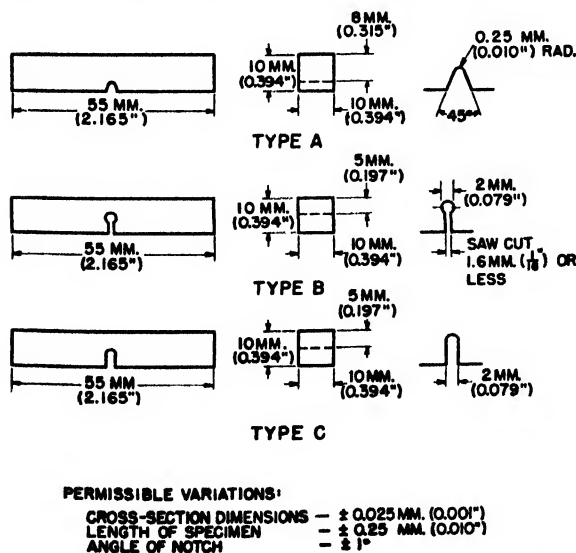


FIG. 49—22. Types of specimens for Charpy impact tests. (From "ASTM Standards, 1955.")

It should be possible to complete the test in from three to five seconds from the time of removal of the specimen from the bath, in which time the specimen temperature will have changed only very slightly. In tests at liquid nitrogen temperatures, the evaporation of the liquid on the specimen carried out of the bath may actually lower the surface temperature of the specimen to a slight extent.

Significance and Interpretation of Notched-Bar Impact Test Results—Before attempting to evaluate the significance of any notched-bar test, it is of the utmost importance to recognize that the behavior of a metal in such a test is strongly dependent on the geometry of test piece, rate of loading, and temperature of testing chosen as the basis of the test. Because of the extremely complex interrelationships among these variables, results obtained in some particular test cannot be translated directly, at least in our present state of knowledge, to a service application in which the conditions of loading are altogether different. Even if the energy-absorbing capacity needed in structures for safe performance could be estimated, which is rarely the case, the energy absorption in a notched-bar impact test could not be used directly as a design figure because of the lack of similarity of geometry and loading conditions. Impact specifications are, therefore, usually based on experience or engineering estimates, as has been seen to be the case in so many other mechanical-property specifications. Most impact-strength specifications are written in such a way as to specify the minimum energy absorption required, as, for example, 15 foot-pounds in the Charpy keyhole-notch test, at the minimum service temperature to be encountered. Although a steel passing such a specifica-

tion would probably perform better in service than one which did not pass, it must be recognized that passing does not necessarily assure the desired service performance. This uncertainty arises directly from the different geometry and loading conditions encountered in service, which are likely to be more severe than those in the small test piece used in the customary notched-bar impact tests.

The need for deriving a more general significance from notched-bar impact tests is obvious in view of such uncertainties as to the meaning of the results. The concept of the "transition temperature" does much to provide this more general basis of interpretation, not only for notched-bar impact tests, but for all other notch toughness tests. If a series of notched-bar impact tests is made on a ferritic steel at successively lower temperatures, it will be found that a temperature or temperature range will be reached where a very rapid decrease in energy absorption occurs. This is called the **transition temperature** or **transition temperature range**. Below this range, the energy absorption is very low, and the behavior is said to be brittle. Above the transition range, the energy absorption is relatively high, and the behavior is said to be tough. It is this transition from tough to brittle behavior that is now considered the crux of all notched-bar testing. The transition temperature provides the least uncertain basis for the comparison of the notch toughness characteristics of different steels. Two steels which would be considered alike in room temperature tests might be found to be markedly different, if compared on the basis of transition temperature, as can be seen in Figure 49—23. If tests are made at any single temperature above or below the transition range of both

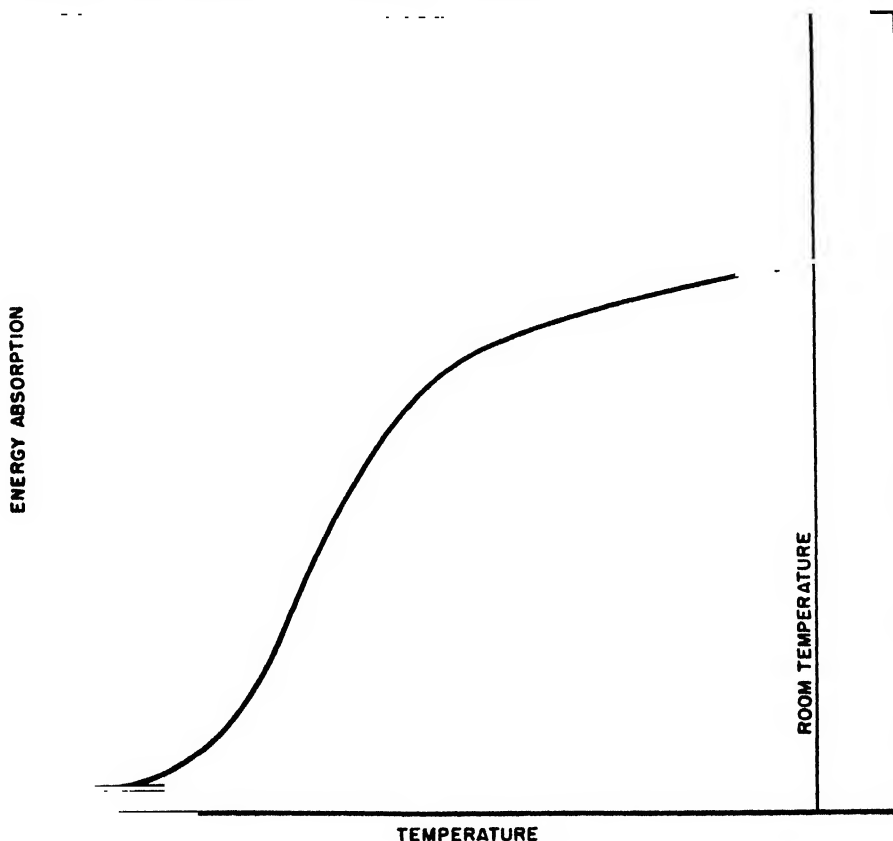


FIG. 49—23. Schematic transition curves for two steels having same energy absorption at room temperature.

steels, it can only be concluded that the two steels are alike.

What, then, is the significance of the transition temperature with respect to service behavior? In structures such as welded ship hulls or welded pressure vessels, where a high degree of continuity exists in the strength members, rapidly propagating brittle-type fractures have in a number of instances resulted in complete structural failures. Presumably, if the construction materials had possessed greater toughness or energy-absorbing capacity under the design and loading conditions existing at the time of failure, the initiation of the fracture would have been prevented. It can be said that for the particular conditions existing at the time of brittle failure, the material which failed was below its transition temperature. In practically all of the brittle types of structural failures, the material involved exhibited entirely normal properties in the ordinary tension test, including a large amount of ductility. In spite of this fact, very little evidence of ductility could be found in the structural failures.

As already indicated, the temperature range of transition from tough to brittle behavior is markedly dependent upon the degree of restraint or state of stress imposed on the material. Sharper notches tend to raise the transition temperature; i.e., brittle behavior can be expected to occur at higher temperatures. Since extremely sharp notches and high degrees of restraint exist in most structures, particularly in welded structures having a high degree of continuity, the range of transition from tough to brittle behavior for most service conditions lies at higher temperatures than in the Charpy keyhole-notch test. This fact is extremely important, since it immediately follows that tough behavior at a certain temperature in a Charpy keyhole-notch impact test does not necessarily insure tough behavior at the same temperature in service. Because of the greater notch severity, the Charpy V-notch impact test shows a higher transition-temperature range, and would, therefore, be expected to correlate more closely with actual service. This indeed appears to be the case, as evidenced by many tests on steels of known service performance. The Charpy V-notch test is coming to be used more widely, and may well replace the keyhole-notch test in many specifications.

In attempts to correlate notch tests with service as well as with one another, it is essential that two types of transition behavior be recognized. In notched test specimens, a change in fracture appearance from a fibrous (shear mode) to a granular (cleavage mode) fracture occurs over a certain temperature range as the testing temperature is lowered. The temperature at some arbitrary point within this temperature range has been called the fracture-transition temperature. A drop in total energy absorption also occurs over the fracture transition range, but in many types of test specimens the energy absorbed at the temperature at which predominantly granular fracture is obtained is still appreciable. In this event, as the temperature is lowered still further, the energy absorbed continues to decrease. For some specimens and for some steels, a point is reached at which the energy absorbed drops suddenly to an extremely low value. This latter drop in energy, when it occurs, may be called the transition to brittle behavior, or, for simplicity, the ductility transition.

A performance curve showing both a fracture transition and a ductility transition might look like the solid curve sketched in Figure 49-24. The drop in energy at T_1 is associated with the change in fracture appearance which occurs at this temperature. The drop at T_2 represents the transition to brittle behavior. The dotted curve

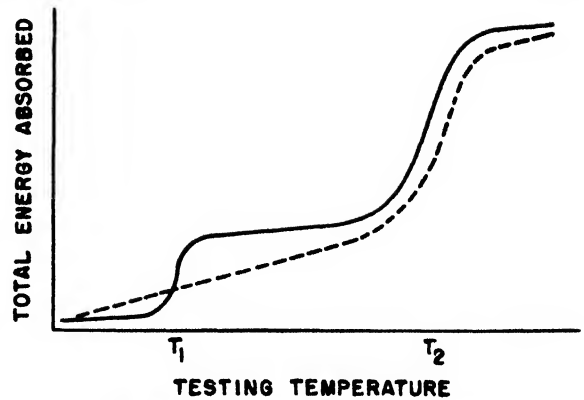


FIG. 49-24. Types of transition curves.

of Figure 49-24 represents an instance in which no sharp ductility transition is obtained as for V-notched Charpy specimens. In this case, the evaluation of performance would have to be based upon some arbitrary low level of energy.

The cleavage mode of fracture is characterized by the separation of two portions of a crystal normal to some definite plane in the crystal. The separation of layers of mica or the cleavage of ordinary rock salt are examples of this type of fracture. A cleavage fracture in a ferritic steel exhibits a shiny, crystalline appearance because the fracture occurs essentially along a single plane in each ferrite grain, thus exposing a series of bright crystalline facets. The shear type of fracture, on the other hand, occurs by a sliding action; i.e., two portions of the grain separate by sliding over one another. The shear fracture is generally associated with greater amounts of plastic deformation than the cleavage fracture, especially during the propagation of the fracture. This results in a dull, fibrous appearance in the shear fracture. In general, therefore, the fibrous fracture can be interpreted as denoting a high energy absorption during crack propagation, whereas, the granular or crystalline fracture is associated with a very low energy absorption during crack propagation. It is for this reason that the latter type of fracture can be propagated at extremely high velocities and over great distances in a structure before the applied loads can be relieved.

The fracture transition, then, is regarded as being related to the ability of the material to resist rapid crack propagation. The ductility transition, on the other hand,

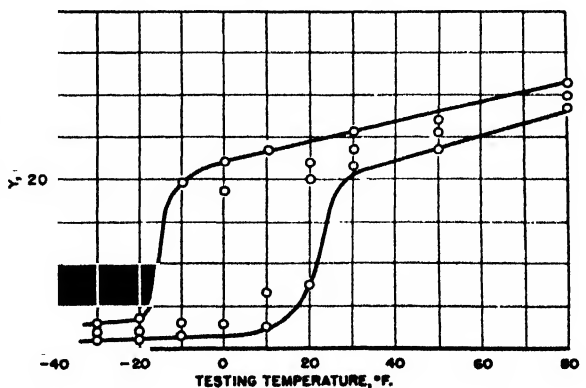


FIG. 49-25. Typical transition curve for Charpy keyhole-notch impact tests on a mild steel. (After R. W. Vanderbeck and M. Gensamer; "The Welding Journal Research Supplement," January, 1930.)

is related to the ability of the material to resist the initiation of a brittle fracture.

Since the transition from tough to brittle behavior in the impact test occurs over a range of temperature and the test results characteristically show considerable scatter in this range, some arbitrary method of defining the transition temperature or the transition-temperature range is necessary. A typical set of Charpy keyhole-notch impact data is shown in Figure 49-25. A procedure which is sometimes used to determine the ductility-transition temperature involves drawing an envelope about the results, as shown in Figure 49-25, and selecting a temperature near the middle of the envelope. Another procedure sometimes followed involves plotting the average energy absorption at each testing temperature and selecting the temperature at which the average curve crosses a selected energy level. It should be noted that the width of the scatter band will depend on the number of tests and the intervals between testing temperatures, but that the position of the middle of the scatter band will be less sensitive to these effects. Care should, of course, be taken that sufficient tests are made in the transition range that the middle of the range can be established within some predetermined number of degrees.

The results of Charpy V-notch impact tests are somewhat different in character, in that the transition from high to low energy absorption occurs over a wider temperature interval than observed for the keyhole specimen, as shown in Figure 49-26. The ductility-transition temperature is usually selected as the temperature at which a curve drawn through the average energy-absorption values reaches some selected low-energy level. The energy level selected is based on service experience or correlation with other tests. For rimmed and semikilled medium-carbon steels, a level of 10 to 15 ft. lbs. appears to give the best correlation with service performance. Other levels may be required, however, for other steels.

The fracture-transition temperature is usually based on some predetermined percentage of cleavage fracture. In some tests, where the fracture and ductility transi-

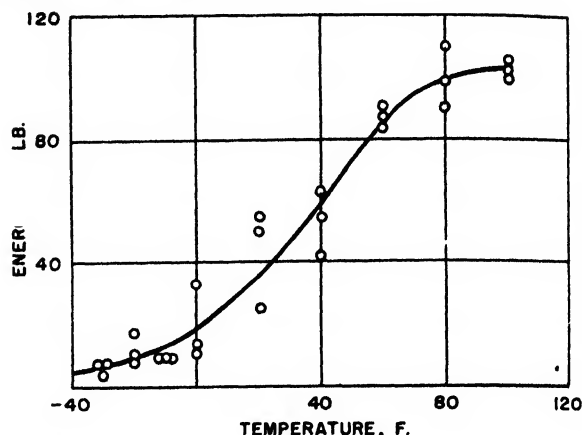


FIG. 49-26. Typical transition curve for V-notch Charpy impact tests on carbon steel.

tions are clearly separated, the fracture transition is also delineated by the change in energy absorption associated with the change in mode of fracture.

In conclusion, it can be said that the importance of the notched-bar impact test lies in its ability to discriminate between steels of varying susceptibility to brittle fracture, providing the results are interpreted on the basis of the temperature range of transition from tough to brittle behavior. The test is an extremely useful tool in the development of new and improved products with greater notch toughness, because of the possibility of detecting differences in materials which are not manifested in other tests, such as the tension test. Of particular value is the ability of the test to reflect differences in notch toughness caused by differences in steelmaking practices, as, for example, differences in deoxidation practice. Furthermore, on the basis of empirical correlations of Charpy V-notch impact-test results with service performance, it is now possible to evaluate with considerable reliability the suitability of steels for service in which a brittle-fracture hazard exists.

SECTION 5

FATIGUE TESTING

In a great many types of service applications, steel parts are required to withstand repeated or cyclic stressing; moving parts of machinery such as shafts, connecting rods, gears, etc., are examples of such applications. It has long been recognized that failures can occur in a machine part under repeated stress application at stresses well below those which the part is capable of withstanding under static load application. The failures which occur under repeated or cyclic stressing are referred to as **fatigue failures**. The importance of fatigue failures is well attested by the large percentage of failures in machine elements which are attributable to fatigue. It has been estimated that over 80 per cent of the failures in machines are a direct result of fatigue action.

Fatigue failures are progressive in nature, in that a crack is formed at some local spot or nucleus on the surface of the part after a certain number of load reversals, and is gradually propagated across the part. Finally, the remaining section becomes so small that it can no longer carry the applied load and complete failure ensues. A fatigue fracture generally appears brittle, even in metal which would be considered quite ductile in an ordinary tension test; the bright facets of the fracture led to the

erroneous concept that the metal had "crystallized." The extremely localized nature of the fatigue failure is one of its most distinguishing characteristics and one which must be constantly kept in mind in considering the danger of fatigue.

Fatigue failures can almost always be traced to a nucleus which is situated at some surface irregularity, such as a notch, a scratch, a flaw, or an abrupt change in section. It is evident, therefore, that the specific details of design and loading are of paramount importance in establishing the fatigue life of a particular part. The environment of the part is another aspect of the service conditions which is of great importance in establishing fatigue life in service. If the environment is at all corrosive, the fatigue resistance can be greatly impaired. It is obvious that the fatigue problem provides an excellent example of the importance of evaluating the actual service conditions to which a material is to be subjected, as discussed in the introduction to this chapter. Considerable data have now been accumulated which indicate that, if the actual loading conditions in the critical area of a part are accurately determined, the fatigue life of a particular material in that part can be

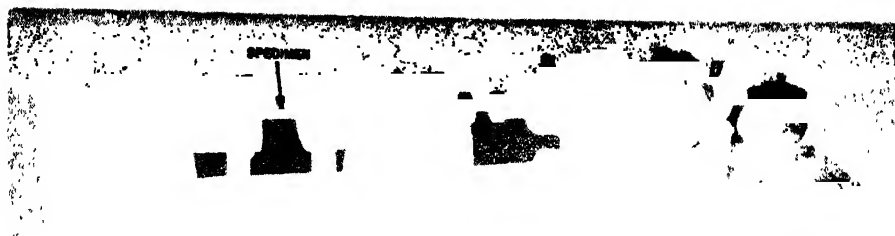


FIG. 49—27. R. R. Moore fatigue-testing machine. (Courtesy, The Baldwin-Lima-Hamilton Corp.)

predicted from simple laboratory tests. This conclusion has been reached on the basis of a comparison of results of both full-scale fatigue tests on actual parts, and service performance, with results on simple laboratory specimens.

The types of fatigue tests which are to be discussed here are those small-scale laboratory tests which are employed primarily to study materials, as opposed to tests of actual parts. Applicability of the results to service performance will be subject to the considerations of the relationship between test and service conditions mentioned above. It will be assumed that the tests to be discussed are at least capable of rating the relative fatigue properties of materials, and that, by properly taking into account particular service conditions, they may in many cases be directly applicable for design

purposes. Since so many of the applications to which steels are subjected involve repetitive loads, fatigue testing is considered of great importance since it is the only reliable method for evaluating the suitability of steels for this type of service.

Types of Fatigue Tests—The most commonly used type of fatigue test is the rotating-beam test, in which the specimen is subjected to a bending moment while being rotated. Any given fibre of the specimen is thus subjected alternately to compression and tension stresses of equal magnitude. One widely used machine of this type is the R. R. Moore machine shown in Figure 49—27. A schematic diagram of the loading arrangement is shown in Figure 49—28A, from which it is evident that a uniform bending moment is applied over the length of the specimen. Another type of rotating bending test utilizes a cantilever loading arrangement shown schematically in Figure 49—28B.

Another important type of fatigue test is the repeated bending or direct flexure test, in which the specimen is bent back and forth but not rotated. This type of test is particularly useful in the testing of specimens of flat rolled products. The direct flexure test has the further advantage that surface preparation of the specimen is not necessary, thus permitting the test to be made on specimens having the actual surface to be exposed in service. The mechanical type of machine shown in Figure 49—29 introduces the load into the specimen, which is fixed at one end, through an adjustable crank.

Direct flexure tests may also be of the resonant frequency type, in which the specimen is vibrated at its fundamental frequency by some oscillating applied force. Because of the characteristics of resonant vibrations, very small forces applied at or near the resonance frequency of the specimen are capable of producing large amplitudes of vibration and correspondingly high stresses. Some resonant frequency machines make use of an oscillating magnetic field tunable to the resonant frequency of the specimen. The specimen is supported at the nodes and vibrates as a free-free beam. By taking advantage of resonance vibration in this manner, specimens of relatively large cross-section can be tested

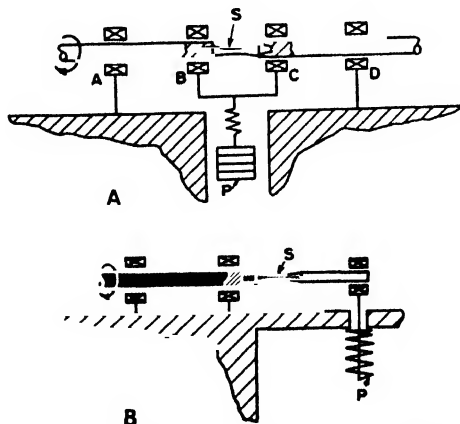


FIG. 49—28. Schematic loading arrangements for: (A) R. R. Moore and (B) cantilever-type of fatigue-testing machines. "S" indicates specimen and "P" indicates load, in both cases. (From "Manual on Fatigue Testing," published by American Society for Testing Materials, 1949.)

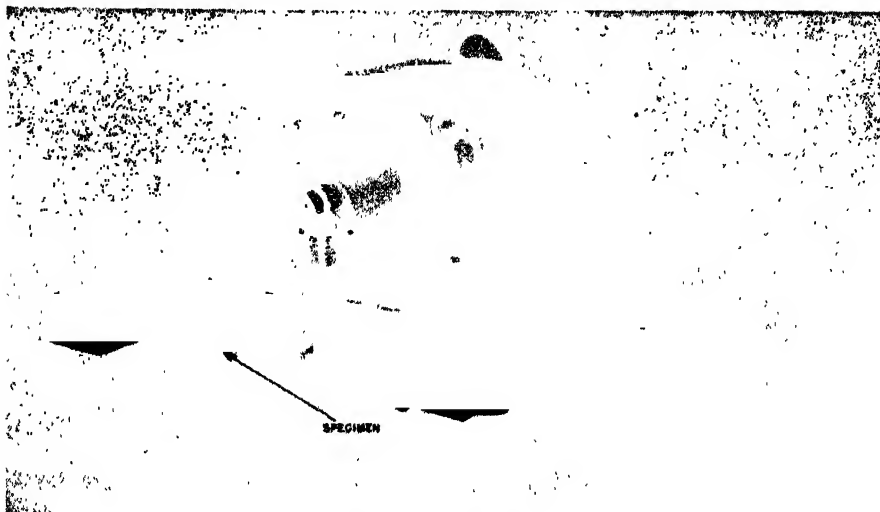


FIG. 49—29. Krouse direct flexure fatigue-testing machine. (Courtesy, Krouse Testing Machine Company.)

which would require very large machines if direct mechanical loading were employed. Other resonant frequency machines make use of mechanical rather than magnetic oscillators.

For certain types of fatigue, it may be desirable to conduct tests under direct, or axial loading conditions, i.e., the load is applied directly along the axis of the specimen. One of the major considerations in the design of axial-load fatigue machines is the provision of a loading arrangement which will insure the application of a truly axial load. Eccentricity of loading introduces

Table 49—V. Effect of Surface Condition on Endurance Limit

Moore & Kommers, rotating beam	
0.49% C quenched and drawn, 197 Brinell steel	
Finish	Endurance Limit, (Lb. per Sq. In.)
High polish (long.)	51,000
00 Emery	48,000
Ground	45,000
Smooth turned	43,000
Rough turned	42,000

Thomas, rotating cantilever

0.33% C steel

Finish	Endurance Limit, (Lb. per Sq. In.)
High polish (long.)	41,500
FF Emery	40,500
No. 1 Emery	40,000
Coarse emery	39,000
Smooth file	38,500
Turned	36,500
Bastard file	35,500
Coarse file	33,000–34,000

bending moments in the specimen and may have a very pronounced effect on the observed fatigue properties. Most axial-load machines are designed so that tests can be conducted in which the mean stress is not zero, i.e., in which the stress is not completely reversed. In many service applications, a part may not be subjected to alternate compressive and tensile stresses of equal magnitude, as is the case in the rotating beam type of test. For example, the stress may vary from zero to a maximum of 10,000 lb. per sq. in. tensile stress, in which case the fatigue properties are quite different than if the loading produced a range of stresses from 10,000 lb. per sq. in. in compression to 10,000 lb. per sq. in. in tension. It should be pointed out that the direct flexure test also affords some possibility of variation in mean stress. In some types of direct-flexure machines, it is possible to bend the specimen back and forth about an average position different from its equilibrium position, thus arriving at an average stress other than zero.

As already indicated, the environment to which a part is subjected can exert a profound influence on the observed fatigue behavior. If corrosive conditions are involved in a cyclic loading application, it is necessary to conduct the fatigue test under the same corrosive conditions. Usually, the standard types of machines and specimens are used in corrosion-fatigue testing, it merely being necessary to provide a jacket about the specimen suitable for carrying the corrosive medium.

In the planning of corrosion fatigue tests, it is important to consider the relationship of testing frequency and rate of corrosion and to maintain a similar relationship to that existing in service.

Fatigue-Testing Specimen Preparation—The most critical aspect of fatigue testing is the preparation of the test specimen. As already pointed out, fatigue behavior is very sensitive to surface conditions, and unless extreme care is taken in surface preparation, large scatter and unreliable results can be expected. Table 49—V gives some indication of the effect of surface finish on observed endurance limit for a steel. In the preparation of cylindrical specimens for use in rotating bending tests, the specimen is first rough machined, great care being taken to obtain a concentric test section in order that the stresses can be calculated accurately. Following the rough machining operation, a finish machining cut is taken. This finishing operation should be made with a

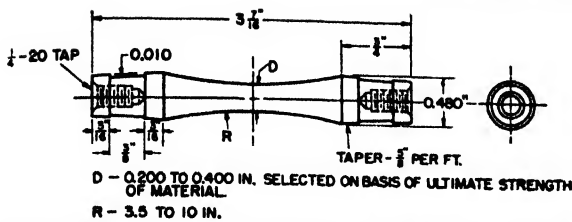


FIG. 49-30. Sketch illustrating rotating-beam fatigue specimen (R. R. Moore type). (From "Manual on Fatigue Testing," published by American Society for Testing Materials, 1949.)

sharp tool and a light feed in order to prevent bending, overheating, or excessive cold working of the specimen. For steels harder than about Rockwell C-40, grinding should be used for the finishing operation. The finishing cut should allow about 0.003 inch to 0.005 inch on the diameter for polishing, which is the next step in the specimen preparation. In order to carry out this operation successfully, it is important to recognize that this polishing is actually a cutting and not a buffing operation. Buffing may cold work the surface layers of the specimen sufficiently to affect the test results. A wide variety of polishing procedures are employed in different laboratories, but, in general, fatigue specimens are polished by a series of alternate circumferential and longitudinal polishing operations in which successively finer abrasives are employed. In each stage, the scratches from the previous operation are completely removed, the final polishing being carried out in the longitudinal direction of the specimen, since longitudinal scratches are less harmful than those in the circumferential direction. The specimen is generally rotated slowly in a lathe during polishing, about five stages being used in the polishing operation. One criterion of satisfactory polish,

recommended by some laboratories, is that the surface produced should be satisfactory for metallographic examination at 100 magnifications. One of the types of specimens commonly used in rotating-beam tests are shown in Figure 49-30. In fatigue tests on notched specimens, great precautions must be taken in preparation of the root of the notch. The root of the notch should be very carefully lapped with a rotating, abrasive-bearing wire of the same radius as the notch root. Lapping should be continued until all circumferential scratches left by the machining operation have been removed.

In testing specimens of flat-rolled products, it is generally desired to include the actual surface of the manufactured product in the test, especially if this is the surface to be exposed in service. If such is the case, no extensive surface preparation is needed. It is desirable, however, to remove burrs from the edges of the specimen and to produce a very slight chamfer or radius of about 0.005 inch on the corners. Such a procedure will usually eliminate spurious edge effects which could affect the final results.

Presentation of Fatigue-Test Data—Data on the fatigue strength of metals are generally presented in the form of so-called "S-N" diagrams, in which fatigue life is plotted as a function of the applied stress. Such a diagram is shown in Figure 49-31. Test results are shown as open circles. It is convenient to use a logarithmic scale in plotting the number of cycles, since the duration of tests at low stress levels may extend to hundreds of millions of cycles. Most steels exhibit what is known as a fatigue limit or an endurance limit, i.e., a limiting stress below which an infinite number of stress cycles can be endured. The value of the endurance limit is the fatigue characteristic most frequently reported for steels. In some applications, such as aircraft parts, however, steels may be used at stresses above their

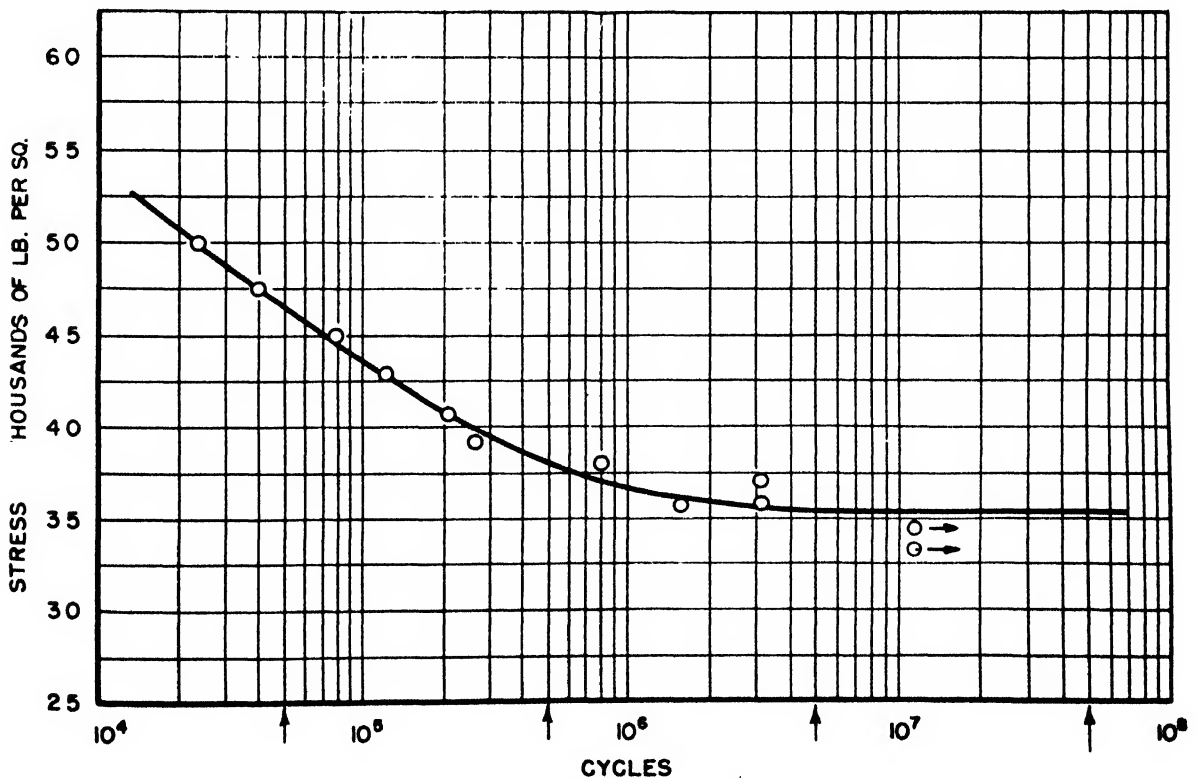


FIG. 49-31. Typical S-N diagram for a steel having a definite endurance limit.

endurance limit in order to permit savings in weight. Such parts are removed from service after some period of use based on some selected number of stress cycles. It is important to know, therefore, the highest value of stress which the material can withstand without failure for a given number of cycles. This value of stress is known as the **fatigue strength**, and is used not only for steels above the endurance limit but also in describing the fatigue characteristics of materials which do not show a true fatigue limit, such as austenitic stainless steels. Another fatigue characteristic sometimes reported is the fatigue life at some given stress level. For example, a part may be required to operate at a certain stress level. It then becomes necessary to select a material which can endure this stress for the greatest number of cycles.

In experimental determination of the endurance limit, it is customary to make the first test at a stress level well above the endurance limit, and to gradually lower the stress level in subsequent tests until the endurance limit is reached. If the approximate value of the endurance limit is not known beforehand, a tension test can be made, and the first fatigue test made at a stress level corresponding to about two-thirds of the tensile strength. It is obviously important to make the first tests

at high stress levels, since initial tests at stresses below the endurance limit are of very little value, because the proximity of a selected stress to the value of the endurance limit cannot be ascertained.

The S-N diagram for steels consists of a sloping portion and a horizontal portion corresponding to the endurance limit. For steels in a medium range of hardness, the knee of the curve will generally occur somewhere between one million and ten million cycles, so that an endurance limit value based on ten million cycles is usually satisfactory. Fifty million cycles may be somewhat more reliable, however, and judgment based on experience with the shapes of S-N diagrams for various materials should be used in selecting the maximum number of cycles for the series of tests to determine the endurance limit.

Significance of Small-Scale Fatigue Tests—There has been much discussion in recent years of the applicability of fatigue data derived from tests on small, highly polished bars to the prediction of fatigue life in service. One extreme point of view has been that such data are practically worthless and may even rate materials in the wrong order with respect to their service performance. This appears to be an unduly pessimistic attitude, especially in view of certain recent researches which in-

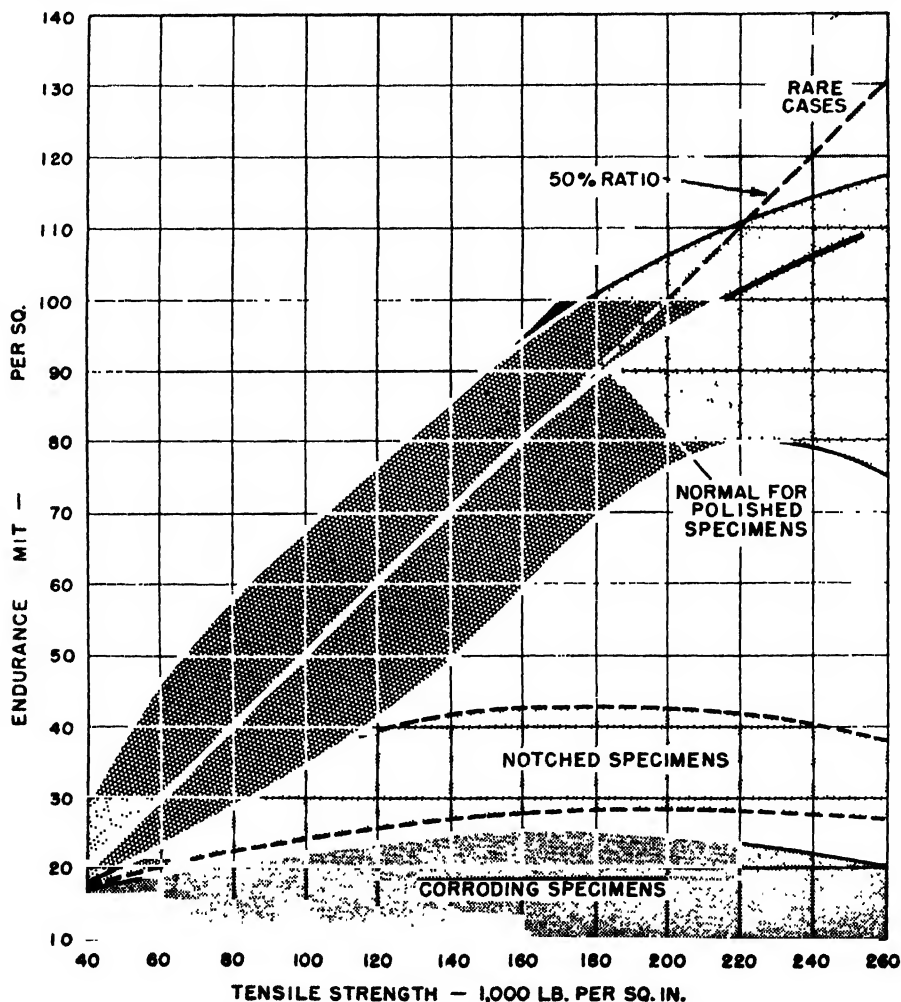


FIG. 49-32. Endurance limit as a function of tensile strength for polished, notched and corroding specimens. (From "Prevention of Fatigue of Metals," by the staff of Battelle Memorial Institute. Published by John Wiley & Sons, Inc., New York, 1941.)

dicating that if the service conditions are properly analyzed and if the state of stress at the critical point of a part is known, the endurance limit of the part can be related to that determined in a simple polished-bar test. Fatigue tests on full-sized automobile parts have provided the basis for this conclusion. It should not be inferred from these remarks, however, that results of polished-bar tests should be applied indiscriminately to predictions of service behavior. Frequently, the service conditions will be so complex as to preclude the possibility of accurate analysis. In this event, the only alternative is a simulated service test.

In general, the endurance limits of structural steels vary in a fairly regular way with tensile strength. An outstanding exception is provided by certain steels such as Bessemer steel or some of the high-strength low-alloy, phosphorus-bearing steels, in which the ratio of yield strength to tensile strength is higher than that normally found in most steels. In these steels, the endurance limit is relatively high for a given tensile strength, which may indicate that endurance limit is more closely related to the yield point than to tensile strength. The endurance ratio is defined as the ratio of the endurance limit to the static tensile strength and is in the neighborhood of 0.50 for most steels. Figure 49-32 is a schematic diagram for ordinary steels indicating the general relationship between endurance limit and tensile strength for polished specimens, severely notched specimens, and corroding specimens.

In considering the effect of notches on fatigue, it is customary to use as a measure of the notch effect the fatigue strength reduction factor, which is usually designated as K_t , and is defined as the ratio of the fatigue strength of a member or specimen with no stress con-

centration to the fatigue strength in the presence of stress concentration. It is obvious that the strength reduction factor has no meaning except in terms of a specific notch geometry. A concept of notch sensitivity originated by Peterson relates the strength reduction factor K_t to the theoretical stress concentration factor K_i resulting from the notch. K_i is the ratio of the maximum stress in a notched section to the nominal or average stress across the entire section computed from mathematical analyses or determined experimentally. Peterson defines notch sensitivity by a "q" factor which depends on the relative values of K_t and K_i in the following way:

$$q = \frac{K_t - 1}{K_i - 1}$$

Notch sensitivity according to this definition varies from zero (where $K_t = 1$) to unity (where $K_t = K_i$). This concept is useful in describing the reaction of a material to notches of varying degrees of severity. Since most actual parts subjected to cyclic loading actually contain stress concentrations of some type, it is important that the effect of notches on fatigue properties be considered.

In summary, it can be said that small-scale tests are very useful in comparing the behavior of different materials under repeated stress and can be used in studies of the effects of metallurgical variables on fatigue characteristics. Such tests not only form a useful guide in the development of new materials of improved fatigue strength, but also permit the evaluation of various surface treatments such as carburizing, nitriding, shot peening, cold rolling, etc. The effect of different environments on fatigue behavior also can be studied conveniently in laboratory tests.

SECTION 6

HIGH-TEMPERATURE TENSION, CREEP, RUPTURE AND HARDNESS TESTING

The design of load-bearing structures for service at atmospheric temperature is generally based on the yield strength—or for some applications, on the tensile strength—determined in the ordinary room-temperature tension test. In service at ordinary temperature with a design stress determined in such a way, the metal behaves essentially in an elastic manner, that is, the structure undergoes an elastic deformation immediately upon load application and no further deformation occurs with time; when the load is removed the structure returns to its original dimensions.

At elevated temperature the behavior is different. A structure designed according to the principles employed for atmospheric temperature service continues to deform with time after load application, even though the design data may have been based on tension tests at the temperature of interest. This deformation with time is called creep, since at the design stresses at which it was first recognized it occurred at a relatively slow rate. A somewhat similar, though not exactly analogous, phenomenon is the flow of tar under its own weight on a warm day. A detailed discussion of the subject of creep has been presented by Smith.

When creep was originally encountered—the subject first received experimental attention only several decades ago—it was logical to decrease the stress to a sufficiently low value that creep would not occur. Much of the early work was devoted to such an effort. It was discovered, however, that such a limiting creep stress was lower the greater the sensitivity of the measuring apparatus and that quite low stresses would have to be

employed to preclude the occurrence of measurable creep. In fact, the required stresses were so low and the section size correspondingly so large that the application of metals to high-temperature service was appreciably retarded. Accordingly, a new and more rational technique of design was developed and is now employed. This recognizes the existence of creep and is concerned not with avoiding its occurrence, but with limiting it to tolerable values within the contemplated service life of the structural member.

The rate of creep depends, for a given material, upon the stress, temperature and history of the material. Accordingly, the function of the creep-testing laboratory is to determine the dependence of creep upon such variables.

The possibility that creep may continue until fracture occurs raises another limitation in addition to that of the allowable deformation. In the use of metals at elevated temperature, not only must the design stress be chosen so that the deformation shall not exceed a limiting amount for the contemplated service life, but also it must be such that fracture will not occur. This latter characteristic is empirically related to the former. The variation of time for fracture or, as it is more commonly called, rupture, is determined in the so-called rupture test.

THE HIGH-TEMPERATURE TENSION TEST

The most common method for measuring the strength of metals is the tension test, which provides useful

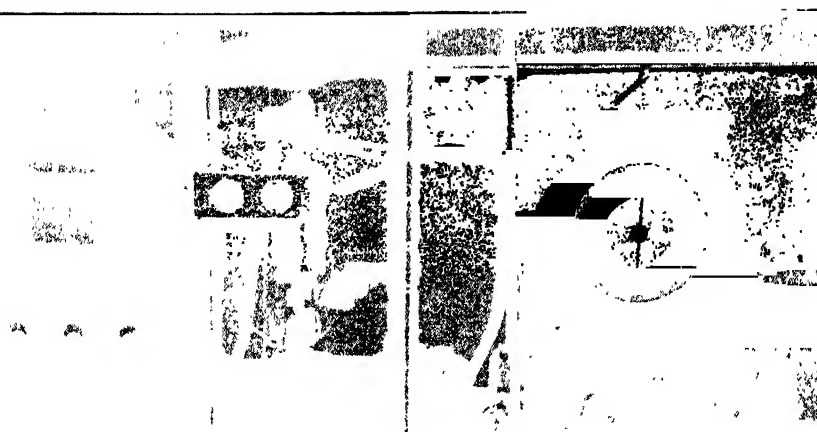


FIG. 49-33. Elevated-temperature tension-test equipment showing specimen ready for testing in hydraulic tension test machine. Temperature control equipment at left.

information up to the temperature at which creep is encountered. The special techniques involved in performing the tension test at elevated temperature are covered by ASTM Specification E 21-43. Elevated temperature tension tests are made in conventional tension testing machines. The specimen is heated by a furnace supported on one column of the machine, the furnace being free to swing into and out of axial position. Figure 49-33 shows a general view of the equipment, including temperature controls.

Tests may be made on either bar or thin flat stock. For bar tests a 0.505-inch diameter specimen with threaded ends is gripped within the furnace; for sheet tests a longer specimen having a reduced gage section at mid-length is gripped outside the furnace.

The extension is recorded autographically by an extensometer, designed for this specific purpose, used in conjunction with a standard autographic recorder of the Selsyn motor type. With such equipment, accurate determinations of high-temperature yield strength may be made up to about 1400° F. This is well above the maximum temperature at which short-time yield-strength data should be used for design purposes; for high-temperature applications more reliable design information is obtained from creep or creep-rupture tests. Measurement of tensile strength, omitting determination of yield strength, may be made up to 2400° F. While of no particular value for design purposes, information on tensile strength, elongation and reduction in area at these very high temperatures is often useful as a guide for hot-working operations or for studying structural changes within the material.

The results of a series of elevated temperature tension tests on carbon steel and a wide variety of alloy steels are summarized in Figure 49-34. The strength of metals usually decreases with increase of temperature, but it will be seen from Figure 49-34 that the strength of plain carbon steel actually increases as the temperature of test is raised from room temperature to about 500° F.

This increase of strength and the accompanying decrease of ductility in certain steels is called "blue brittleness," because in this temperature range, the oxide film which forms on the steel has a bluish tinge. With further in-

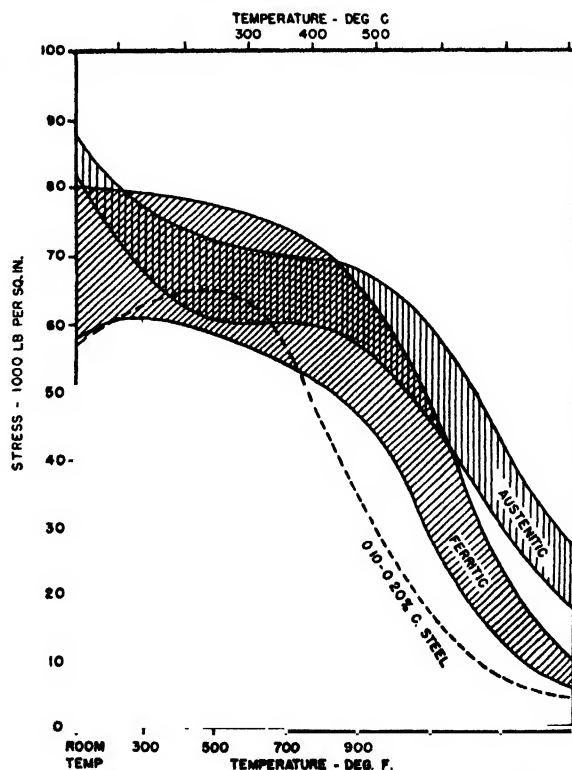


FIG. 49-34. Tensile strength of various steels as measured at temperatures between room temperature and 1500° F.

crease of temperature, the strength of the material decreases, until at about 700° F, the strength is approximately the same as that at room temperature, after which the strength continues to decrease with increasing temperature. The increase of strength at 400° to 600° F is believed to be due to a selective segregation of carbon and nitrogen atoms during the progress of the tension test, and is usually called "strain-aging." The segregation, which may result in a special type of precipitate, is so small in amount that it cannot be detected under the electron microscope. The chemical composition of the precipitate is unknown.

THE CREEP TEST

Since strain hardening by cold working is progressive in steel stressed at temperatures below 700° to 800° F, and since creep is small and occurs at a decreasing rate at these moderately elevated temperatures, the tension test suffices for determination of strength values for design purposes in a temperature range from room temperature up to about 700° F. Above this temperature, steel will flow continuously under the applied load and design cannot be based on the yield strength or proportional limit as measured by the tension test. The rate of flow or creep rate is related to the stress, and the purpose of the creep test is to determine the stress which will result in a given creep rate.

The method of measuring creep resistance is simple

enough in principle, but in practice it requires considerable laboratory apparatus, and great care and precision in its operation. Disregarding for the moment the exact type of apparatus, the following fundamental steps are almost universally employed. The specimen is held at a constant temperature in an electric furnace, and is subjected to a static tensile load. The load causes the specimen to elongate gradually, and the amount of elongation is measured periodically. The total elapsed time of each test may be a matter of hours, weeks or months. Some creep tests have been run for more than ten years. In this country, the customary testing time is from 1000 to 3000 hours. The general method for creep testing is covered by ASTM Specification E 22-41.

At the E. C. Bain Laboratory for Fundamental Research at the United States Steel Research Center, Monroeville, Pa., there are six stands for creep tests, twelve stands for rupture tests in air, and two stands for rupture tests under an inert atmosphere or vacuum. A general view of these facilities including some of the temperature-control equipment is shown in Figure 49-35. The specimen of steel to be subjected to creep testing is machined to the shape shown in Figure 49-36. Indicating arms are attached to the specimen to bring the amount of elongation taking place in the gage length to a common point for measurement with a micrometer microscope. The assembly is then screwed into cast alloy-steel pull rods, and the whole unit is slipped into a tubular electric resistance furnace. The specimen is



FIG. 49-35. General view of creep and rupture testing equipment in the E. C. Bain Laboratory for Fundamental Research at the United States Steel Research Center, Monroeville, Pa.



FIG. 49—36. Creep specimen (bottom) and assembly used at the E. C. Bain Laboratory for Fundamental Research at the United States Steel Research Center, Monroeville, Pa.

held in a steel framework (Figure 49—37). It is anchored at the bottom, and attached to a lever at the top. When the specimen is loaded by hanging weights on the lever, the extension taking place in the specimen is brought to a single point by the indicating arms (Figure 49—36), on each of which is welded a platinum reference surface carrying engraved reference marks in the form of crosses. One surface moves vertically relative to the other, and by measuring the relative vertical displacement of the reference marks with a microscope carrying a micrometer eyepiece, the amount of elongation which has taken place in the specimen may be measured to an accuracy of about ± 0.00002 inch in direct reading. The operator is shown making a test measurement in Figure 49—38.

The temperature of the specimen should be held

within $\pm 1^\circ \text{F}$ over the gage length and from day to day during the 3000-hour ($4\frac{1}{2}$ -month) test period, since a change of as much as 2°F may cause enough thermal expansion or contraction to introduce an error into the readings. Such accuracy is necessary, since the amount of flow which can be tolerated in commercial parts operating at elevated temperature is very small—from a fraction of 1 per cent up to 1 per cent in 10 or 15 years—and accurate measurements of creep rates of 0.1 per cent or 0.01 per cent per 1000 hours must be made within the test period.

When the change of length taking place in the specimen from day to day is plotted against the elapsed time, a creep curve is obtained, whose typical form is shown in Figure 49—39. When the load is first applied, an immediate elastic extension (A) occurs. Then the speci-

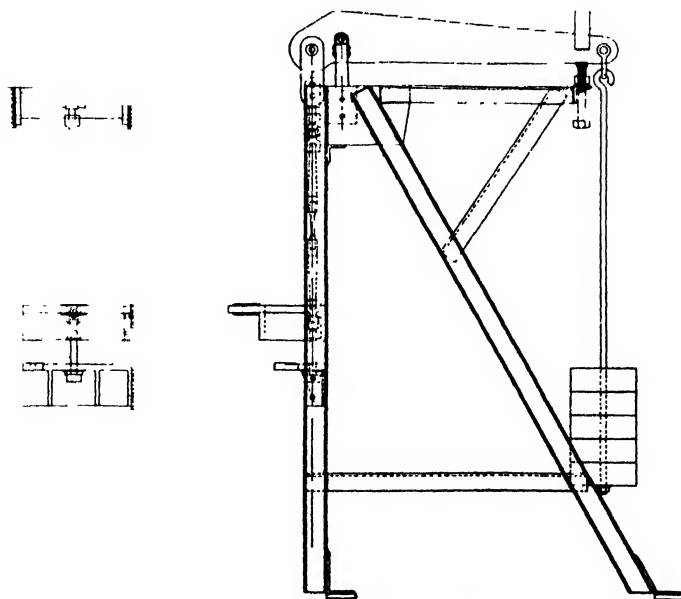
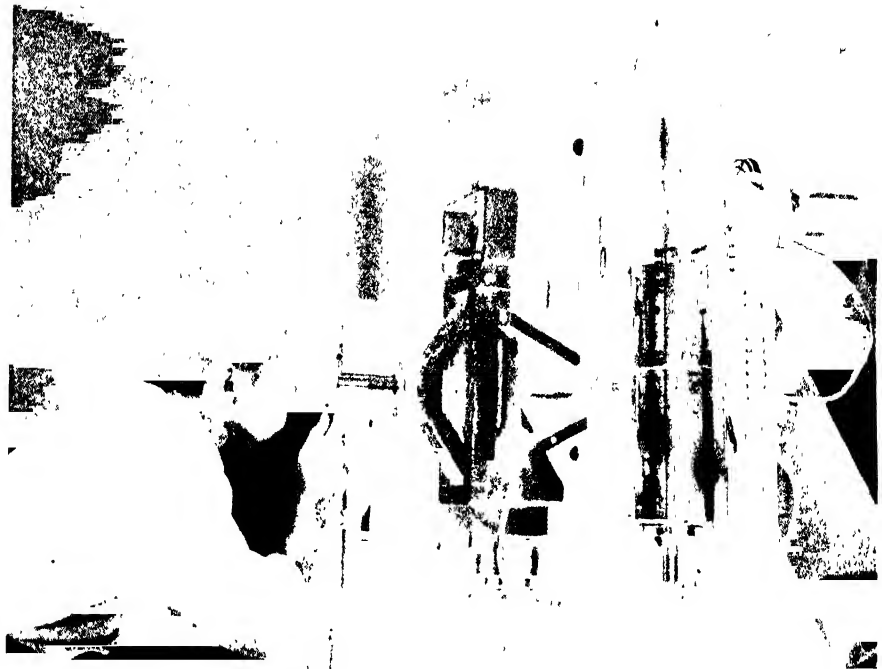


FIG. 49—37. Diagrammatic representation of creep and rupture test stand.

FIG. 49—38. Creep test microscope in position for making measurements through window in wall of furnace at the E. C. Bain Laboratory for Fundamental Research at the United States Steel Research Center, Monroeville, Pa.



men stretches gradually, at a decreasing rate, during the "first stage" of creep (B). The rate then becomes constant for a period of time during the "second stage" of creep (C). The slope of the creep curve in this second stage is the creep rate commonly used for design purposes. Finally, if the load or temperature is high enough, or the time long enough, the creep rate increases in the "third stage" (D), leading to fracture of the specimen. At the end of the testing period, if fracture has not occurred, the load is removed and elastic contraction (E) occurs, corresponding approximately to the elastic extension found upon application of the load at the start of the test. Thus, it is apparent that metals creeping under stress at high temperature can and do show both plastic and elastic properties simultaneously. The amount of permanent deformation is represented by (F).

The exact shape of an individual creep curve depends on the stress employed and upon the temperature, as well as upon the composition and structure of the metal. While elastic extension is always found at the beginning and elastic contraction is always found at the end of the creep test, it is rare in any one test to find periods of decreasing, constant, and increasing creep rate. An actual creep curve for a steel at 1000° F under a tensile load of 10,000 lb. per sq. in. is shown in Figure 49—40.

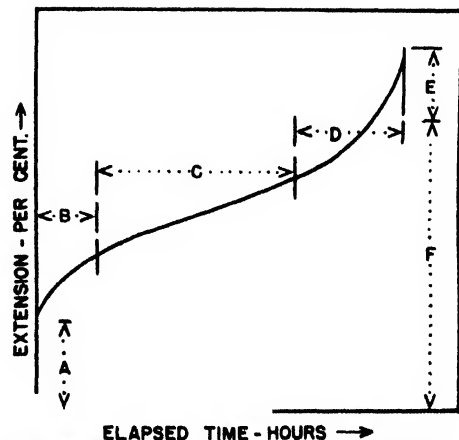


FIG. 49—39. Schematic creep curve. Extension plotted against elapsed time. (A) Elastic extension; (B) creep at decreasing rate; (C) creep at approximately constant rate; (D) creep at increasing rate; (E) elastic contraction; (F) permanent change of length.

THE MAKING, SHAPING AND TREATING OF STEEL

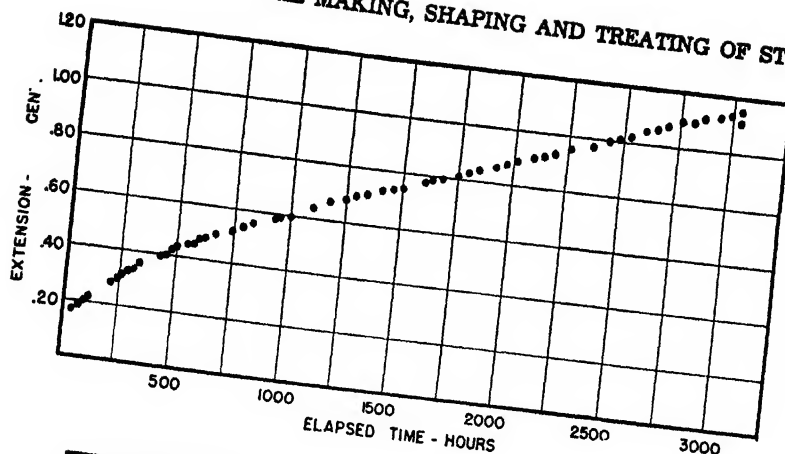


Fig. 49-40. Actual creep curve of alloy steel under stress of 10,000 lb. per sq. in. at 1000° F.

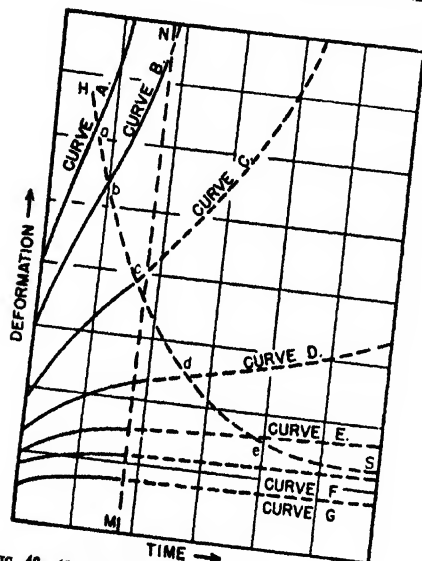


Fig. 49-41. Schematic plot of typical creep curves for seven specimens of the same steel; each specimen tested under a different stress ranging from a very high stress, curve A, to a very low stress, curve G, at a constant temperature. (Taken from "The Interpretation of Creep Tests" by P. G. McVetty, Proc. ASTM, Vol. 34, Part 2, 1934; pp. 105-116.)

Two standards of creep strength are commonly used in this country: (a) the stress producing a creep rate of 0.0001 per cent per hour, often expressed as 1 per cent per 10,000 hours (a little over a year); or (b) the stress per 100,000 hours (about 11 years). The second standard is used in designing moving parts such as steam turbines, in which the total creep must be very small, for example, a fraction of 1 per cent in 20 years.

The stress for a selected creep rate for the material in question must be ascertained by experiment. This is done in the following manner: several creep tests are run under different stresses at a single temperature and the creep curves plotted on the same chart. A family of curves is shown in Figure 49-41. Three or four tests on any given material are usually sufficient to indicate its behavior at a given temperature. The creep rate during the second stage of creep is measured and plotted against the applied stress. On log-log coordinates, the points are found to lie approximately on a straight line as shown in Figure 49-42. The stress for a creep rate of 0.0001 per cent per hour or 0.00001 per cent per hour may then be obtained by interpolation or extrapolation.

In Figure 49-41 the line MN represents the end of a testing period which is too short to show the true long-time behavior of the material, since the later increase of creep rate in curve D, for example, would not have been detected, and the useful strength of the material would have been overestimated. For accurate determination of long-time creep strength, standard testing periods of from 1000 to 3000 hours are now generally employed to overcome this difficulty. Short-time creep tests may tell us that some materials are relatively

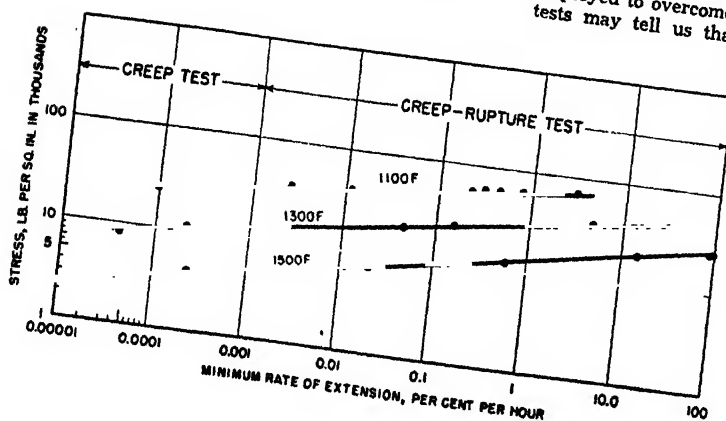
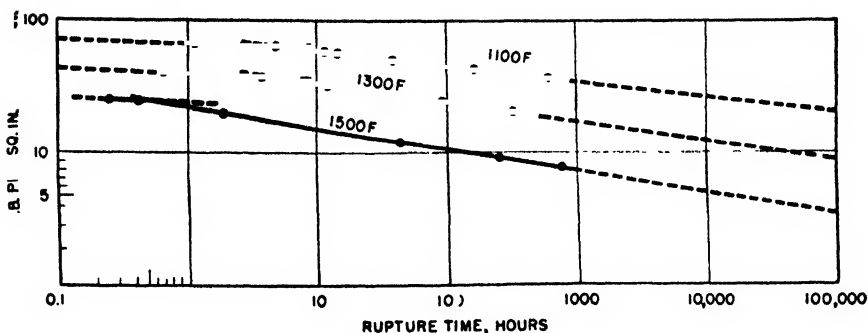


Fig. 49-42. Effect of stress on creep rate of 18-8 Mo (Type 316) steel at various temperatures.

FIG. 49—43. Effect of stress on time for rupture of 18-8 Mo (Type 316) steel at various temperatures.



stronger than others over the period of the test, but they tell little that is not revealed by a high-temperature tension test. Hatfield's time-yield and the standard German creep test, which deal with the amount of creep between the thirtieth and fortieth hours after loading, obviously cannot show the effect of a structural change which would not occur at the test temperature until several weeks after application of the load. Extrapolation from such short-time creep tests generally results in an over-estimate of the creep strength of the material under investigation.

THE RUPTURE TEST

The rupture test (sometimes called the stress-rupture or creep-rupture test) is identical with the creep test, except that the loads and consequently the creep rates are higher, and the test is carried to failure of the material. The apparatus for carrying out the rupture test is usually the same as that employed for the creep test, except that a different instrument is used for measurement of the elongation. In the creep test, the total strain is usually less than 0.5 per cent, while in the rupture test the total strain may amount to as much as 50 per cent or more. The rate during the second stage of creep may be measured as in the customary creep test and when the rate and corresponding stress are plotted on a log-log diagram, extrapolation to a lower creep rate, such as the standard 1 per cent per 10,000 hours, will give a fair prediction of the creep behavior of the material (Figure 49-42). This appears to be about the most reliable method of predicting long-time creep behavior from short-time tests, and has proved to be quite useful in preliminary surveys of the creep behavior of new materials being considered for elevated-temperature service.

In reporting rupture data, it is customary to plot the applied stress against the time for failure on log-log coordinates as shown in Figure 49-43. It has been found that after a certain time, the type of fracture changes from predominantly transgranular to predominantly intergranular, that is, from rupture primarily through the grains to separation primarily along the grain boundaries. Although this change is frequently associated with the increase in the downward slope of the log stress-log rupture time curve, convincing proof of such relationship is still lacking. Because of the change in slope, however, it is desirable to carry the rupture tests out to a considerable time period. Rupture values are usually reported as the stress for fracture in 100, 1000, 10,000, or 100,000 hours.

THE HIGH-TEMPERATURE HARDNESS TEST

The high-temperature hardness test is one of various elevated-temperature mechanical tests which has not

been widely used, but which promises to be valuable as a tool in determining the high-temperature behavior of metals. Two test methods have been developed for determining the hardness at elevated temperatures, one

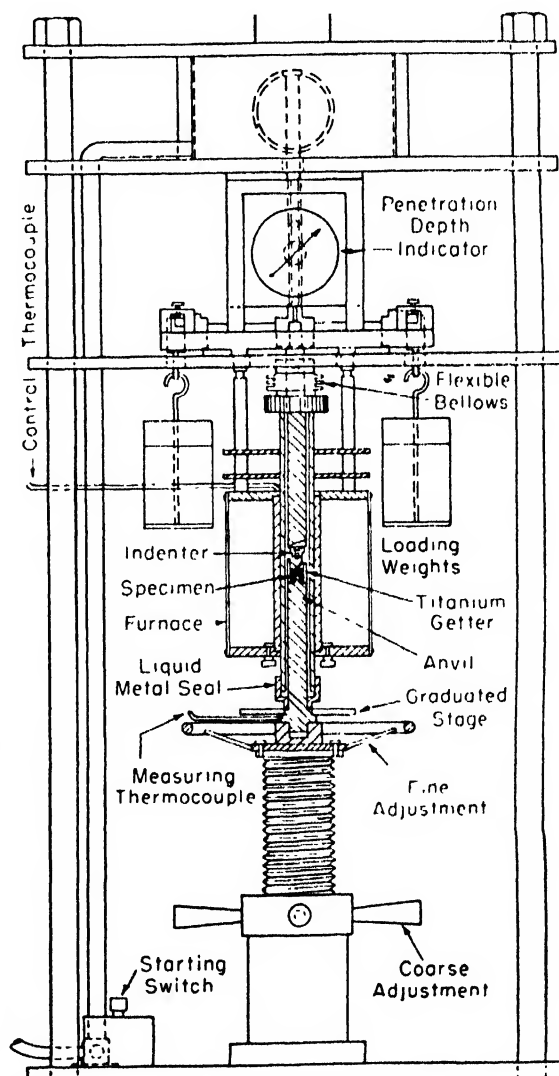


FIG. 49-44. Schematic sketch of hardness tester used at the Edgar C. Bain Laboratory for Fundamental Research at the Research Center of the United States Steel Corporation.

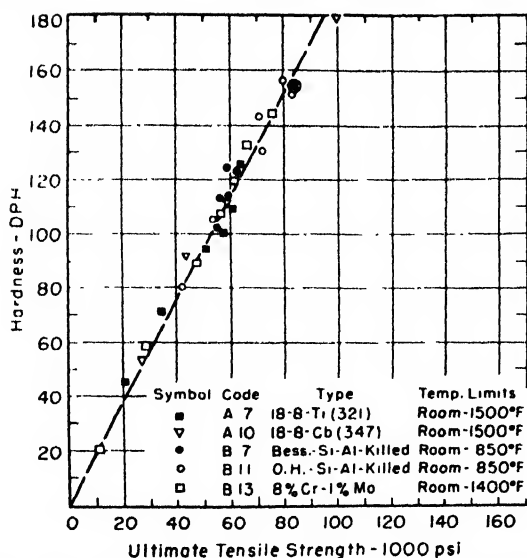


Fig. 49-45. Relation between hot hardness and ultimate tensile strength for several ferritic and austenitic steels.

employing static loading and the second employing dynamic loading. A certain degree of standardization of test equipment employing static loading has been in evidence recently; in addition, refinements in testing techniques have made it possible to determine high-temperature hardness with the same accuracy and degree of reproducibility as in room-temperature hardness testing. Little attempt has been made to standardize dynamic loading apparatus or techniques. Little correlation is found between static and dynamic results. This lack of correlation is related primarily to the strong dependency of mechanical behavior at elevated temperatures on rate of deformation.

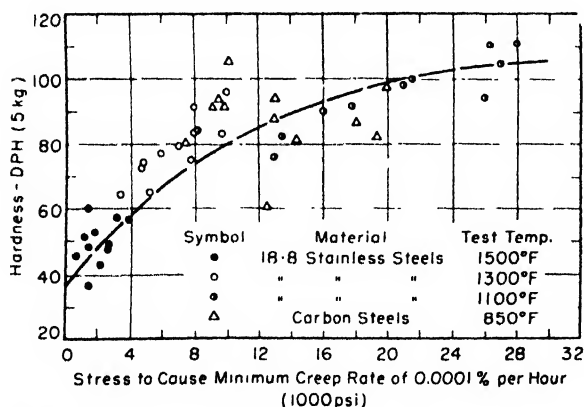


Fig. 49-46. Relation between hot hardness and creep strength of 18 Cr-8 Ni stainless and carbon steels.

The design of hot-hardness apparatus employing static loading has been based essentially on standard room-temperature equipment, including Vickers and Rockwell testers. Inert atmosphere, or a vacuum, have been employed in the test chamber to minimize oxidation. In all instances, small samples are used and a considerable number of hardness impressions are made on each sample at constant temperature or at different temperatures up to at least 1500° F. In hot-hardness apparatus based on the Vickers tester, the diagonals of the impressions are measured after cooling the specimen to room temperature.

A schematic sketch of the equipment used at the Edgar C. Bain Laboratory for Fundamental Research at the Research Center of the United States Steel Corporation is shown in Figure 49-44. In this apparatus the specimen, whose test surface is given a metallographic polish, and a Vickers-type indenter are heated to the desired temperature in a virtually inert atmosphere. The measuring thermocouple is spot-welded on the surface of the specimen. A dead-weight load ranging between 3 and 10 kg. is automatically applied to the indenter while in contact with the specimen. Following a loading cycle of one minute, the load is removed automatically and the sample rotated about its axis, which is offset from the indenter axis, in preparation for making a new impression at the same or at a different temperature. This apparatus has been employed in studying solid-state reactions such as strain-aging, phase changes during tempering and recovery and recrystallization phenomena. Other studies have shown that hardness results on steels at elevated temperatures are closely related to the tensile strength as shown in Figure 49-45, and approximately related to the creep strength as shown in Figure 49-46 and the creep-rupture strength as shown in Figure 49-47.

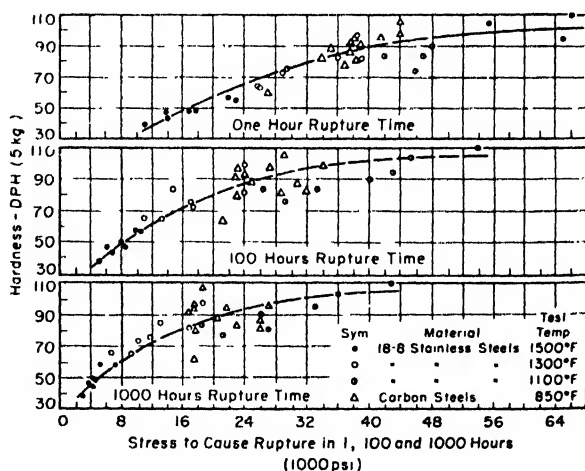


Fig. 49-47. Relation between hot hardness and creep-rupture strength for 18 Cr-8 Ni stainless and carbon steels.

SECTION 7

USE OF HIGH-TEMPERATURE DATA IN DESIGN

In studying the flow of metals under stress at elevated temperature, it should be noted that, to a much more marked degree than at room temperature, the lower the strain rate, the lower the apparent strength of material. The relation between the strength of carbon

steel as determined from the short-time tension test, and the strength of the same material as determined from creep and rupture tests is shown in Figure 49-48. At the lowest strain rates, permanent deformation occurs at high temperature under stresses less than the

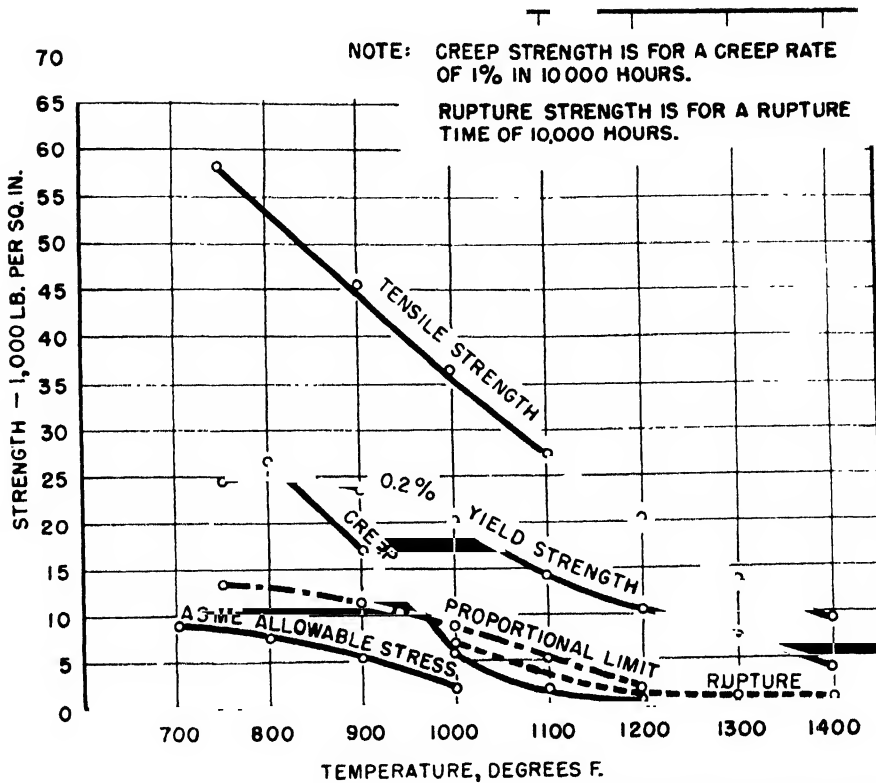


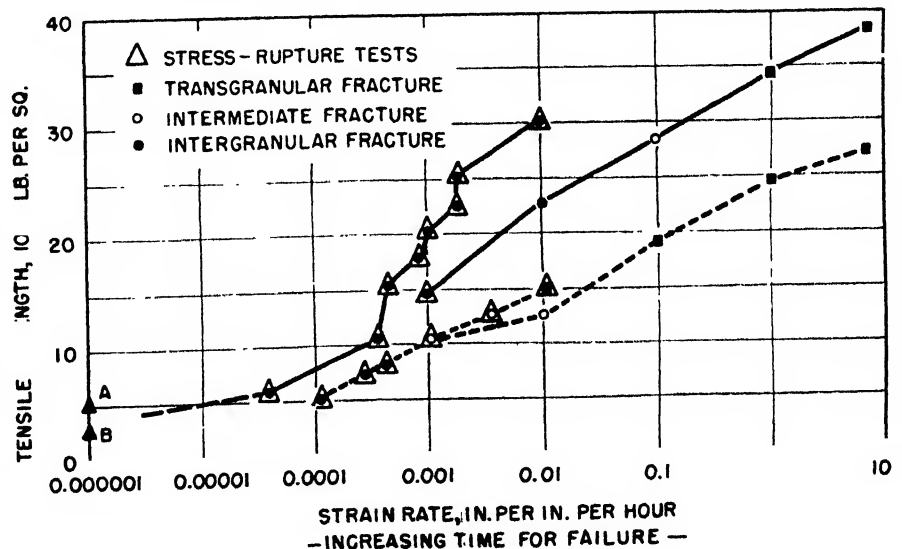
FIG. 49—48. The effect of temperature on the strength of killed, 0.10 to 0.20 per cent carbon steel as indicated by tension, creep, and rupture tests.

yield strength or proportional limit as determined from short-time tension tests. In fact, continuous flow occurs under even the smallest applied stresses and design must be based on a dynamic rather than a static concept of the strength of materials.

Another important effect of the decrease of strain rate is the change in the type of fracture from transgranular to intergranular as the rate is decreased. The change in type of fracture is accompanied by a decrease in the amount of elongation. This behavior is character-

istic of all metals and alloys, and final failure under creep conditions tends to be brittle instead of ductile. The exact amount of elongation accompanying creep failure is usually unknown, since failure under creep conditions does not usually occur for many years. Extrapolation indicates that, in molybdenum steel, the amount of elongation at 1100° F decreases from about 38 per cent in a short-time tension test carried out at a strain rate of about 7.5 inches per inch per hour to less than 10 per cent under creep conditions.

FIG. 49—49. The effect of strain rate on the apparent strength of pearlitic (solid line) and spheroidized (dashed line) molybdenum steel at 1100° F. A and B indicate reported stresses for creep rate of 1 per cent per 10,000 hours in molybdenum steel at 1100° F.



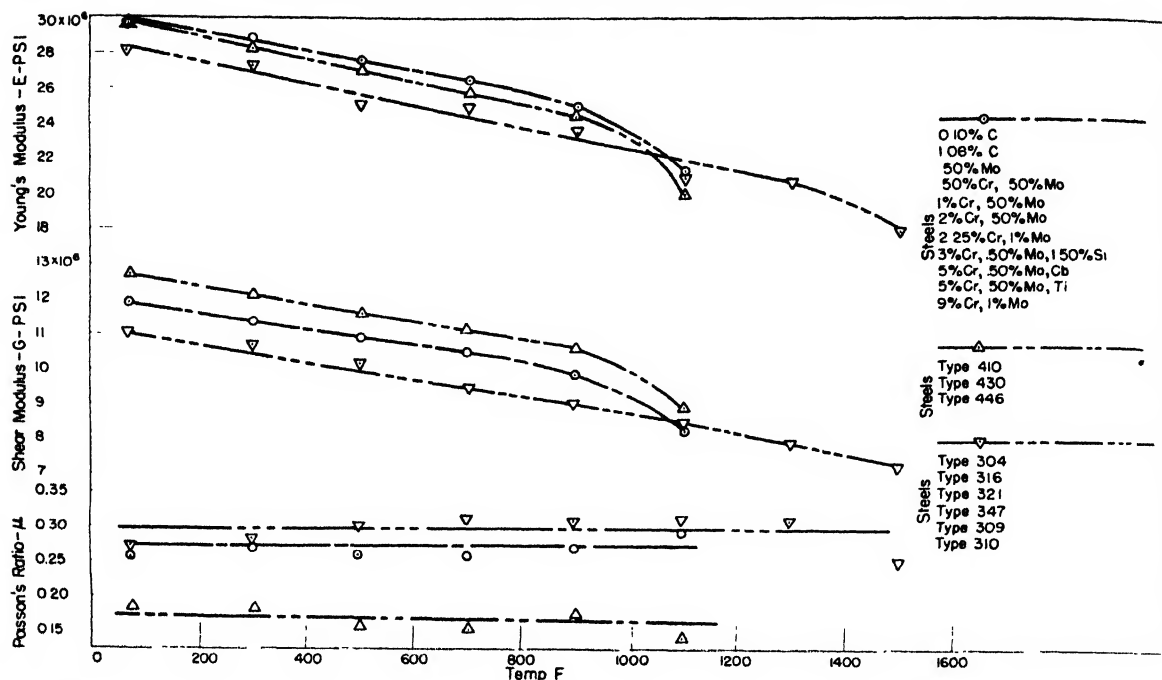


FIG. 49—50. Variation of elastic moduli with temperature for various ferritic and austenitic stainless steels.

A clearer picture of the effect of strain rate on the strength of material is obtained when strain rates are plotted against the applied stress, as shown in Figure 49—49. In this study, a series of controlled strain-rate tests were carried out on spheroidized and pearlitic molybdenum steels at 1100° F. The most rapid test, at a rate of about 7.5 inches per inch per hour, is typical of the strain rates employed in the ordinary short-time elevated-temperature tension test. Tests were also carried out at 1, 0.1, 0.01, and 0.001 inches per inch per hour. Slower strain-rate data from stress-rupture and creep tests of these materials are also represented and show the same trend of decrease of strength. In fact, at a strain rate of 0.000001 inches per inch per hour, about 1 ten-millionth of the strain rate employed in the short-time tension test, the apparent strength of molybdenum steel at 1100° F, has decreased from 37,000 to 2,500 lb. per sq. in. This clearly illustrates the necessity of determining the strength of materials in the creep range from creep and rupture tests rather than from short-time tension tests. High-temperature design stresses, as set by the American Society of Mechanical Engineers, are established on the basis of long-time creep and rupture strength, and the limiting temperature is determined by the scaling resistance of the material. Below 700° to 800° F, the stresses are obtained from short-time tension data. Service experience and the behavior of similar materials are also taken into consideration. Allowable working stresses for the standard ferrous alloys may be found in Sections I and VIII of the ASME Boiler Construction Code and in the API-ASME Code for Unfired Pressure Vessels.

Under certain conditions of design at elevated temperature the permissible strain is of the order of magnitude of the elastic strain. It is necessary in such a case to compute working stresses on the basis of elastic properties of the material. In general, the design of structural members on this basis requires values of three constants of elasticity, the elastic modulus in tension and compression (Young's modulus), the shear

modulus and Poisson's ratio. Of equal importance is the fact that these elastic constants are also needed in computing the magnitude of thermal stresses.

The elastic moduli can be measured under conditions of static or dynamic loading. Under static loading, Young's modulus can be determined from a simple tension or a bend test and the shear modulus can be determined from a torsion test. Poisson's ratio (μ) is then computed from the following relationship:

$$\mu = \frac{E}{2G} - 1$$

where E is Young's modulus and G is the shear modulus. By subjecting a small cantilever beam to simultaneous bending and twisting by application of a single load it is possible to determine both E and G from a single test. Such a method has been employed to determine the variation of the elastic moduli under static loading for a number of commercial, ferritic and austenitic stainless steels up to 1500° F. Average curves for the results obtained are shown in Figure 49—50.

Under dynamic loading, high-frequency cyclic tests of relatively low stress amplitude are generally used. Both E and G can be determined from the measured velocity of longitudinal and transverse waves produced by high-frequency pulses. Dynamic tests on steels have been made up to 1600° F.

Theoretically it is predicted that the variation of the elastic constants, E and G , is linear with temperatures up to nearly the melting point of metals; however, deviations from linearity are caused by various phase and crystallographic lattice changes, magnetic changes and grain-boundary gliding in polycrystalline metals. Because of the low stresses encountered, the dynamic results are not affected by grain-boundary gliding. At temperatures where gliding at the grain boundaries is nil, below 700° F for plain carbon, 900° F for alloyed ferritic, and 1300° F for austenitic stainless steels, the static and dynamic results agree very well.

SECTION 8

MISCELLANEOUS MECHANICAL TESTS

Metals are subjected to a great variety of applications which may involve mechanical properties not directly measured in the more common mechanical tests already described. For this reason, a great many specialized tests have been developed which are usually aimed at a closer approximation to some important aspect of the actual service conditions than the ordinary tests provide. A few of the more important miscellaneous mechanical tests are briefly described in this section, in order to provide some indication of the types of tests which have been developed.

A. Compression Testing—Frequently, in the design of structural members which are to be subjected to compressive working stresses, it is desirable to design on the basis of compressive yield strength rather than tensile yield strength, particularly if there is reason to believe that the compression properties of the material under consideration differ from the tension properties. The data obtainable from a compression test may include the proportional limit, the elastic limit, the yield strength or yield point, and in some cases "compressive strength." The term compressive strength has been defined by the American Society for Testing Materials as the maximum compressive stress which a material is capable of developing. This strength figure has a definite value only for a material which fractures in compression. For other materials, arbitrary compression strength values are sometimes reported which are based on some degree of distortion which is regarded as indicating complete failure of the material.

An ASTM tentative specification for the compression testing of metallic material in other than sheet form has been drawn up and is designated as E9-49T. It is recommended that standard specimens be in the form of circular cylinders, the important feature in specimen preparation being parallelism of the ends and perpendicularity of the planes of the ends to the specimen axis. As in the case of tension testing, axial loading is of great importance. In some instances, a special subpress is used in conjunction with the regular testing machine in order to facilitate truly axial application of the compression load.

Compression members are frequently fabricated from sheet material, particularly for use in aircraft. In the design of such members, it is necessary to use the compression properties of the sheet material. Obviously, edgewise compression tests are not simple on thin sheets because of buckling difficulties. Several methods of testing sheet specimens in compression have been proposed. One of these is the "pack" method, in which a composite specimen is built up of several layers of sheets. In this way, a specimen of sufficient thickness to avoid buckling is provided. Another type of test provides support against buckling by special jigs. One such jig consists of a number of rollers which rest against the faces of the specimen. Another type of fixture which was developed at the National Bureau of Standards simply uses flat tool-steel blocks lubricated with a high-pressure lubricant to support the specimen (Figure 49-51). The specimen is allowed to overhang the supports slightly for loading and attachment of extensometers. Still another type of sheet metal compression test is the cylinder method, in which the flat specimen is formed, by bending rolls, into a cylinder about one and one-half inches in diameter and soldered along the longitudinal joint. This cylinder is very resistant to

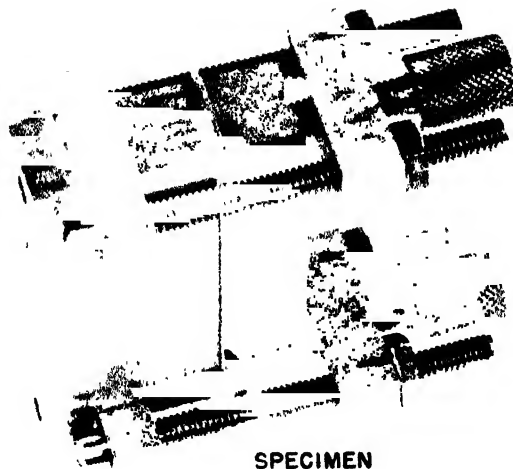


FIG. 49-51. National Bureau of Standards jig for compression tests on sheet metals. (Courtesy, U. S. Department of Commerce, National Bureau of Standards.)

buckling and the ends can be accurately machined to insure axial loading. Another advantage of this method is the accessibility of the specimen for strain measurement. The principal disadvantage of the cylinder method is that a small amount of cold work is unavoidably introduced in forming the cylinder.

B. Bend Testing—The bend test, as the name implies, is intended to evaluate the ability of a material to undergo bending during forming operations to which it may be subjected. Generally, the bend test is conducted as a "go-no go" test; i.e., either the specimen meets the desired bend requirement or fails by cracking. In some instances, however, a ductility value is derived from the bend test by placing gage marks on the outside or tension side of the bend and measuring the elongation after completing the bend. This procedure is covered by ASTM Designation: E16-39 and was developed primarily for testing of welds.

Ordinarily the bend test is much simpler, merely involving a determination of whether or not a specified bend can be made satisfactorily. A typical method of stating a bend-test specification for a plate material, for example, is as follows: "The bend-test specimens shall stand being bent cold through 180 degrees without cracking on the outside of the bent portion to an inside diameter which shall have the following relation to thickness or gage of material." A set of bend diameters is then specified for various thickness ranges, the bend diameters increasing with increasing plate thickness. Edge conditions are very important and generally sufficient edge preparation is permitted to avoid an initial edge fracture. The method of bending is not specified and a large number of bending devices have been designed, the design usually being aimed at convenience so that large numbers of tests can be run in a relatively short time.

C. Cupping Tests—A number of so-called "cupping"

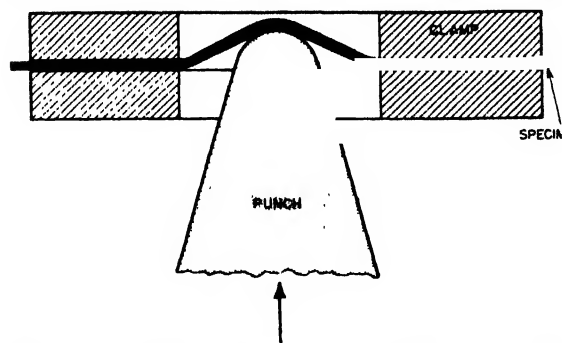


FIG. 49—52. Schematic representation of cupping test for sheet metals.

tests have been developed for the purpose of measuring the ductility of sheet metal under conditions where the sheet is stretched in all directions simultaneously. Cupping tests are made on different machines, or testers, known as the Erichsen, the Olsen, the Guillery, the Wazau, etc., of which the first two are the most commonly used. While they differ in many respects, the Erichsen and the Olsen testers are similar in the manner of applying the test. In both, the specimen of sheet or strip is clamped between two rings or dies, and a smooth ball, mounted upon or attached to a plunger, is forced against the flat surface of the specimen enclosed within the area of the ring, as shown in Figure 49—52, thus stretching it into the form of a cup, and continuing until the material is fractured. In the Olsen type of machine, the depth of the cup causing fracture is measured in thousandths of an inch by a recording or measuring device, the indicator of which is actuated directly from the surface of the sample. The end point of the test is indicated by a pressure gage, the pointer of which drops back upon fracture of the specimen. In the Erichsen tester, the plunger is somewhat cone-shaped with a

smooth spherical end; the fracture of the specimen is detected visually by a mirror attachment, and the depth of the cup is measured in millimeters.

In a great many actual sheet-metal forming operations, the blank is required to stretch in all directions. It would be expected, therefore, that a cupping test would prove a better criterion of the behavior to be expected in such forming operations than would a simple tension test in which the material is stretched in only one direction. Actually, the correlation between cupping tests and actual performance has in general been disappointing, except in cases where large differences in formability exist. Cupping tests are widely used for inspection purposes, however, since they provide a quick indication of ductility and some indication of the surface condition to be expected after drawing by the degree of roughness or coarseness developed on the cup during the cupping test.

D. Strain-Sensitivity and Strain-Aging Sensitivity Tests—Steel products are very frequently subjected to cold-forming operations prior to or during fabrication for their final use and may go into service in the cold-worked condition. As discussed in the chapter on plain carbon steels, the properties of cold-worked steel may change progressively with time, this change being known as strain-aging. The question arises, therefore, as to how the changes in properties brought about by cold working and strain-aging will affect the performance of the material in service.

It is well known that straining and strain-aging exert a profound influence on the notch toughness characteristics of certain steels, tending to increase the susceptibility of these steels to brittle fracture. One of the most informative methods of evaluating the effect of straining and strain-aging on notch toughness is the determination of the shift in transition temperature in the notched-bar impact test. The interpretation of the relationship between test results and service behavior is, of course, subject to the same limitations as empha-

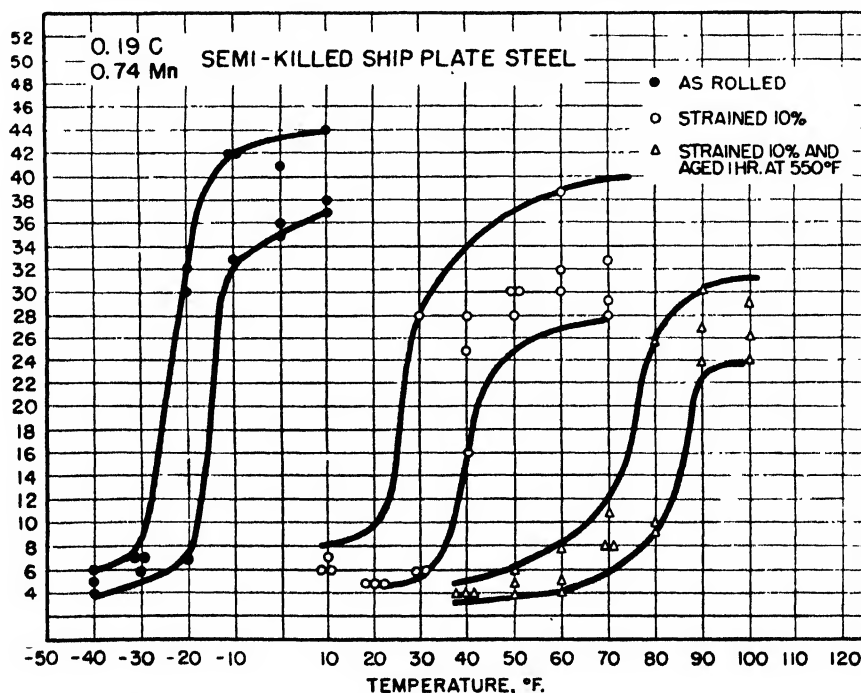


FIG. 49—53. Effect of strain aging on transition temperature in the Charpy impact test.

sized earlier in the general discussion of notched-bar impact tests. It is possible, however, to obtain extremely useful comparisons among different steels and to provide relative measures of the extent to which the notch toughness is impaired by straining and strain-aging.

One testing procedure which has proved convenient and useful involves the cold rolling of oversize blanks for Charpy-type specimens. The degree of oversize is based on the desired amount of cold working; for example, if ten per cent reduction is desired, the blank is made about 0.0394 inch oversize and reduced to the standard dimension of 0.394 inch. Two sets of specimens sufficient for the determination of transition temperatures are prepared in this manner. One set is tested as soon as possible after rolling and notching, while the other set is artificially aged for one hour at 550° F. This treatment is believed to produce the maximum effect of strain aging on notch toughness. The shifts of transition temperature caused first by straining, and second by straining and aging give an indication of the extent to which the ability to resist brittle fracture has been impaired. Examples of the types of transition behavior which may be obtained are shown in Figure 49—53.

Another test which is sometimes used to indicate the effects of straining and strain-aging on notch toughness is the Graham tapered-bar test. Varying amounts of cold work are produced by drawing a tapered circular bar through a die so as to produce a uniform cross section. The bar is then notched at various points along its length which correspond to various amounts of cold work and is broken as a cantilever specimen at each notch. The maximum amount of cold work is usually ten per cent. The cold-drawn bar can also be aged before testing. Although this test is relatively fast and simple, it has the disadvantage of not permitting the determination of transition temperatures, and is thus subject to the uncertainties which arise in impact testing at a single temperature.

Steels which exhibit pronounced strain aging show an increase in tensile strength when tested at temperatures in the neighborhood of 400° F over that obtained in room-temperature tests. This increase in tensile strength is sometimes used as a measure of the effects of strain aging, but should not be substituted for a notch toughness test unless a correlation has been established.

Another aspect of strain aging, which is of great practical importance, is the return of the yield point in temper-rolled sheets intended for deep-drawing operations. Such sheets are normally temper rolled after annealing in order to eliminate the yield-point elongation and the accompanying tendency for the formation of stretcher strains or fluting during forming. In steels which are susceptible to strain aging, however, the yield-point elongation and stretcher-strain tendency will return with time. Accelerated aging, in which the temper-rolled material is held at an elevated temperature such as 400° F for some predetermined length of time, is frequently applied in order to indicate the aging tendency. The extent to which the yield point elongation returns in a tension test provides some indication of the manner in which the material can be expected to behave in a drawing operation after a considerable lapse of time at atmospheric temperature.

E. Torsion Testing—In the torsion test, a specimen is subjected to twisting or torsional loads analogous to those encountered in drive shafts, crank shafts, etc. Information on the strength in torsion, particularly the yield point or yield strength, is important in designing for such applications. Torsion tests are not extensively used, since a satisfactory estimate of the yield point in

torsion can generally be made from the yield point in tension. In some instances, however, a direct measurement of torsion properties may be desirable and occasionally may be specified.

Torsion data are usually obtained in the form of a torque-twist curve, in which the applied torque is plotted against the angle of twist. Torsion produces a state of stress known as pure shear, and the shear stress at yielding can be calculated from the torque at yielding and the specimen dimensions. Actually, the stress varies from a maximum at the surface of the specimen to zero at the axis. In the elastic range, the variation is linear, and the maximum stress for a cylindrical specimen can be readily calculated from the following relation:

$$S = \frac{16 T}{\pi d^3}$$

where: S = maximum shear stress (lb. per sq. in.)
 T = torque in inch-pounds
 d = diameter of specimen.

In the plastic range, the calculation of the maximum shear stress is more complicated and the reader is referred to the method developed by A. Nadai, which considers the twisting of a cylindrical bar in the plastic range.

In the elastic range, the shear strain is proportional to the shear stress, the constant of proportionality being known as the **modulus of rigidity**. The modulus of rigidity is about 10,000,000 for steel.

Impact tests utilizing dynamic torsional loads are used to a considerable extent in testing brittle materials such as tool steels. Since the ductility in torsion is greater than in tension, a greater energy absorption is obtained in the torsion impact test than in a beam type of impact test. Sensitivity is thereby improved making separations possible in the torsion impact which are not possible in the notched-bar impact test. The energy absorbed by the specimen in the torsion impact test is measured by the loss in rotational energy of a flywheel which engages one end of the specimen and breaks it.

F. Shear Testing—The term "shear testing" as used here refers to determinations of the resistance of metals to shearing in dies, i.e., cutting by shearing. "Shear test" is sometimes also used to refer to the torsion test, which, as indicated above, measures the resistance to deformation under shear stresses. The sense in which the terms "shear" or "shear strength" are used should always be clearly indicated in order to avoid any misinterpretation.

If a load-penetration diagram is determined while shearing a metal in dies, it will be found to be similar in general shape to the load-extension diagram in tension testing. The maximum load observed during the shearing operation divided by the area being sheared is taken as the shearing strength or shearing resistance of a material. It is necessary to state the exact testing conditions in reporting shearing resistance, since the value obtained will depend markedly on the die arrangement. The clearances, shear angles, and sharpness of the cutting edges will all affect the observed value of shearing resistance. The degree of penetration of the punch into the metal when the fracture begins is also usually reported. It has been generally observed that the shearing resistance of medium carbon steels is from two-thirds to three-fourths of the ultimate tensile strength.

G. Wear Testing—H. W. Gillette has defined the wear of a metal part as "its undesired gradual change in dimensions in service under frictional pressure." Wear generally involves two stages, in the first of which de-

formation occurs, and in the second of which removal of material may occur. Wear of metals may involve the contact of metal on metal, as in shafts and bearings, brakes and wheels, valves and seats; or it may involve the contact of non-metals on metals, as in the case of coke chutes, steam shovel buckets, etc. The phenomenon of wear is so complex that it is extremely difficult to interpret, and is one kind of service for which suitability can be reliably evaluated only in terms of actual service tests. Various wear tests have been used for specific purposes, but are only valid if the test method produces wear in the same manner in which it is produced in service. A more detailed discussion of wear and wear testing can be found in the ASTM "Symposium on Wear of Metals," published in 1937, and in "The Abrasion Resistance of Metals," by R. D. Haworth, published in the ASM Transactions, 1949.

H. Damping Capacity Tests—Damping capacity is a measure of the rate at which a material dissipates energy of vibration, or in other words, a measure of the ability to damp out vibrations. Damping depends upon internal friction in the metal, which is manifested at stresses well below those at which gross yielding occurs. Internal friction probably arises from minute amounts of plastic flow on a submicroscopic scale, a process which results in heating and a loss of vibrational energy. The ability to damp vibrations is of importance in certain structures subjected to vibrations, where there may be a danger of resonant vibrations arising. Resonance can lead to

large amplitudes of vibrations and excessively high stresses. A choice of a material with relatively high damping capacity, which can also satisfy the ordinary mechanical requirements, may be of some benefit in avoiding resonance conditions. High damping capacity materials are also of value in supports for moving machinery, in that the transmission of vibrations to the supporting structure may be reduced.

A commonly used method of measuring damping capacity involves the measurement of the rate of decrease of amplitude of torsional vibrations. One end of a cylindrical specimen is clamped in a rigid base with the specimen in a vertical position. On the other end of the specimen, a heavy inertia bar is clamped. This inertia bar is rotated through an angle corresponding to the desired stress level in the specimen, usually by means of magnets, and then released. The specimen is thus set in torsional vibration, and the rate of decrease of vibration amplitude is measured by some suitable method. In a recently developed machine, a light beam is focused on a mirror on the inertia bar, and the beam is reflected onto a rotating drum carrying a strip of sensitive photographic paper. Measurements of the rate of decrease of vibration amplitude from the photographic record permit a calculation of the damping capacity. Damping capacity is usually expressed in terms of **specific damping capacity**, which is defined as the ratio of the energy loss per cycle to the elastic potential energy at the maximum amplitude of the cycle.

SECTION 9

NONDESTRUCTIVE TESTS

Certain nondestructive tests, although not providing a direct measurement of mechanical properties, are extremely valuable in locating and isolating material defects which could greatly impair the mechanical performance if permitted to remain in a machine element or structural member placed in service. Since the part or article inspected by nondestructive methods is in no way altered or affected, it is possible to inspect the entire article. Furthermore, in many cases, it is feasible to carry out 100 per cent inspection of all product intended for applications which demand especially high quality. In this way, uncertainties as to whether representative sampling has been attained are eliminated. It should also be pointed out that an extremely important function of nondestructive tests is in the examination of parts which have been in service, such examination frequently being possible without removing the part from service. Incipient failures can oftentimes be detected in this manner, thus permitting removal of the part from service before serious damage is done. Since nondestructive testing provides such an important supplement to the conventional mechanical test procedures in evaluating suitability for mechanical service, it is appropriate that a few of the more important nondestructive testing methods be briefly described in this chapter on mechanical testing.

Radiography—Radiography is one of the oldest and most widely used methods of nondestructive testing. Its use is based upon the great penetrating power of X-rays or gamma rays, which, depending on their initial intensity, are capable of penetrating several inches of steel. X-ray tubes operating at voltages as high as three million volts are now in use capable of producing an X-ray beam which can penetrate six to seven inches of steel. Gamma rays, which are given off spontaneously by certain radioactive materials such as radium and by syn-

thetically produced isotopes such as Co^{60} , Ir^{192} and Cs^{137} , are capable of penetrating greater thicknesses than X-rays, and have the important advantage that the equipment required is entirely portable and requires no maintenance. The sensitivity of gamma rays to defects in thinner sections, however, is considerably inferior to that of X-rays, and X-rays are, therefore, more commonly used for radiographic work.

As a beam of X-rays or gamma rays passes through an object, it is absorbed to a degree depending upon the thickness of the object and upon the specific absorbing capacity of the material. Metals are characteristically strong absorbers of X-rays. If a cavity in the object being radiographed lies in the path of the beam, the effective thickness which must be traversed by the beam is decreased and the intensity of the emitted beam is greater in the vicinity of a point opposite the cavity. In this way, an image of the cavity is formed, which can be considered as an X-ray shadow picture. Any large cavity or inclusion which differs in absorbing capacity from the bulk of the test object will act in such a manner as to produce variations in intensity from point to point of the beam as it passes through the object. These variations in intensity or X-ray shadows, so to speak, are generally recorded on photographic film, but may also be observed visually by a fluorescent screen which is activated by the X-ray beam.

Considerable experience is necessary for the expert interpretation of radiographs, particularly in identifying the source of an indication and in judging how harmful the defect may be. A background of practical experience is usually built up by sectioning objects on which certain radiographic indications have been obtained, in order to positively correlate the indication with its source. The principal fields of application of radiography are in testing castings and welds. In castings, defects

such as pipe, shrinkage, hot tears, blowholes, and sand or slag inclusions may be detected by radiographic inspection. In welds, detection of hot tears, shrinkage cracks, blowholes, slag inclusions, lack of fusion, and lack of penetration is possible. One of the principal shortcomings of radiography is its inability to detect very thin defects, unless the plane of such defects lies parallel to the beam. A crack in which the surfaces are very close together is an example of such a defect. Since this type of defect can be very harmful in service, some other method of detection such as magnetic or sonic methods must be adopted.

Magnetic Methods—The magnetic methods of nondestructive testing to be described here are not aimed primarily at an evaluation of actual magnetic properties of a given steel specimen, as carried out for electrical or magnetic steels, but are employed for the measurement of some other characteristic of interest which is reflected by variations in magnetic behavior. Some of the principles of magnetism involved in such methods are discussed in Chapter 2, under "Fundamental Principles of Physics." The magnetic tests of primary interest here are those which are used in detecting physical discontinuities in steel. These methods depend upon establishing a uniform magnetic field around the steel part or test object, either by a magnetizing coil or by passing an electrical current through the part. Any physical discontinuity in the metal creates a magnetic flux leakage which can be detected by the use of magnetic powders or by a pick-up coil.

Magnaflux and Magnaglo are two widely used magnetic powders which are particularly useful in detecting fine surface and subsurface cracks. The Magnaglo powder differs only in that the magnetic particles are coated with a fluorescent material which glows under ultraviolet light. In this way, the contrast of the magnetic powder indication can sometimes be improved. In use, the powders are applied wet or dry to a magnetized part. If cracks are present, the leakage flux causes a local concentration of magnetic particles in the vicinity of the discontinuity.

The well known Sperry rail tester is an example of a magnetic flaw detector utilizing a pick-up coil. In using this device, a heavy direct current is passed through a rail section from a testing car as the car moves along. A uniform magnetic field is thus set up around the rail. If a transverse fissure is present in the rail the magnetic field is disturbed. A small coil moving along just above the track surface detects this disturbance by variations in induced current, which are recorded and used to indicate the location of the defect.

Ultrasonic Methods—Sonic methods have long been used in the detection of flaws or defects in steel. The clearness of the ring given off when an object is struck is well known to give an indication of soundness or freedom from gross discontinuities. The recent development of the ultrasonic testing techniques represents the culmination of much research to improve the sensitivity of sonic methods.

The ultrasonic method of flaw detection depends on the fact that when a beam of ultrasonic waves is projected into the metal object, a small physical discontinuity in the path of the beam acts as an obstacle to the beam. The reflection method, which is the most useful of the various ultrasonic techniques for flaw detection, makes use of the fact that a portion of the incident beam may be reflected from a discontinuity lying in the path

of the beam. By timing the interval between the generation of the initial pulse and the reception of the reflected pulse, the location of the defect can be determined. The ultrasonic waves are generated by a quartz crystal, making use of the piezoelectric effect, which causes the crystal to expand or contract when subjected to an electric charge. By using a rapidly oscillating electric field, the crystal is made to vibrate at the same frequency as the oscillation of the field. This effect is reversible, permitting the same crystal to be employed as a detector, in which case an oscillating electric charge is set up in the crystal as it is subjected to mechanical vibration. Measurement of this oscillating charge provides a method of detecting ultrasonic waves.

The reflection method of ultrasonic inspection has been successful in detecting a large number of types of defects such as pipe, shrinkage, cracks, flakes, fatigue cracks, concentrations of inclusions, segregations, and laminations. The method is rapid and is useful in production line work. As in the case of any nondestructive test, a background of experience must be built up in the interpretation of the indications with respect to the nature of the defect producing a particular type of indication.

Chapter 50

GAGE NUMBERS

In the metal industries, the word gage is used in various systems, or scales, for expressing the thickness or weight per unit area of thin plates, sheets, and strips, or the diameters of rods and wires. Specific diameters, thicknesses, or weights per square foot are denoted in gage systems by certain numerals prefixed to the word gage; for example, No. 12 gage, No. 20 gage, No. 30 gage, or simply 12 gage, 20 gage, and 30 gage. Gage numbers are used only in connection with thin materials; that is, usually when the thickness is not more than one-quarter inch or the weight per square foot is not more than 10 pounds, although most gage tables actually begin at about one-half inch, or 20 pounds per square foot, and one table begins at double these quantities. Heavier and thicker materials are always indicated by weight per unit area or length, or by thickness in English or metric units.

It is advisable, at this point to emphasize the danger of confusion in the use of gage numbers to indicate thicknesses and diameters. This danger is present in domestic as well as in foreign trade, and can be avoided by specifying thickness or diameter in inches, centimeters, or millimeters, or in weights per square foot or per square meter, or by giving other equivalents, in absolute units, of the gage desired. The relations and equivalents of the principal gages are shown in the several tables in this chapter.

Origin of Gages—This custom of indicating thickness and diameter by gage numbers originated in the early days of the metal industries, and the gage numbers were probably first employed to designate the different sizes that could be most readily produced by different stages, or steps, in the processes of manufacture. Inasmuch as these manufacturing processes sometimes varied considerably, not only for different commodities but often among different manufacturers of the same product, and as an individual system of measurement was often considered a trade advantage, a great number of gage systems came into existence. It has been said that at one time there were in use in this country and in England more than fifty different wire gages and several distinct gages for sheets. This condition alone would give rise to considerable confusion, but, as if confusion were to be sought rather than avoided, different names were often applied to the same or practically the same gage system. Also, the same names or symbols were frequently employed to designate different gage systems. All these systems were not only dissimilar with respect to each other in actual thickness denoted by the gage numbers, but the different numbers in the same table seldom bore any mathematical relation to each other.

Relation of Gage Number to Thickness—These gage systems had but one characteristic in common, namely, that the higher the gage number the thinner was the material. This relation of gage numbers to actual thicknesses has always been maintained, and the association of high gage number with thin material or with small

diameter has become fixed, by long custom, in the minds of people associated with the metal industries. With but few exceptions, this relation of gage number to thickness or diameter persists to the present time. The exceptions are the Sheet Zinc Gage, Belgian Zinc Gage, Paris (French) Gage, and Music Wire Gage, in which the gage numbers increase with the thickness of the sheet or the diameter of the wire.

British Gages—Chief among the early gage systems were the Birmingham gages, one for sheets and another for wire, and the Stubs' gages. As Stubs was from Warrington, his most popular gage was often called the Warrington Wire Gage, but was also known as Peter Stubs' Gage. In this country, Peter Stubs' Gage and the Birmingham Wire Gage are considered to be identical. The first attempt at reform in gages was to list the equivalent of the gage numbers in decimals of an inch. This was done first individually by Stubs, and later by organized action of the British Board of Trade, in 1883. In that year, a gage was prepared which was intended as a standard for both wire and sheets but was later found to be unsuitable for sheets. This gage became the legal British standard gage on March 1, 1884, and is now known as the British Imperial Standard Wire Gage, designated in the British Empire by the initials W.G. or B.W.G., and in the United States by the initials I.S.W.G. or S.W.G. When this gage was found to be unsuitable for sheets, a new gage, called the Birmingham Sheet and Hoop Iron Gage, was prepared by revising the old sheet gage. The new gage has since been used in England, merely by common consent until 1914, at which time it was established legally as the British Standard Gage for Iron and Steel Sheets and Hoops, represented by the symbols B.G. Custom gages for galvanized sheets now usually have B.G. suffixed to the gage numbers, but the weights have no systematic relation to the weights of the British Standard Gage (B.G.) for uncoated sheets. The earlier English gages, Birmingham Wire Gage and Stubs' Warrington Wire Gage, are no longer in common use except for telephone and telegraph wire. The relationships between the more common gages mentioned above are given in Table 50—III.

United States Sheet Gages—The next attempt at standardizing gages was made in the United States, in 1892. On March 3rd of that year, the United States Standard Gage for Sheet and Plate Iron and Steel (Table 50—III) was established by an Act of Congress as the only standard gage for these materials after July 1, 1893. This gage is a weight gage based upon weights per square foot in pounds avoirdupois. The gage table as established by Congress began with 20 lbs. per square foot, No. 7/0's gage, and ended with 0.25 lb. per square foot, No. 38 gage, but the light side of the table has been extended by custom to 0.1875 lb. per square foot, or No. 44 gage. In this country the gage is standard for all uncoated iron and steel sheet and plate, and is also used for tin plate in the lighter gages.

Galvanized and Long Terne Sheets have individual

gages based on the U.S. Standard Gage with allowances made for the thickness of the coating in each table. Thus, for the same gage number the weight shown in the Galvanized Sheet Gage regardless of coating weight, is 0.1562 pounds per square foot heavier than the weight shown in the U.S. Standard Gage (uncoated product). In the Long Terne Sheet Gage, each gage number may have various weights depending upon the coating weight. With a commercial coating (6 pounds per double base box) each gage number is 0.016 pounds per square foot heavier than the weight shown for the corresponding U.S. Standard Gage number. (See Table 50—IV).

As stated in the preceding paragraph, the U. S. Standard Gage is based on weights per square foot in pounds avoirdupois. Table 50—III shows the approximate thickness for each gage number adopted by the originators of the gage, who based these thicknesses on the density of wrought iron, which is 480 pounds per cubic foot or

0.2778 pound per cubic inch. Since the adopted standard density for steel is about 2 per cent heavier than that of wrought iron (489.6 vs. 480 pounds per cubic foot), the thickness equivalents for steel are slightly less than those listed in the U.S. Standard Gage. Although this change is legal because the governing factors in the gage schedule are weights and not thicknesses, much confusion occurs in converting from weight to thickness for steel sheets. Consequently, the manufacturers of steel sheets in this country have adopted a new gage, known as the Manufacturers' Standard Gage for Sheet Steel (Table 50—I). The gage numbers and corresponding weights in this gage are identical to those contained in the U.S. Standard Gage, but the equivalent thicknesses are less since they are based on the density of steel, not that of wrought iron. The conversion factor used in determining these thicknesses is actually greater than the density of steel by an amount necessary to allow for the facts that sheet weights are calculated on the basis of ordered width and length with shearing tolerances on the over side, and that sheets are thicker in the center than they are at the edges where thickness is commonly and most conveniently measured.

The factor commonly used in converting from weight to thickness of steel sheets is 41.82 pounds per square foot per inch thick (see Footnote, Table 50—III).

Density of Iron and Steel—In the foregoing discussion, the density of steel was given as 489.6 pounds per cubic foot or 40.8 pounds per square foot per inch of thickness, which figure has been adopted as the standard density of steel of the grades and kinds generally used in plates. The actual density of steel varies slightly with composition and treatment, and thus may be at variance with the adopted standard density as can be seen from Table 50—II, which presents data collected from various

Table 50—I. Manufacturers' Standard Gage for Sheet Steel

Gage thickness equivalents are based on 0.0014945 in. per oz. per sq. ft.; 0.023912 in. per lb. per sq. ft. (reciprocal of 41.820 lb. per sq. ft. per in. thick); 3.443329 in. per lb. per sq. in.

Manufacturers' Standard Gage No.	Ounces per Square Foot	Pounds per Square Inch	Pounds per Square Foot	Inch Equivalent for Steel Sheet Thickness	Manufacturers' Standard Gage No.
3	160	0.069444	10.0000	0.2391	3
4	150	.065104	9.3750	.2242	4
5	140	.060764	8.7500	.2092	5
6	130	.056424	8.1250	.1943	6
7	120	.052083	7.5000	.1793	7
8	110	.047743	6.8750	.1644	8
9	100	.043403	6.2500	.1495	9
10	90	.039062	5.6250	.1345	10
11	80	.034722	5.0000	.1196	11
12	70	.030382	4.3750	.1046	12
13	60	.026042	3.7500	.0897	13
14	50	.021701	3.1250	.0747	14
15	45	.019531	2.8125	.0673	15
16	40	.017361	2.5000	.0598	16
17	36	.015625	2.2500	.0538	17
18	32	.013889	2.0000	.0478	18
19	28	.012153	1.7500	.0418	19
20	24	.010417	1.5000	.0359	20
21	22	.0095486	1.3750	.0329	21
22	20	.0086806	1.2500	.0299	22
23	18	.0078125	1.1250	.0269	23
24	16	.0069444	1.0000	.0239	24
25	14	.0060764	0.87500	.0209	25
26	12	.0052083	.75000	.0179	26
27	11	.0047743	.68750	.0164	27
28	10	.0043403	.62500	.0149	28
29	9	.0039062	.56250	.0135	29
30	8	.0034722	.50000	.0120	30
31	7	.0030382	.43750	.0105	31
32	6.5	.0028212	.40625	.0097	32
33	6	.0026042	.37500	.0090	33
34	5.5	.0023872	.34375	.0082	34
35	5	.0021701	.31250	.0075	35
36	4.5	.0019531	.28125	.0067	36
37	4.25	.0018446	.26562	.0064	37
38	4	.0017361	.25000	.0060	38

Table 50—II. Approximate Densities of Different Varieties of Iron and Steel.

Material (In Wrought Form)	Density (at 60° F.)		
	Grams per cc.	Lb. per Cu. In.	Lb. per Cu. Ft.
Pure Iron (99.9% Fe)	7.86	0.284	491
Soft Steel (0.06% C) ..	7.87	0.284	491
Carbon Steel (0.40% C)	7.84	0.283	489
Tool Steel (0.90% C) ..	7.82	0.282	487
Wrought Iron	7.40-7.90	0.267-0.285	461-493
Stainless Steel (18% Cr, 8% Ni) ...	8.03	0.29	501
Stainless Steel (17% Cr, 0.12% C) ..	7.75	0.28	484
Stainless Steel (27% Cr, 0.35% C) ..	7.47	0.27	467
High Speed Tool Steel (18% W)	8.75	0.316	546

From these values, it is evident that the weight gage thickness equivalents cannot be applied with accuracy to many of the high alloy steels.

The Tin Plate Gage—For tin plate and short terne plate, long custom has established the Tin Plate Gage, which is practically the same for this country and England. This gage is expressed in pounds per base box, rather than in gage numbers. By base box is meant 112 sheets, each 14 x 20 inches, or other combinations of number and size of sheets that will cover an area of

Table 50—III. Relationship of Gage Numbers in Common Use.

WEIGHT GAGES					THICKNESS GAGES						
Name of Gage	United States Standard Gage, U.S.S.G.		Galvanized Sheet Gage, G.S.G.		Tin Plate Gage, T.P.G.		Steel Wire Gage Washburn & Moen or W. & M. Wire G. U. S. Steel W. G. Steel W. G.	Music Wire Gage, M. W. G.	Brown & Sharpe Gage, B. & S. G. A. W. G.	Stub's Iron Wire Gage, W. W. G. B. W. G.	Name of Gage
	Uncoated Carbon Steel Sheets and Light Plates				Lb. per Sq. Ft.	Oz. per Sq. Ft.					
Principal Use	Equivalent Thickness, Inch	Lb. per Sq. Ft.						Thickness, Inch	Thickness, Inch	Thickness, Inch	Principal Use
Gage No.											Gage No.
7/0's	0.4902	20.0000						0.4900	0.004	0.5800	7/0's
6/0's	.4596	18.7500						.4615	.005	.5165	6/0's
5/0's	.4289	17.5000						.4305	.006	.4600	5/0's
4/0's	.3983	16.2500						.3938	.007	.4096	4/0's
3/0's	.3676	15.0000						.3625	.008	.3648	3/0's
2/0's	.3370	13.7500						.3310	.009	.3249	2/0's
0	.3064	12.5000						.3065	.010	.2893	0
1	.2757	11.2500						.2830	.011	.2576	1
2	.2604	10.6250						.2625	.012	.2294	2
3	.2451	10.0000						.2437	.013	.2043	3
4	.2298	9.3750						.2253	.014	.1819	4
5	.2145	8.7500						.2070*	.016	.1620	5
6	.1991	8.1250						.1920*	.018	.1443	6
7	.1838	7.5000						.1770*	.020	.1285	7
8	.1685	6.8750			7.0312	112.5		.1620*	.022	.1144	8
9	.1532	6.2500			6.4062	102.5		.1483*	.024	.1019	9
10	.1379	5.6250			5.7812	92.5		.1350*	.026	.0907	10
11	.1225	5.0000			5.1562	82.5		.1205*	.029	.0808	11
12	.1072	4.3750			4.5312	72.5		.1055*	.031	.0720	12
13	.0919	3.7500			3.9062	62.5		.0915*	.033	.0641	13
14	.0766	3.1250			3.2812	52.5		.0800*	.035	.0571	14
15	.0689	2.8125			2.9687	47.5		.0720*	.037	.0508	15
16	.0613	2.5000			2.6562	42.5		.0625*	.039	.0453	16
17	.0551	2.2500			2.4062	38.5		.0540*	.041	.0403	17
18	.0490	2.0000			2.1562	34.5		.0475*	.043	.0359	18
19	.0429	1.7500			1.9062	30.5		.0410*	.045	.0320	19
20	.0368	1.5000			1.6562	26.5		.0348	.047	.0285	20
21	.0337	1.3750			1.5312	24.5		.0317	.049	.0253	21
22	.0306	1.2500			1.4062	22.5		.0286	.051	.0226	22
23	.0276	1.1250			1.2812	20.5		.0258			23

GAGE NUMBERS

25	.0214	.8750	1.0312	16.5	.895 .8750	195	4X	.0204	.059	.0179	.020	25
26	.0184	.7500	.9062	14.5	.863 .827 .804	188 180 175	4XL DX 3X	.0181	.063	.0159	.018	26
27	.0169	.6875	.8437	13.5	.771 .750 .748 .712	168 163 155	3XL 2X	.0173	.067	.0142	.016	27
28	.0153	.6250	.7812	12.5	.6875 .680 .657 .638	148 143 139	2XL DC	.0162	.071	.0126	.014	28
29	.0138	.5625	.7187	11.5	.6250 .620 .588 .574 .565 .5625 .542	135 128 125 123	DX LXL	.0150	.075	.0113	.013	29
30	.0123	.5000	.6562	10.5	.514 .505 .500 .491 .459	118 112 110	IC	.0140	.080	.0100	.012	30
31	.0107	.4375	.5937	9.5	.435 .436 .413	107 100	ICL	.0132	.085	.0089	.010	31
32	.0100	.4062	.5625	9.0	.390 .3750 .367	95 90		.0128	.090	.0080	.009	32
33	.0092	.3750	.5312	8.5	.321 .3125 .286	85 80 75		.0118	.095	.0071	.008	33
34	.0084	.3437	.5000	8.0	.276 .2656 .253	70 65 60		.0104	.100	.0063	.007	34
35	.0077	.3125			.2031 .1953 .1875	65 60 55		.0095	.106	.0056	.005	35
36	.0069	.2812						.0090	.112	.0050	.004	36
37	.0065	.2656						.0085	.118	.0045		37
38	.0061	.2500						.0080	.124	.0040		38
39	.0057	.2344						.0075	.130	.0035		39
40	.0054	.2187						.0070	.138	.0031		40
41	.0052	.2109						.0066	.146	.0028		41
42	.0050	.2031						.0062	.154	.0025		42
43	.0048	.1953						.0060	.162	.0022		43
44	.0046	.1875						.0058	.170	.0020		44

* Three intermediate fractional gages sometimes used are omitted in this table.
Table above is based on the theoretical weight which make the weight of a plate one foot square and one inch thick 40.8 pounds. Sheets and light plates are gaged on the edges, and the spring in the rolls causes the centers to be slightly thicker than the edges. To have the estimated weights of sheets and light plates equal the actual weight, the average weight of a square foot one inch thick is taken as 41.82 pounds.

Table 50—IV. Gage Weights for Long Terne Sheets of Various Coating Weights

Gage Weights in Ounces and Pounds per Square Foot, for the Gages and Coatings Given

Long Terne Gage No.	Commercial		0.35 Ounce		0.45 Ounce		0.55 Ounce		0.75 Ounce		1.10 Ounce		1.45 Ounce	
	Oz. per Sq. Ft.	Lb. per Sq. Ft.	Oz. per Sq. Ft.	Lb. per Sq. Ft.	Oz. per Sq. Ft.	Lb. per Sq. Ft.	Oz. per Sq. Ft.	Lb. per Sq. Ft.	Oz. per Sq. Ft.	Lb. per Sq. Ft.	Oz. per Sq. Ft.	Lb. per Sq. Ft.	Oz. per Sq. Ft.	Lb. per Sq. Ft.
10	90.25	5.641												
11	80.25	5.016												
12	70.25	4.391												
13	60.25	3.766												
14	50.25	3.141												
15	45.25	2.828												
16	40.25	2.516	40.35	2.522										
17	36.25	2.266	36.35	2.272										
18	32.25	2.016	32.35	2.022	32.45	2.028								
19	28.25	1.766	28.35	1.772	28.45	1.778								
20	24.25	1.516	24.35	1.522	24.45	1.528	24.55	1.534	24.75	1.547				
21	22.25	1.391	22.35	1.397	22.45	1.403	22.55	1.409	22.75	1.422				
22	20.25	1.266	20.35	1.272	20.45	1.278	20.55	1.284	20.75	1.297	21.10	1.319	21.45	1.341
23	18.25	1.141	18.35	1.147	18.45	1.153	18.55	1.159	18.75	1.172	19.10	1.194	19.45	1.216
24	16.25	1.016	16.35	1.022	16.45	1.028	16.55	1.034	16.75	1.047	17.10	1.069	17.45	1.091
25	14.25	0.892	14.35	0.897	14.45	0.903	14.55	0.909	14.75	0.922	15.10	0.944	15.45	0.966
26	12.25	0.766	12.35	0.722	12.45	0.778	12.55	0.784	12.75	0.797	13.10	0.819	13.45	0.841
27	11.25	0.703	11.35	0.709	11.45	0.716	11.55	0.722	11.75	0.734	12.10	0.756	12.45	0.778
28	10.25	0.641	10.35	0.647	10.45	0.653	10.55	0.659	10.75	0.672	11.10	0.694	11.45	0.716
29	9.25	0.578	9.35	0.584	9.45	0.591	9.55	0.597	9.75	0.609	10.10	0.631	10.45	0.653
30	8.25	0.516	8.35	0.522	8.45	0.528	8.55	0.534	8.75	0.547	9.10	0.569	9.45	0.591

Nominal Coating Weights, pounds per double base box

12

15

20

30

40

31,360 square inches. The gages of tin plate were formerly designated by symbols and names, as IC (Common), IX (X or Extra), DC (Double Common), 2X (two-X), ICL (Light), corresponding respectively to 107 lb., 135 lb., 139 lb., 155 lb., and 100 lb. per base box. In these symbols each X represents a specific additional weight and each L a specific decrease in weight. These symbols are falling into disuse, giving way to the more logical method of designation of pounds per base box.

U. S. Wire Gages—The wire gages in general use have never been legally standardized in the United States; the practice had been for each of the steel wire manufacturers to adopt his own gage. Chief among the historical gages are the gage of the American Steel and Wire Company, now a Division of United States Steel, which adopted the Washburn and Moen gage, and the nearly identical Roebling gage. Therefore, upon recommendation of the Bureau of Standards, these manufacturers' gages were merged into one gage designated as the Steel Wire Gage (Stl. W.G.), or the United States Steel Wire Gage (U.S. Stl. W.G.), which was accepted as the standard gage for all steel wire other than music wire. This is now the most commonly used steel-wire

gage in this country. For all sheets and wire made of metals other than iron and steel, the Brown and Sharpe Gage (B. & S. G.), or American Wire Gage (A.W.G.), is recognized as the standard gage in the United States. It was prepared by Messrs. Brown and Sharpe of Providence, R. I., at the request of leading manufacturers of nonferrous wire in this country. Another gage, known as the Edison or Circular Mil gage, is used by electrical engineers to simplify their calculations. This gage is based on the circular mil which is the area of a circle with a diameter of one mil (0.001 inch). Other gages in use are the Trenton Iron Company's gages, and Stubs' Steel Wire Gage. None of the wire gages mentioned above, however, has any legal authorization in this country. The only wire gage recognized in Acts of Congress is the Birmingham Wire Gage (B.W.G.), also known as Stubs' Iron Wire Gage, although it is not used to any extent by the wire manufacturers in the United States, except for telephone and telegraph wire (See ASTM A-111-52). This gage is sometimes used in designating the thickness of hoop and other strip steel products, but the tendency in these fields is to abandon the use of gages entirely and specify all thicknesses in thousandths of an inch.

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